Ireid

File (library)

RI 8202

Bureau of Mines Report of Investigations/1976

Support Capabilities of Pneumatically Stowed Materials

UNITED STATES DEPARTMENT OF THE INTERIOR

Report of Investigations 8202

Support Capabilities of Pneumatically Stowed Materials

By Roy L. Soderberg and Donald R. Corson
Spokane Mining Research Center, Spokane, Wash.

March 3, 1849

UNITED STATES DEPARTMENT OF THE INTERIOR
Thomas S. Kleppe, Secretary

BUREAU OF MINES
Thomas V. Falkie, Director

This publication has been cataloged as follows:

Soderberg, Roy L
Support capabilities of pneumatically stowed materials / by Roy L. Soderberg and Donald R. Corson. [Washington] : Bureau of Mines, 1976.

48 p. ; 26 cm. (Report of investigations - Bureau of Mines ; 8202)

Bibliography: p. 46-48.

1. Mine filling. 2. Pneumatic machinery. I. Corson, Donald R., joint author. II. United States. Bureau of Mines. III. Title. IV. Series: United States. Bureau of Mines. Report of investigations - Bureau of Mines ; 8202.

TN23.U7 no. 8202 622.06173

U.S. Dept. of the Int. Library

CONTENTS

ILLUSTRATIONS

ILLUSTRATIONS--Continued

TABLES

SUPPORT CAPABILITIES OF PNEUMATICALLY STOWED MATERIALS

by

Roy L. Soderberg[1] and Donald R. Corson[2]

ABSTRACT

Ground support for metal, nonmetal, and coal mines was examined through fill placement with a pneumatic stower to increase extraction, reduce ground pressures, and improve general mine safety. Several pneumatically stowed stope-fill products were tested to determine the effects of specific gravity, gradation, moisture, and impact velocity on the in-place density of the stowed fill.

Hydraulic sand filling has now served the mining industry for several decades but because of the low density does not always provide ample support to the hanging wall; therefore, the sill and crown pillars sometimes build up excessive stresses that are detrimental to mining.

Cementation of a dense fill adjacent to a pillar to allow 100-pct extraction of the pillar, or a fill that can be placed without the use of water and where water is a detriment to rubber-tired mining equipment, was found to be a possible use of pneumatic fill. Placing of coal mine washery products into abandoned underground workings is another possibility in order to completely eliminate surface impoundments.

INTRODUCTION

Many underground mines have made rapid advances in the past 50 years and are expected to make even more progress in the next few years because of increased demand, increased prices, and improvements in equipment. The mining of large, low-grade deposits in and near cities and many other urban areas will require more use of backfill to minimize subsidence and increase the percentage of extraction. Mining of deep deposits will require support to resist heavy ground pressures and help to reduce the incidence and severity of rock bursts.

Various mining methods often utilize some form of backfill. Shrinkage stopes leave ore in the stope as mining advances, removing only enough broken

---

[1] Mining engineer.

[2] Supervisory mining engineer.

ore to give working room in the stope. When all the ore is finally drawn out, the walls are allowed to freely cave.

Cut-and-fill stopes with "gob fill" use development waste rock where available, or material from short raises driven into the hanging wall.

HYDRAULIC BACKFILL

Most cut-and-fill stopes now use hydraulically placed fill of either stream sand or deslimed mill tailings, which have the advantages of being easy and economical to place, making a smooth, hardworking floor, and using a waste product that would otherwise require surface disposal (10, 14).[3]

Hydraulic backfill has the disadvantage of introducing excess water into a mine, requiring additional ditch cleanup and added pumping costs. In large hydraulic placements, such as in sublevel open stopes, there is always the danger of a major failure in the fill should a bulkhead break in a several 100-foot-high stope of saturated sand. Such failure would, of course, be a major underground disaster. Another disadvantage of hydraulic fill is the poor initial density with nearly 50-pct voids. Of course, such material does not provide adequate support to the stope walls when ground pressures are high. This can contribute to high stresses in the crown and sill pillars, leading to rock bursts when the pillars fail.

In flat-bedded deposits, or undercut-and-fill stopes, it is difficult to place hydraulic fill tight against the back. The excess water is also difficult to handle in flat seams as there is no natural drainage to collection points without extra excavation. Thus, the water is always underfoot, making stope working conditions wet, slippery, and unsafe. This creates particular problems for mines using mobile equipment and having a clay or soft shale floor.

ACKNOWLEDGMENTS

Thanks are due to Graham Ball, manager of Radmark Engineering of Vancouver, British Columbia, Canada, for his assistance in all phases of this project, and to Ray Mitchell, F. Michael Jones, William McLaughlin, and David Cummins, all of Spokane Mining Research Center, for the fieldwork and technical assistance in testing, compiling data, and manuscript preparation.

PNEUMATIC STOWING--BACKGROUND

Pneumatic backfilling (stowing) has been used to considerable extent in European coal mines. Several articles by German authors (15-16) have discussed operating procedures and the relationship between stowing efficiency, length of pipe, quantity of air, and pressure. Formulas are developed for determining theoretical velocity for varying particle shapes. These reports

---

[3] Underlined numbers in parentheses refer to items in the bibliography at the end of this report.

deal with machines using high pressure air (25 to 40 psig), which is considerably higher than the 8- to 10-psig operating pressure of the unit involved in this study.

Shortly after World War II, the Federal Bureau of Mines procured a German pneumatic stower for testing in the Pennsylvania coalfields. Two reports (17-18) described the use of these machines for backfilling a room-and-pillar coal mine. These early reports provided no information regarding density achieved, velocity, power consumption, or machine wear data, and no conclusions as to the technical or economic feasibility of pneumatic stowing.

Pneumatic stower usage has steadily declined in the coal mines of Europe and England for three main reasons (9):

1. Mines using stowing were at a competitive disadvantage costwise.

2. It was cheaper to pay the surface subsidence costs of \$0.10 to \$0.95 per ton than pay the stowing costs, which could be \$2.50 per ton of coal.

3. Surface damage by subsidence is less but not eliminated by stowing. Unless the trend changes, stowing undoubtedly will not be done in the coal mines in Europe.

Important conclusions from Spokane Mining Research Center (SMRC) research are as follows:

1. Well-graded materials (with sand and fines) are very sensitive to water content to attain good density.

2. High-gravity materials attain a higher relative density than lower gravity materials.

3. Minus 3-inch plus 1/4-inch material with no fines can attain nearly as high density as well-graded minus 1-1/2-inch gravel and sand mix.

4. Coal shale was found to be very difficult to get a uniformly high density, and the few tests with oil shale retort residue were even poorer.

The pneumatic stower is envisioned to have many uses in the mining industry:

1. Filling large stopes as well as confined areas.

2. As a method of hoisting material under certain conditions.

3. As a method of muck removal behind a boring machine.

4. Placing coal mine waste into underground workings.

SECTION I--PRELIMINARY STUDIES

Sullivan Mine

An initial step toward integrating pneumatic backfilling into a metal-mining operation was a cooperative venture between Radmark Engineering Ltd. and Cominco's Sullivan mine in Kimberley, British Columbia, in July 1966. In this application a pneumatic stower was used to place float waste into large blasthole stopes in an attempt to achieve complete filling in the flatter stopes and to avoid exposure of the workmen in these stopes (19). After overcoming various problems of excessive pipe and feeder wear, more than 48,000 tons of fill were placed in a particular stope. Under Sullivan mine conditions, data from this test indicated a one-third reduction in the cost of fill placement as compared to scraping. Again, no assessment of the quality, or support capability, of pneumatically placed fill was made, but the potential for using this technique in other metal-mining applications appeared favorable.

Bunker Hill Mine

The Bureau's Spokane Mining Research Center then participated in a limited test series conducted in cooperation with Radmark Engineering Ltd. and the Bunker Hill Co. in Kellogg, Idaho (10).

In this study a Radmark[4] stower was utilized to place various materials in a simulated stope excavated in the Bunker Hill tailing pond. A total of five different materials were placed, and their in-place density was determined. Included in the testing were--

1. Unclassified Bunker Hill mill tailings,
2. Smelter slag,
3. Minus 3/4-inch jig tailings,
4. Minus 1-1/2-inch crushed rock, and
5. Minus 2-inch crushed rock.

Results indicated that the system had promise, but further testing under controlled conditions was necessary to resolve some of the difficulties experienced. Among the problems identified were--

1. The feed had to be uniform but could be varied over a wide range for different tests.

2. Ample water with good control was necessary to measure effect of moisture content.

---

[4] Reference to specific equipment or trade names does not imply endorsement by the Bureau of Mines.

3. Automatic nozzle control was considered desirable to allow the impact area to be spread over the face area being stowed.

4. A better method of density measurement was found necessary because of the large variation between individual samples taken in different areas of the pile.

Badger Hill Project

The Radmark unit was subsequently utilized in a phase of the heavy metals program during 1968-70 to evaluate the effectiveness and economics of waste-fill placed by various techniques in an underground opening. Tests were conducted in a drift driven in cemented Tertiary gravels as part of the Badger Hill project near North San Juan, Calif. (20). However, time did not permit evaluation of the placement of plus 3-inch material by dozer or front-end loader, as planned; and all studies were conducted with pneumatically placed minus 3-inch material (fig. 1).

Equipment

A description of the various components of the stowing system (as specified and described by the supplier) is provided below.

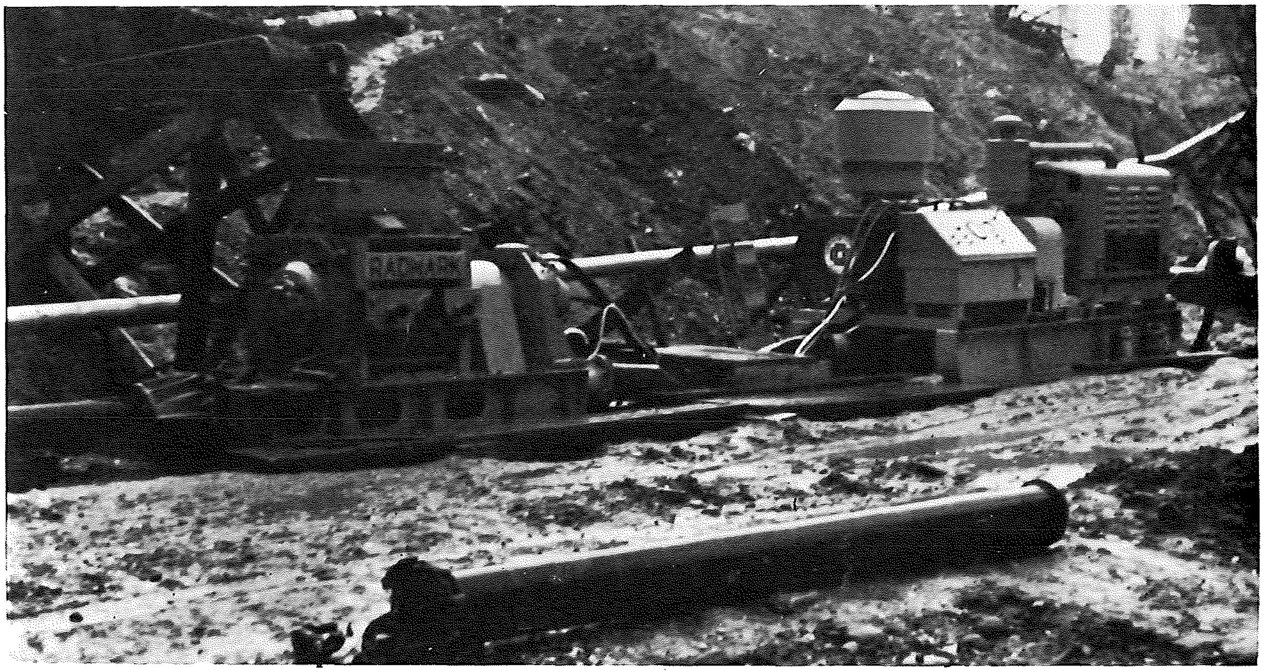


FIGURE 1. - Radmark stower, Badger Hill, Calif.

Blower Unit

The blower was a Gardner-Denver 11CDL23 unit with the following specifications:

Speed	rpm	1,600 to 2,300
Horsepower		150 to 220
Volume	cfm	3,500
Pressure	psi	<15

Stower

The stower is a rotary-valve airlock passing feed material from the low-pressure side of the airlock to the high-pressure pipeline. Constructed to combat the wear experienced in handling abrasive granular materials, it consists of a rotor and adjustable side jaws. The tips of the rotor are replaceable and are surfaced with abrasive-resistant, tungsten-titanium alloy. The adjustable side jaws compensate for wear at the tips and the jaws themselves; they are constructed with replaceable liners manufactured from Almenite W.S. material of 54 Rockwell hardness, formulated for its wear-resistant qualities. The rotor is mounted on double-sealed, self-alining roller bearings. The stower unit is mounted on a skid base that incorporates a "Tee-injector" (manufactured from Almenite W.S.), connected from the machine to the pipeline, that initially places the material in the pipeline.

Drive Unit

The hydraulic motor for the drive unit is a low-speed, high-torque type of Ruston manufacture. A 20-hp, 1,800-rpm electric motor is direct-connected to a variable-volume hydraulic pump (which drives the Ruston motor), model GW 320 Delavan, with relief valves, check valves, filters, accumulator, pressure gages, and 40-gal reservoir. Hydraulic drive for the stower allows for shock loading and speed control, so that the throughput can be regulated as required and is preset at startup.

Ancillary Equipment

Abrasive-resistant pipe of two layers, one inner layer of hard metal (600 to 700 Bhn) and an outer layer of mild steel, was used in 8 inches diameter by 10 feet long. Heavy-duty pipe has a 10-mm wall thickness.

Couplings were standard flange, eight bolts per coupling, 3/4 inch by 3 inches long (the quick-release type is also available that can be connected or disconnected with a hammer). Pipe bends were designed and manufactured with an abrasive-resistant impact section. The bend is coupled with reusable-type couplings.

A deflector is attached to the discharge end of the pipeline to permit the operator to direct the stream of material across the face of the opening. The deflector housing, fabricated from mild steel, is manufactured to allow

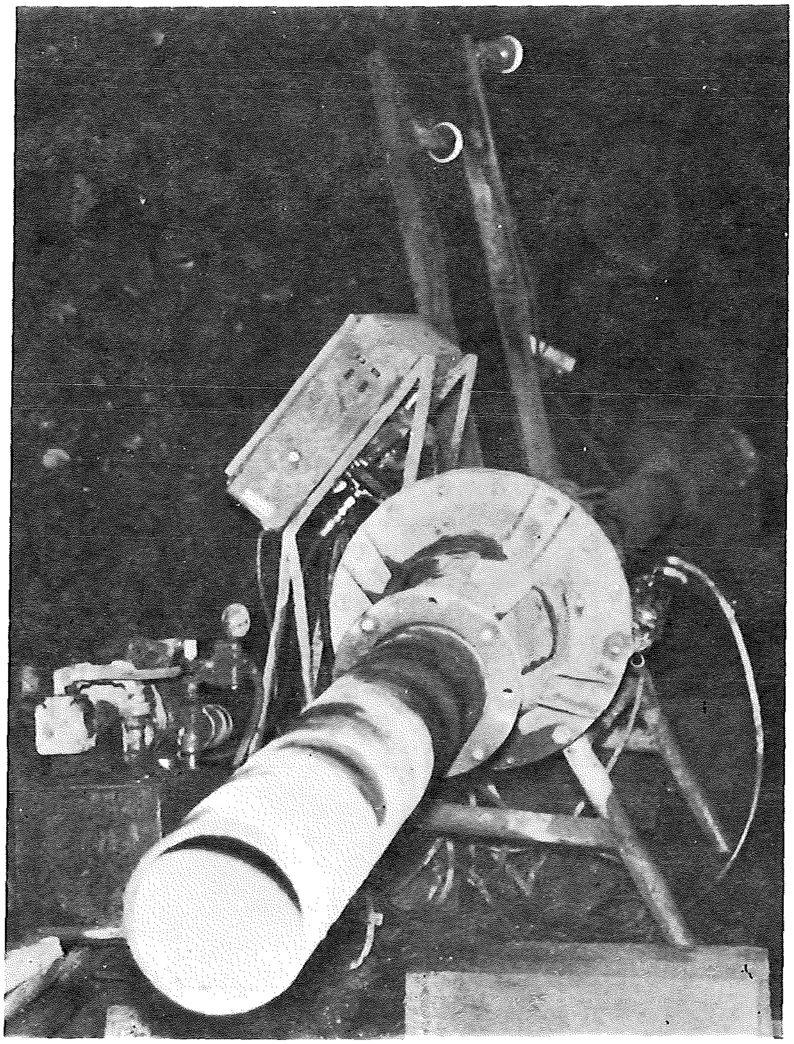


FIGURE 2. - Deflector with hydraulic pump and control panel.

for an abrasive-resistant liner. The deflector is hydraulically operated and can be automated for vertical and horizontal movement (fig. 2).

Control Panel

The control panel is mounted on the pump unit and incorporates the following:

1. A forward/reverse lever,
2. Hydraulic pressure gage (zero to 5,000 psi),
3. Air line pressure gage (zero to 15 psi),
4. Ammeter (zero to 500 amp), and
5. Two mercoid air-pressure switches (zero to 20 psi).

This control panel enables the operator at all times to select the direction of rotation of the stower (it should be alternated) and know the hydraulic operating pressure, the air line conveying pressure, and the power being consumed by the blower. The pressure switches are safety features. An air-pressure switch controls the infeed to the stower and stops the infeed if the air line conveying pressure exceeds its design. An air-pressure switch controls the whole system and stops it if the air line pressure reaches the maximum design pressure. The hydraulic pressure gage controls the infeed to the system and also the power unit. It stops both if the feeder becomes jammed with material larger than it can accept.

Stowing Materials

The materials used in the Badger Hill stowing tests were a washed gravel (minus 2-1/2-inch plus 1/4-inch) and a washed sand (fig. 3). These closely simulate the products that would be readily available in a mining and washing operation in which only the clay material would go to outside tailings dams, and all clean material would be needed underground for fill. This material

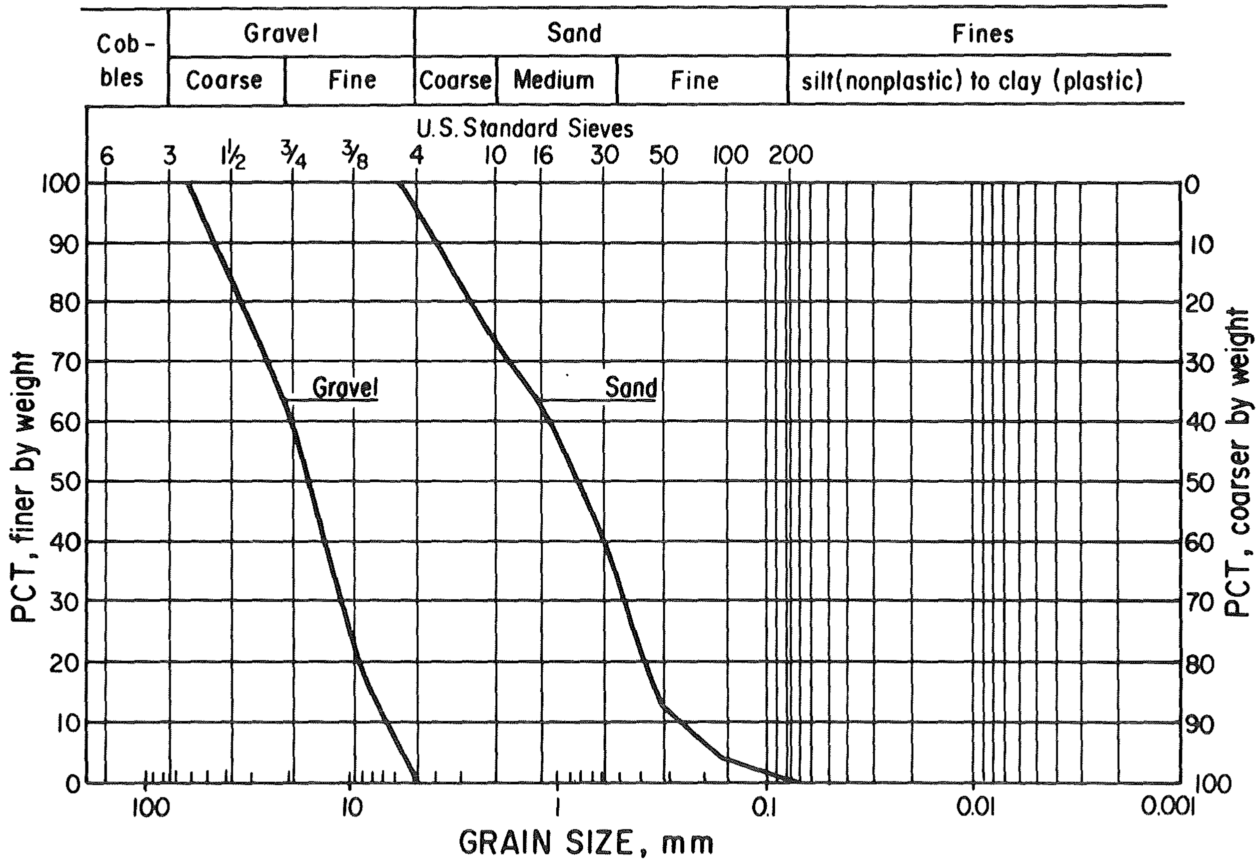


FIGURE 3. - Screen analysis, Badger Hill gravel and sand.

was mixed as well as possible with the front-end loader; the gravel-to-sand ratios were 2:1 and 1:1. The mix was then placed in a small bin, having a hand-operated arc gate, which fed directly onto a conveyor belt. Because of the free-flowing qualities of the material, it was quite easy to maintain an even flow to the stower unit, and with a little practice, would be fed at the desired rate.

Operation

Before controlled experiments began, the machine was tested at various feed rates to provide operating experience and to ascertain how much material could be effectively handled before the material "slugged"; that is, came out the discharge in spurts. At a feed rate of 200 to 240 tph, the material did "slug," the pressure went up to 8.5 psi, the revolutions per minute decreased to 1,550, and some of the material actually dropped out the end of the discharge pipe. At this point, the line could be expected to plug at any time; luckily, however, it did not. This test indicated the maximum feed rate of the machine. Employed for this test were 100 feet of pipe, a 30° elbow, 50 feet of pipe, a 60° elbow, and 20 feet of pipe into the deflector.

The operating pressure was 5.0 to 6.0 psi at a feed rate of 100 tph, with a blower speed of 2,100 rpm. An overload, or a block in the line, increased the air pressure, which at 8 psi (or other preset value) activated a mercoid switch that shut off the conveyor belt. The relationship between feed rate, blower speed, and operation pressure for Badger Hill is shown in table 1.

TABLE 1. - Feed rate, blower speed, and operating pressure--sand-gravel mix, Badger Hill, Calif.

Feed, tph	Operating pressure, psi	Blower speed, rpm
25	3.5	2,100
100	5.0	2,100
125	6.0	2,100
200	8.0	1,750
220	8.0-8.5	1,700
240	8.5	1,550

This machine was designed for 100 tph. Initial tests showed that it could manage this volume and still maintain the blower speed at maximum, at which point the volume of air could be maintained to secure good density.

The tests were conducted in a crosscut off the main adit, and the material transported through 170 feet of pipe with two bends, as described previously. The throw from the deflector ranged from 35 feet down to 8 feet as the opening was filled.

Results and Discussion

Of paramount importance for the test series was the density of the placed material, since the relative density is directly proportional to the support capability. Parameters set up for the tests were feed rate, length of pipe, bends in discharge line, air pressure, and distance from deflector to impact area. The Washington Dens-O-Meter was used to measure the effects of these variables on the in-place density of the stowed material.

It was found that using 20 feet of pipe between 60° elbow and the deflector did not provide sufficient time for the material to attain maximum velocity after being slowed by the bend. The compaction was at least 10 pcf lower than that attained when this 60° elbow was eliminated.

It was also found very difficult to secure the exact mix of gravel and sand desired for testing, but a compensating wide variety of mixes was obtained. To reduce dust, water could be added at either the stower or underground through a water ring in the pipeline immediately before the deflector. The latter was preferable, to keep the transport load as low as possible.

If the stream of material were aimed at one place for a long time, the fines and some coarse material would build up in the impact area at a high density. Consequently, much of the coarse material would rebound and roll to the side or down slope, making a very loose fill. The best densities were attained when the area being filled was confined, or when a full sweep in both directions could be made with the deflector. The procedure would allow the rebound to be packed in with new material, giving a higher average density. This could be accomplished with a little practice by the operator.

With clean sand and gravel, a 45° slope could be maintained by impacting directly against the fill. A slight amount of clay would allow a much steeper slope immediately, but if any water came from the back, the entire mass would become unstable and, therefore, unusable. A free-draining material was mandatory.

Horizontal bulkheads, about 6 feet high, were placed across the crosscut to simulate the back of a fill area, so that there would be room after removing the bulkhead to make a density measurement at the back. Density measurements of the fill directly under the bulkhead (as mentioned above) showed that the material was packed at a relatively high density while densities in the rebound area were as much as 20 pcf lower. When the full height of the 13-foot crosscut was filled, this condition seemed to be true also, even though the back was quite irregular. By being confined in this manner, the coarse material did not rebound as much as on an open face.

Maximum- and minimum-density tests were conducted on various combinations of gravel and clean sand to compare compaction achieved with the minimum-maximum line. Composition of the various mixes and their minimum and maximum dry densities from laboratory tests are shown below on table 2. The results of the stowing tests are shown on table 3. These densities are also thought to be questionable because of the wide spread in the density in the impact area as compared to the rebound area.

The number of tests was limited by time, room, and quantity of material. Nevertheless, they did indicate that the pneumatic stower could be an effective method of providing ground support. After completion of the stowing tests in the Tertiary gravel adit near Grass Valley, Calif., the equipment was moved to SMRC's auxiliary laboratory located approximately 28 miles northwest of Spokane.

TABLE 2. - Relative density, sand-gravel mix, Badger Hill, Calif.

Pct gravel (minus 2-1/2 plus 1/4 inch)	Pct sand (minus 1/4 inch)	Minimum density, pct	Maximum density, pcf	Density from table 3, pcf
100.00	-	105	115	-
-	100.0	103	123	-
33.3	66.7	96	127	123
50.0	50.0	119	137	95-124
66.7	33.3	118	139	101-139
75.0	25.0	115	132	99-115

TABLE 3. - Radmark stowing tests, sand-gravel mix, Badger Hill, Calif.

Sample	Field dry density, pcf	Pct moisture	Pct minus 2-1/2-inch plus 1/4-inch gravel	Pct minus 1/4-inch sand	Blower speed, rpm	Operating pressure, psi	Feed rate, tph	Impact distance, feet	Transport distance, feet	Elbows	
										30°	60°
22B	105.1	7.8	84.91	15.08	1,500-2,100	5-6	50-100	32	197	1	1
23A	111.9	8.7	66.30	33.70	1,500-2,100	5-6	50-100	31	197	1	1
23B	122.9	2.1	56.26	43.74	1,500-2,100	5-6	50-100	28	197	1	1
25A	95.3	4.7	53.16	46.84	2,100	6.0	50-100	25	197	1	1
5A	111.6	10.1	80.20	19.80	2,100	5.0	50-100	32.5	197	1	1
5B	115.0	10.2	74.36	25.64	2,100	5.0	50-100	31.5	197	1	1
6A	110.9	6.7	49.61	50.39	1,750	5.5	50-100	30	197	1	1
6B	105.8	5.6	60.18	39.82	1,750	5.5	50-100	30	197	1	1
7B	139.3	5.3	68.54	31.46	2,100	5.5-6.0	73	14	197	1	1
8A	123.6	6.6	52.63	47.37	2,100	5.5-6.0	73	14	197	1	1
8B	122.5	4.8	66.87	33.13	2,075	5.0	69	12	197	1	1
9A	118.1	5.5	50.00	50.00	2,075	5.0	69	12	197	1	1
9B	112.3	3.4	87.30	12.70	2,150	3.5-4.0	23	14	197	1	1
10A	113.7	.7	85.26	14.74	2,150	3.5-4.0	23	14	197	1	1
10B	101.4	5.7	61.52	38.48	2,000	6.0	78	11	197	1	1
11A	122.9	7.7	35.97	64.03	2,000	6.0	78	11	197	1	1
11B	104.9	6.6	79.74	20.26	1,900	6.0	100	8	197	1	1
12A	109.2	7.8	76.56	23.44	1,900	6.0	100	9	197	1	1
12B	106.1	5.3	79.80	20.20	1,900	6.0	100	12	197	1	1
13A	121.2	5.7	78.40	21.60	1,900	6.0	100	12	197	1	1
13B	136.9	8.7	69.50	30.50	2,100	5.0	100	46	167	1	-
14A	132.9	7.8	81.72	18.28	2,100	5.5	117	45	167	1	-
14B	135.8	8.2	85.89	14.11	2,100	5.5	91	45	167	1	-
15A	98.8	8.7	77.77	22.23	2,100	6.0	125	42	167	1	-
15B	99.0	9.4	74.63	25.37	2,100	6.0	125	40	167	1	-
16A	114.2	7.5	73.79	26.21	2,100	5.5	108	40	167	1	-
16B	134.6	10.1	62.76	37.24	2,100	5.5	108	38	167	1	-
17A	122.1	10.0	60.85	39.15	2,100	5.5	88	38	167	1	-

SECTION II--PRINCIPAL STUDY--SMRC AUXILIARY LAB

Test Facility

To quantify the effect of various significant parameters in achieving a high-quality (dense) fill, a complete test facility was constructed that incorporated modifications in several components of the stowing system.

The diesel engine was too small for the scheduled testing, and a given blower speed could not be set and maintained precisely. The engine was replaced by a 300-hp electric motor with an eddy current coupler and a mechanical speed increaser directly in line to the blower. The blower could then be maintained at any desired speed between 1,600 and 2,200 rpm, which was the range desired for stowing.

A 40-cu-yd bin was constructed and set up against a basalt cliff so that it could be directly filled by a dump truck. An air-operated arc gate discharged material directly onto a Syntron feeder mounted over a conveyor leading to the rotary vane feeder.

The material was then pneumatically transported 156 feet through an 8-inch-ID pipe with a deflector at the end so the feed could be spread over the entire face of the car. The deflector, as originally designed, had a hydraulic cylinder on the underside of the movable pipe for vertical movement and one cylinder on the side for the horizontal. Control could be preset to limit both horizontal and vertical movement automatically, but it was difficult to keep material from being thrown outside the box because of drift to one side or the other. Vertical movement of the pipe was not smooth, as the deflector moved up very slowly due to the weight of the pipe handing out in front of the cylinder. Downward movement was very rapid (for the same reason) and was corrected by adding another hydraulic cylinder on top and piping the two cylinders together so that oil was transferred from one into the other, yielding a smooth operation. Later, a manually controlled valve was added in place of automatic controls. This gave the necessary positive control for placing the material since the horizontal and vertical movements required constant changing as the car was filled.

To provide a test chamber that simulated the mine opening, a stowing box, 8 by 8 by 11.77 feet, was constructed (figs. 4-5). The frame of the stowing box was made of steel with 2-inch timber planking bolted to the frame. Inside the planking, 3/4-inch-thick marine plywood was placed to make an airtight box. The back of the box opposite the open end was built so that air under pressure could be forced through the stowed material. Permeability of stowed coal was thereby also measured (as part of a companion project). During testing, the plywood was destroyed during the first 30 sec of operation by the impact of the minus 1-1/2-inch sand-gravel mix. Four inches of shotcrete with light wire mesh reinforcing was then placed on the plywood at the back of the box and survived most of the test program.

The stowing box was used to measure volumes of stowed material for calculation of bulk density. To facilitate these measurements, horizontal saw cuts

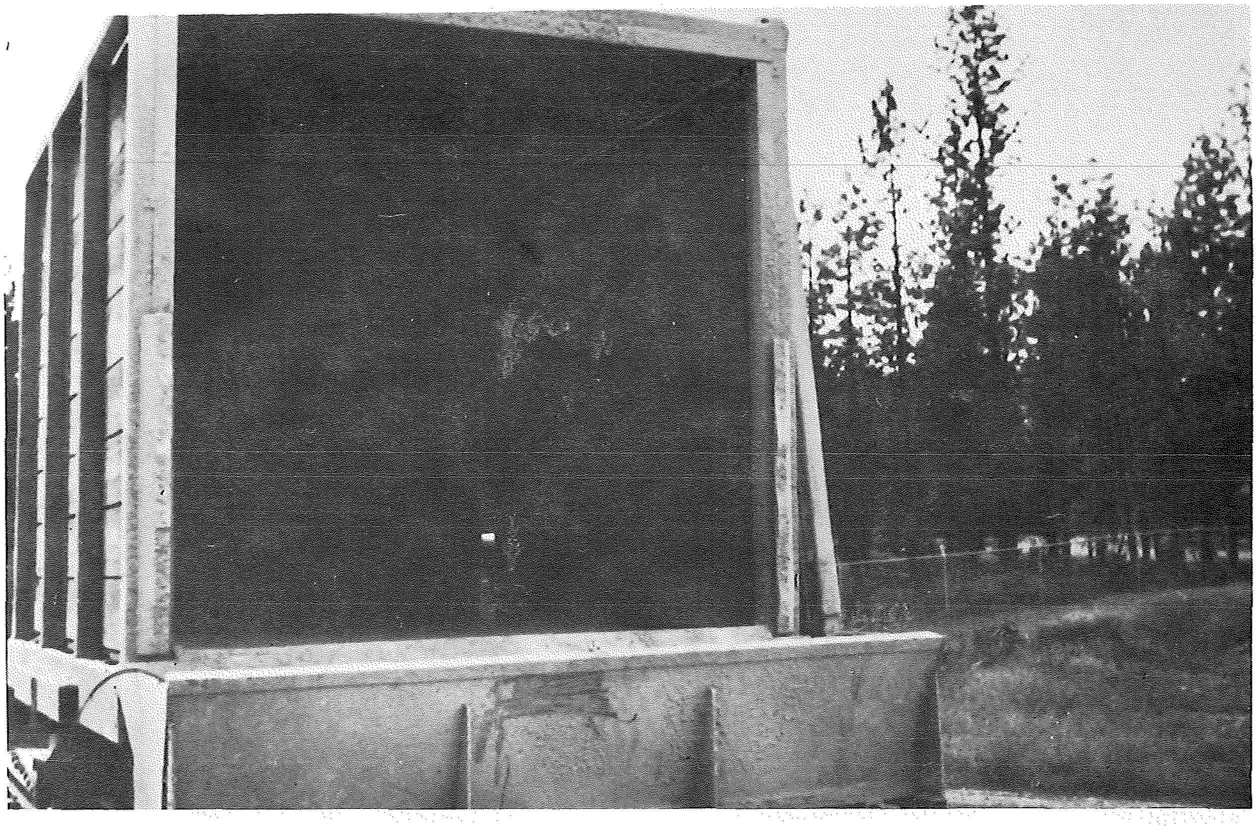


FIGURE 4. - Stowing box, side dump door unlatched and spill box in front.

were placed each 2 feet in height along both sides of the box. Measurements were made from the front of the box horizontally to the material along both sides at enough places to get an average length to calculate the volume.

The test box was equipped with wheels and mounted on rails so that the throw distance could be varied. It could also be moved onto the weighing platform and in front of the waste pit for dumping. One side of the box was hinged, and dumping was accomplished by a hydraulic cylinder. The material was picked up from the waste pit by the front-end loader and loaded into the dump truck for transfer back into the bin for reuse or to a waste dump.

The weighing platform was supported by four 50,000-pound load cells, one under each corner, with digital readout. The weighing mechanism was checked against a 43,420-pound load and found to read 0.6 pct high. A 200-pound man could be weighed with an accuracy of 1 to 2 pounds. The measured volume and bulk weight gave an accurate determination of average density as compared to the average of a large number of spot tests of approximately 0.1 cu ft each. Experience gained during the Badger Hill test program showed a variation of 20 pcf and more in different parts of the pile, which made it impossible to get a reliable bulk density with this method.

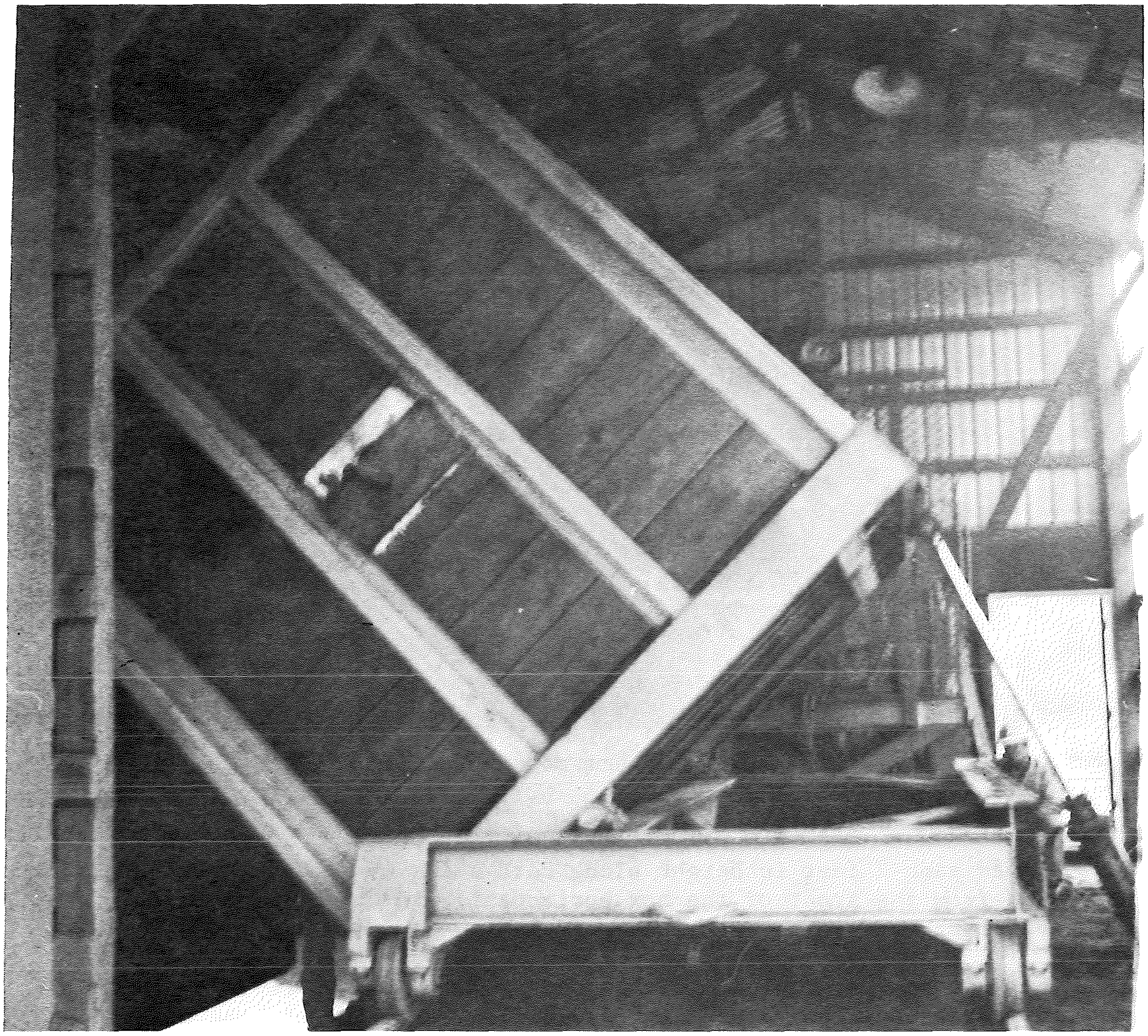


FIGURE 5. - Stowing box in dump position.

Stowing Materials

Tests were run at the auxiliary laboratory on five different products readily available to industry for use as fill material:

1. A mix of one part each of minus 1-1/2-inch plus 3/4-inch ϕ gravel, minus 3/4-inch plus 1/4-inch ϕ gravel, coarse sand, and fine sand, which was called 1:1:1:1 mix.

2. Minus 3/4-inch crushed quartzite.

3. Minus 1-1/2-inch crushed quartzite.

4. Minus 3-inch crushed limestone.

5. Minus 3-inch coal shale (from the Black Diamond mine, Palmer Coking Co., Black Diamond, Wash., located about 150 miles southeast of Seattle).

The gradation curves are shown (fig. 6-8).

It was found nearly impossible to have a uniform product for an entire test series due to the physical breakdown caused by the impact. The coal shale from Black Diamond, Wash., seemed to be especially soft and friable. This would not necessarily be true for another shale from a different area, as there seems to be a great deal of difference in the hardness of coal shale from different localities. This particular shale deteriorated very rapidly when exposed to the weather. Fresh washery waste was used for most of the tests and was not suitable for subsequent testing. A coal shale product with an original gradation of minus 3 inches plus 1/2 inch would contain 50 pct minus 1/4-inch material after one test.

It was found very difficult for coal shale to flow freely from the bin because of the sticky nature of the material. The optimum moisture content for best compaction was 18 pct, and when dumped from the car, the coal shale

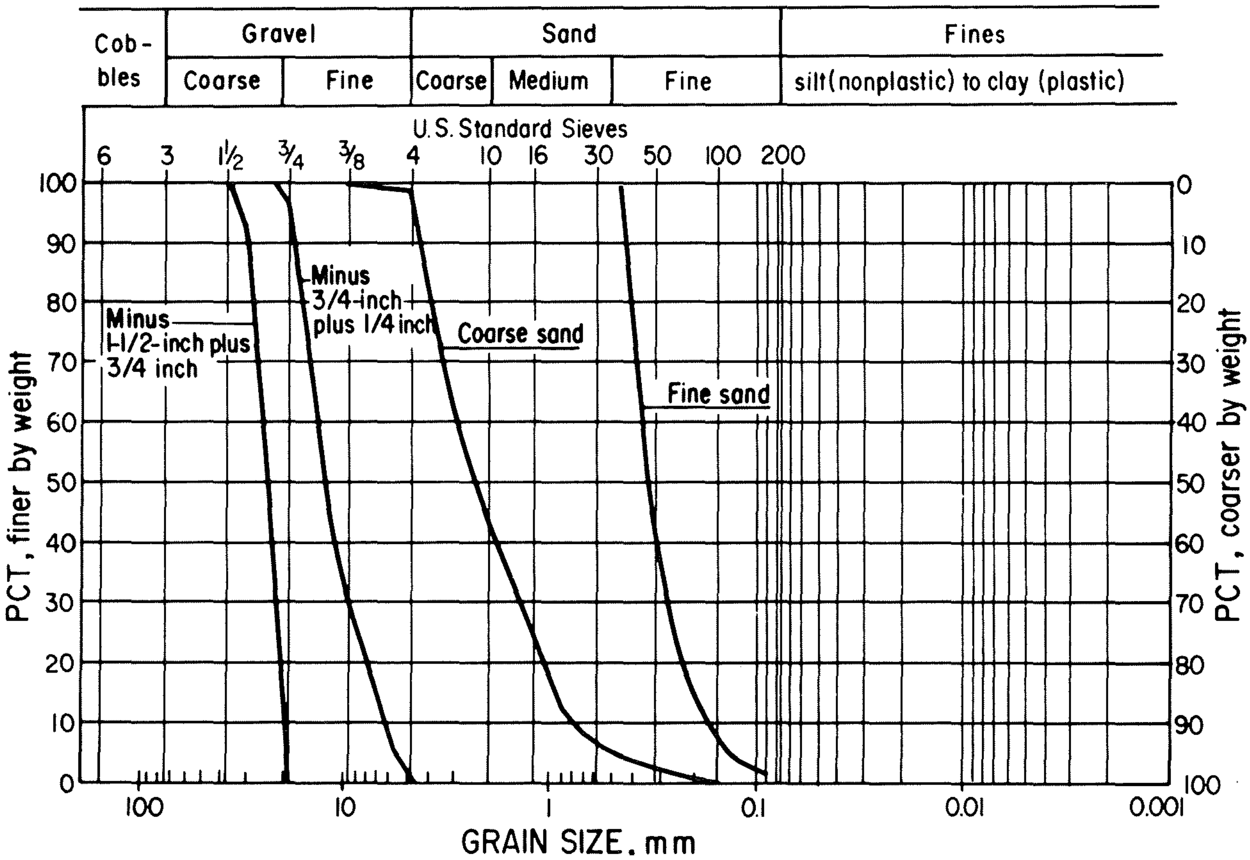


FIGURE 6. - Gradation curves, two gravels and two sands (1:1:1:1 mix).

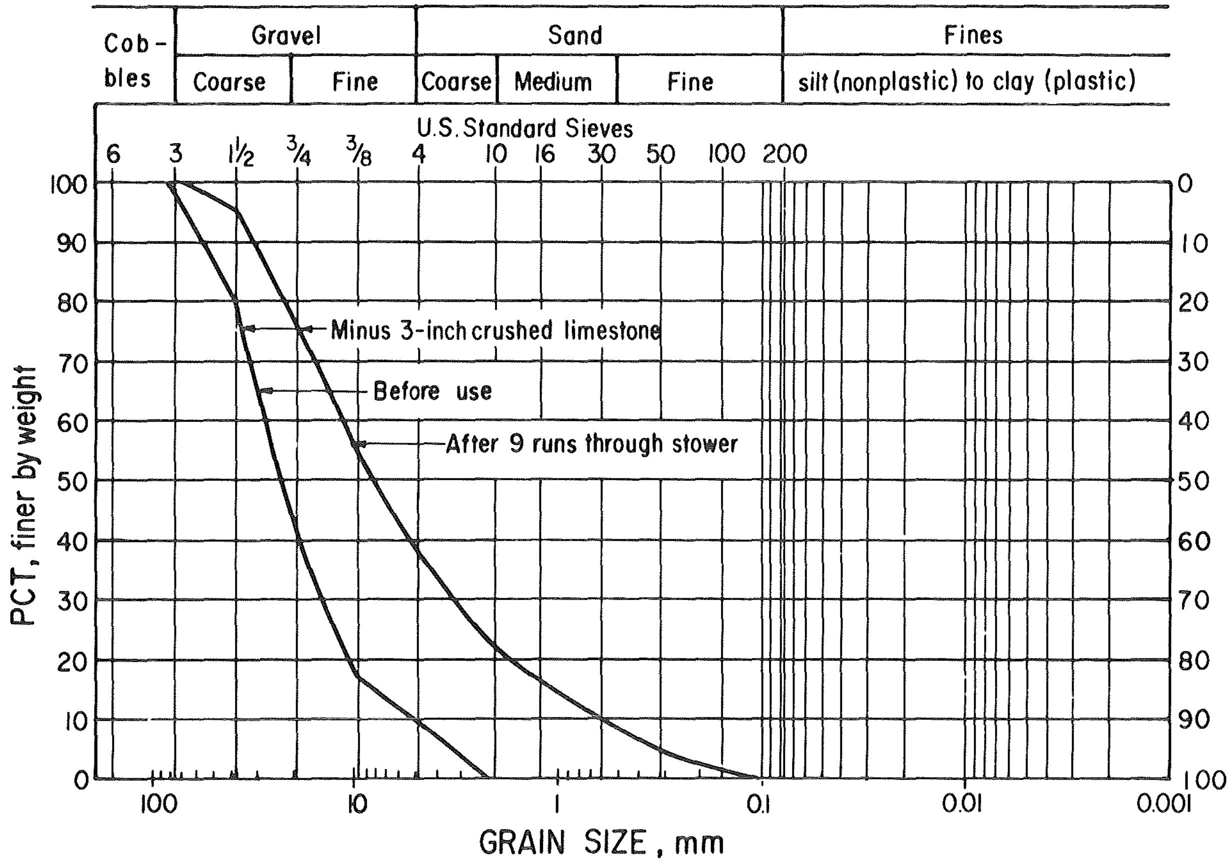


FIGURE 7. - Gradation, minus 3-inch limestone, before and after use.

broke up into large chunks because of the cohesive nature of the material. After it was picked up with the loader, transferred into the truck and dumped into the bin, the moisture content was still about 12 pct, even in the hot part of the year. Many of the lumps were not completely broken up during the move and some compaction took place because of the 20-foot drop into the bin. These lumps caused a very erratic flow onto the Syntron and required an operator with a blow pipe to keep it feeding at all. In an actual operation where coal shale is to be placed in a mine or from a mine to the surface, the material should be presaturated and allowed to drain enough so that it is not sticky on the outside. This would make it easier to feed into the stower, and sufficient water could be added in the pipeline to keep the dust down to an acceptable level. Very dry coal shale, especially the soft material used in test, is impractical to stow because the breakup of the material during transport and on impact created an intolerable amount of dust. Even the further addition of water in the material and in the line to this thoroughly dry material will not eliminate the dust because of the large volume of dry fines and dry surface exposed when the particles break up. Some of the very hard shales that actually approach slate in hardness would not be as dusty as this softer material.

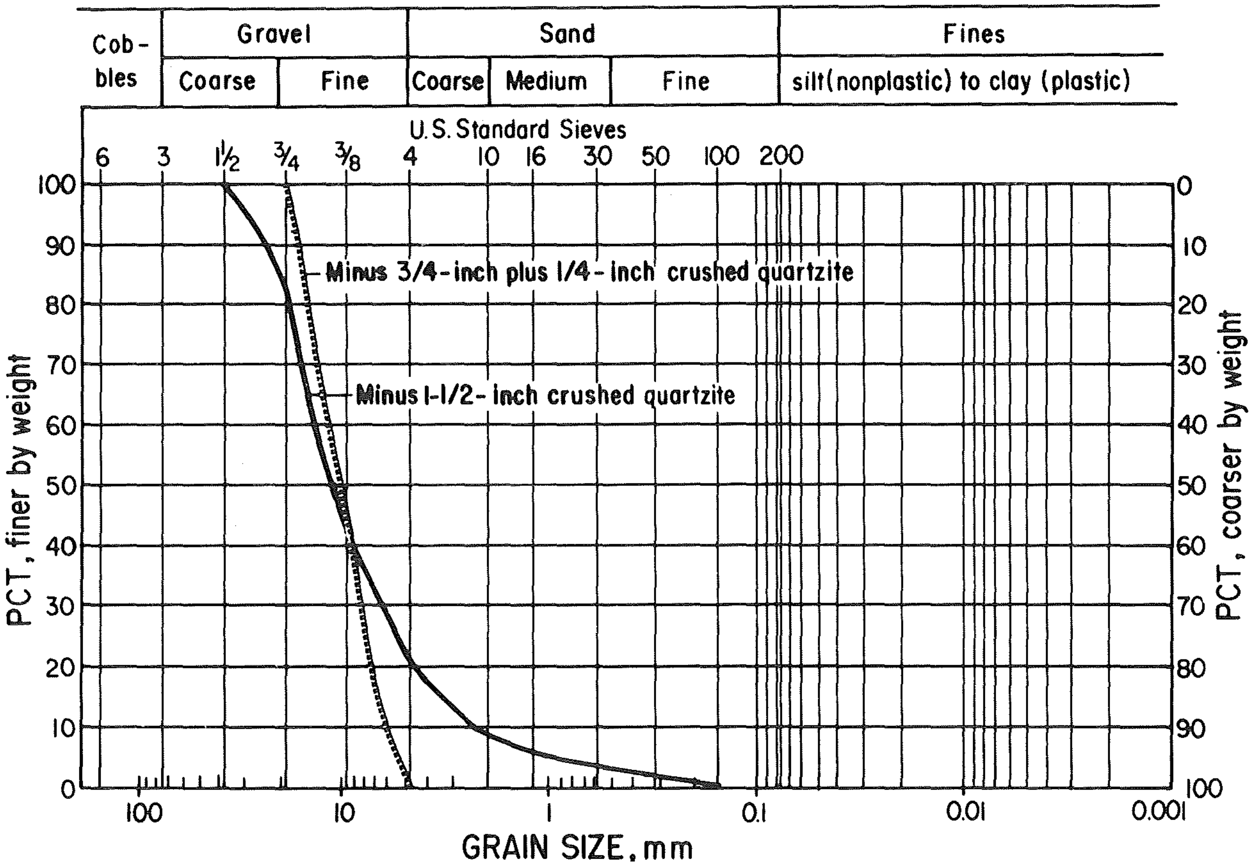


FIGURE 8. - Gradation, crushed quartzite.

Materials 1-4 were easy to handle and could be reused a number of times. The minus 3-inch crushed limestone after only a few tests had a large amount of fines (minus 1/4-inch size) where initially there was none. The larger and more brittle or friable the material, the greater the breakdown in size. Taking note of the difference in gradation, these materials could be kept wet enough to eliminate dust and still be free flowing from the bin. The feed rate could be set by measuring the material on a given length of belt and maintaining the same setting of the Syntron day after day. Additional water could be added in the pipe as needed to bring the moisture up to optimum.

Test Variables

After choosing the mix of the material, the only variables in the experiment were moisture, velocity, and throw distance. Of the three, moisture was the most critical.

Exit Velocity

Impact velocity of any given material depends on its mass, distance traveled in the line, throw distance, and air volume and velocity. These tests were run at 1,600, 1,800, 2,000, and 2,200 rpm of the blower. From the blower performance curves and the back pressure, the air velocity could be calculated to determine the exit velocity of the material.

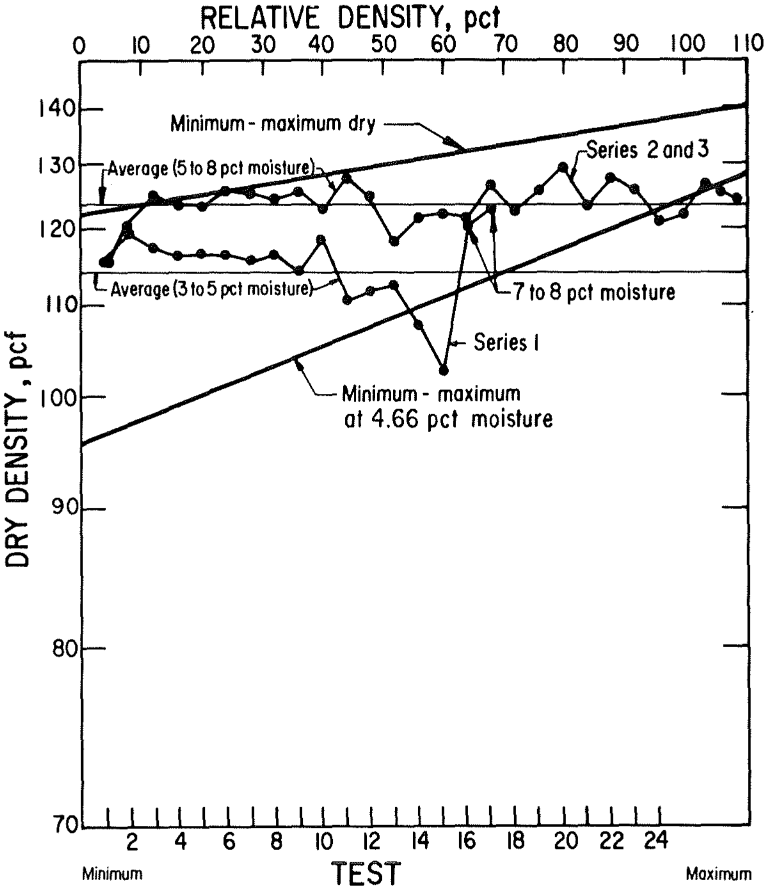


FIGURE 9. - Test series 1 at 3 to 5 pct moisture and series 2 and 3 at 5 to 8 pct moisture (gradation figure 6, 1:1:1:1 mix).

Moisture Content

The moisture content was initially kept low, with just enough water added to keep the dust down to an acceptable level. The densities attained in stowing were below minimum, as shown on the minimum-maximum testing of the sand-gravel mix (fig. 9). After several tests to verify this low density, it was determined that the sand was in the bulking moisture range, and water content had to be increased. To prove this, another minimum-maximum test was made with this same mix but with a moisture content the same as the initial stowing tests and it was found to be much lower than the dry test (fig. 9). This showed why the initial stowing tests were below the dry minimum, which in theory is impossible. It must be understood that minimum-maximum tests on materials must be either dry or saturated while every moisture content between would give a different line and really mean nothing. It was done in this case only to prove that a well-graded mix with fines with moisture to prevent dust will have a lower density than the true minimum for this material. Therefore, the moisture was increased from 3-5 pct to 5-8 pct, and the bulk density increased by about 8 pcf. Additional moisture above 8 pct for this mix was excessive and ran out of the car and out the end of the nozzle when stowing. Moisture content was fixed at 5 to 8 pct for the remainder of the testing.

Throw Distance

The travel distance of the particles was systematically varied such that tests were made at each distance at the various blower speeds and for each material. The distance recorded was the distance from the end of the deflector to the front of the car, so at the start of the stowing, the actual throw distance was 11.77 feet further. At 1,800 rpm, no tests were made at a distance of 35 feet because the material would scarcely reach that far, and at 2,200 rpm, no tests were made at 20 feet because of the danger from the great amount of rebound, especially the round stream gravel. The testing at 1,600 rpm was discontinued after two runs because results were so inferior due to the very low particle velocity.

Procedure

Air Volume and Velocity

Using the modified pneumatic stower with a new power train consisting of 300-hp motor, eddy current coupler, and speed increaser (fig. 10), the output of the cyclo-blower was checked to compare its output with the curve supplied by the manufacturer. A pipeline was installed with an orifice plate so that

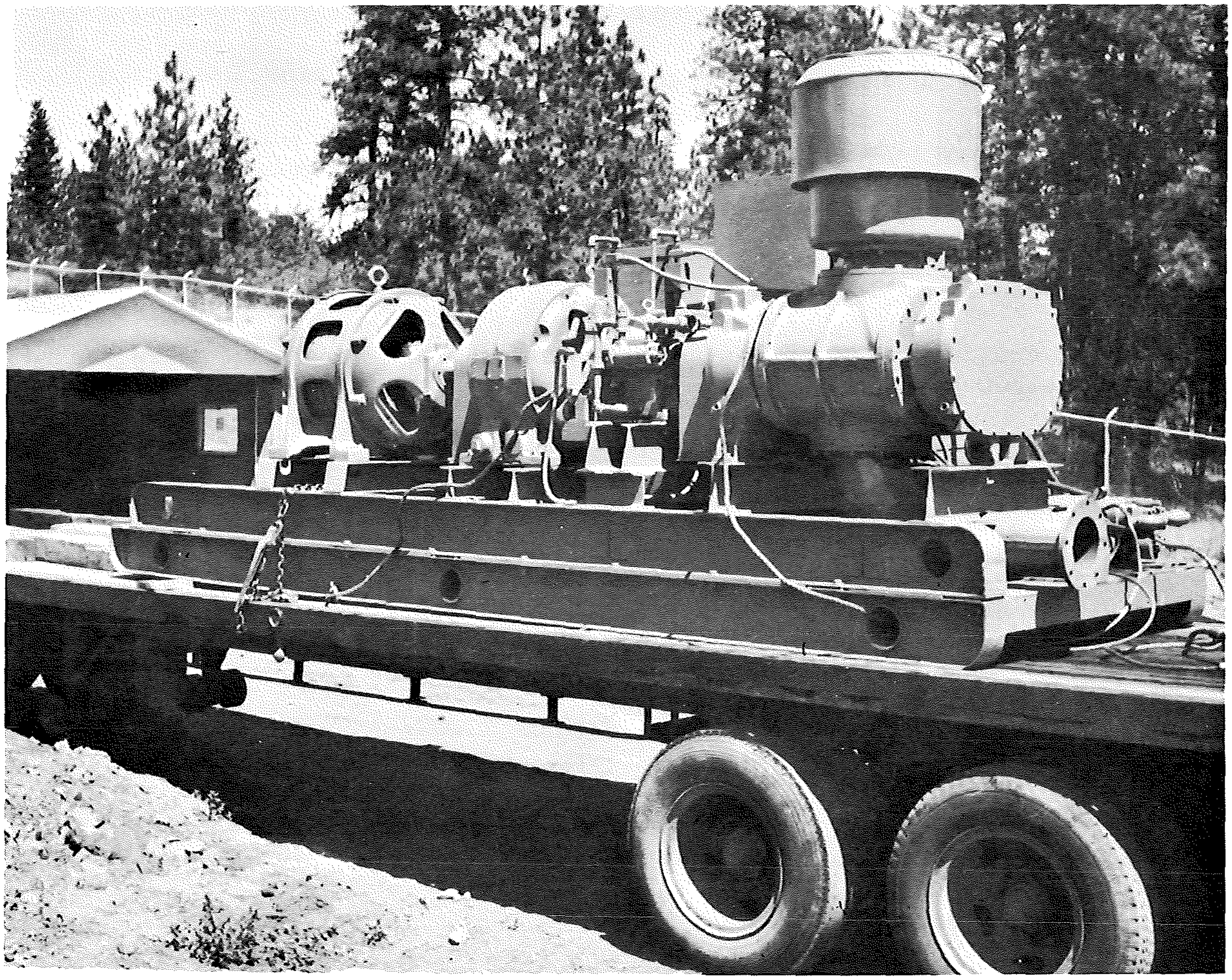
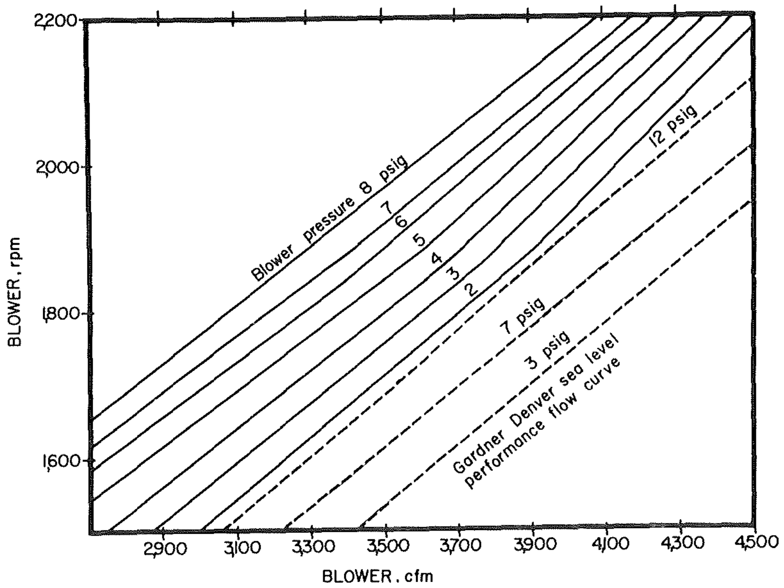


FIGURE 10. - Three-hundred-hp motor, eddy current coupler, and speed increaser to blower.



Blower output-cfm at inlet temperature and pressure

FIGURE 11. - Volume-speed curves–GD-11CDL23 blower.

the blower could be checked at 1,600, 1,800, 2,000, and 2,200 rpm, with back pressures of 2 to 8 pounds. A manometer was used to measure the pressure drop through the orifice. Curves were then drawn for the blower in the pressure ranges to be used as shown in figure 11. This showed a volume curve considerably lower than the original which was for sea level (elevation at SMRC auxiliary lab was approximately 2,365 feet).

Equipment Operation

The operator's console was arranged to make it easy for one man to take care of the feed end of the operation. The various components could be turned on and off only in sequence. Instructions are as follows:

1. The cooling water for the eddy current coupler (ecc) has to be turned on before the 300-hp electric motor will start.

2. Start 300-hp motor; bring up to speed.

3. Set blower speed by regulating ecc.

4. Turn on five Rustrak recorders connected to Bourns' pressure transducers located at intervals along the 156 feet of pipe.

5. Start hydraulic pump to feeder motor.

6. Set feeder rotation forward or reverse. Change rotation daily for even wear.

7. Open air-operated arc gate on bin. This can be done any time as it simply spills onto the pan of the Syntron.

8. Start conveyor.

9. Start Syntron.

A pressure switch shuts off the conveyor and Syntron if the line pressure exceeds a set figure (11 pounds), indicating a plugged line or the possibility of one. Another pressure switch shuts down the whole system if pressure reaches maximum design.

Initially, deflector movement is nearly all horizontal as filling begins in the bottom corner and gradually gets higher and higher. The face assumes about 45° to 50° angle during the entire stowing operation. Some of the coarse material rebounds and lands on the floor of the car in a loose state and is periodically pushed back into the pile by a sweep of the deflector. As the car gets full, the coarse material rebounds out of the car and is caught in a trough hanging across the front of the car. This can be dumped by an electric hoist into the dump area and reduces cleanup effort.

It was notable that a smooth face and high density could be attained only after the operator gained a bit of experience. After the car is measured, the area is cleared of the scatter (especially on the weighing platform), the trough in front of the car is emptied into the dump area, and the car is moved onto the weighing platform by an air tugger. The electronic readout for the load cells is generally turned on for warmup about a half hour before needed. Each of the four load cells are read and recorded one at a time, then read again. A multiplication factor of 2 gives the total weight. The car is dumped with the hydraulic cylinder, scraped clean, and returned to the scale for a tare weight. Moisture samples are taken from the dumped material, and the material is returned to the bin with the Hough front-end loader and a dump truck. The Rustrak tapes are marked with the date, material, revolutions per minute, and test number. The density calculations are done after the moisture content is known. Under ideal conditions, two tests could be run per day with any material except the coal shale.

The two men conducting the stowing tests wore earphones and microphones so they could be in direct contact at all times. This was necessary at startup and shutdown times and for water control and any emergency stop or start. (It also protected the operators from the noise, which at the motor end was over 100 db.)

Data and Analysis

Density

Some explanation of testing is needed in order to understand the results obtained. Table 3 shows the wide variety of densities of the Badger Hill tests, with the first 20 divided roughly into 2 different mixes of sand and gravel as shown graphically in figures 12-13. Note also the higher density of samples 13B-17A after the 60° elbow was eliminated near the discharge nozzle. The wide scatter of density measurements only indicates the difficulty of trying to arrive at a true bulk density by small samples in a bulk material that has such a wide density range. This forced a change in measuring density, which was done at the test facility at the auxiliary laboratory when the entire car was weighed and measured.

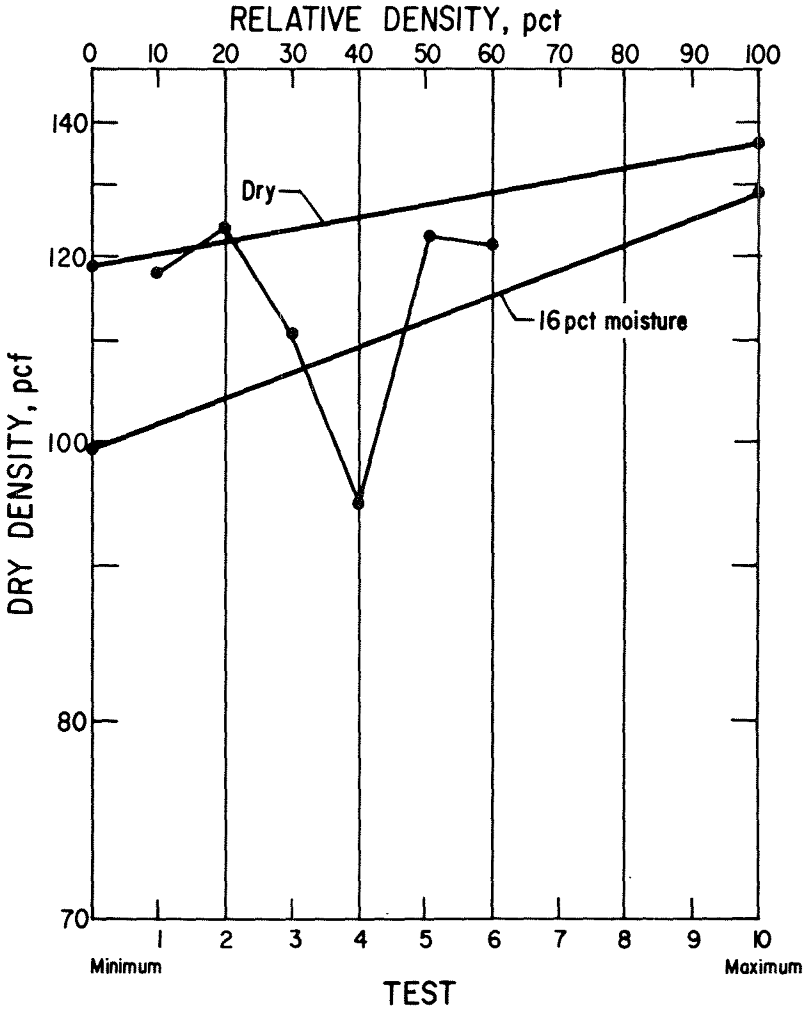


FIGURE 12. - Relative density, half gravel and half sand, Badger Hill, Calif.

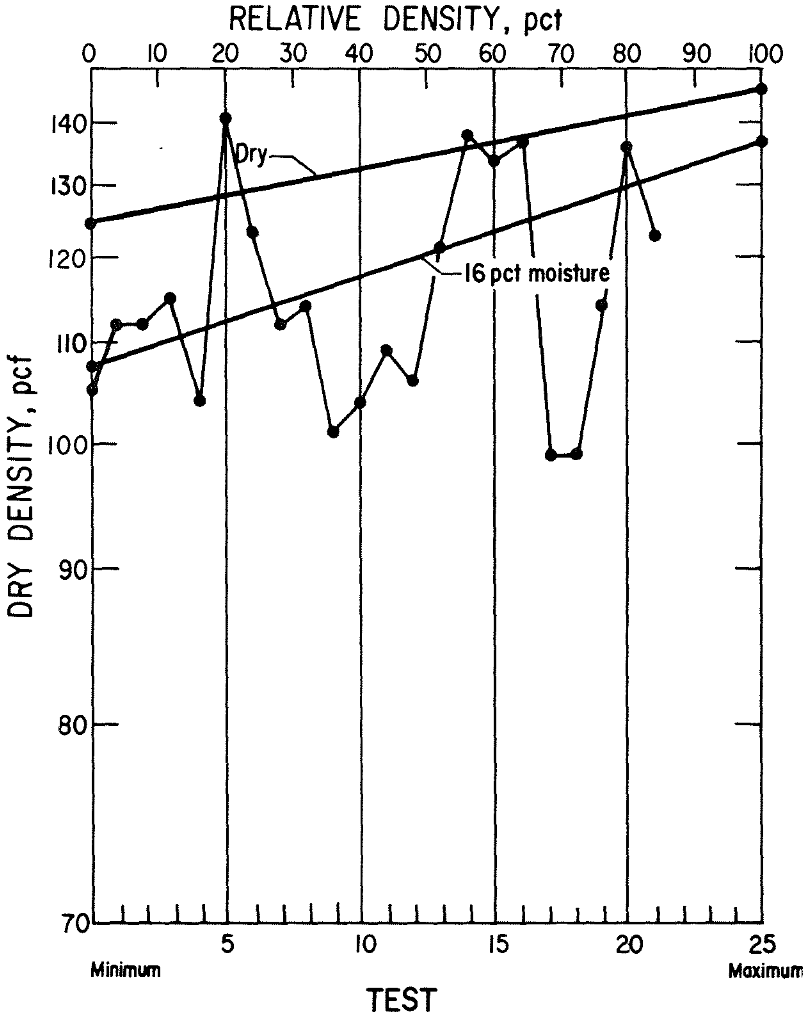


FIGURE 13. - Relative density, two-thirds gravel and one-third sand, Badger Hill, Calif.

Table 4 shows the results of 86 tests with 7 material combinations. Table 5 is a breakdown of the seven material combinations according to moisture content and revolutions per minute with averages. The sand-gravel mix at 5 to 8 pct moisture was nearly 8 pounds heavier on a dry weight basis than the same mix with 3 to 5 pct moisture. The material with a high percentage of fines was very moisture-sensitive, but the coarse material had only enough water to keep the dust to a tolerable level. Figures 14-18 show graphically the results as compared to the minimum-maximum line.

TABLE 4. - Radmark stowing test, Spokane Mining Research Center

Test	Date	Target distance, feet	Blower speed, rpm	Density, pcf		Feed rate, tph	Moisture, pct	Times used
				Wet	Dry			
NEW SAND-GRAVEL MIX--EQUAL PORTIONS OF FINE SAND, COARSE SAND MINUS 3/4-INCH PLUS 1/4-INCH GRAVEL AND MINUS 1-1/2-INCH PLUS 3/4-INCH GRAVEL								
1	11/18/71	25	1,800	120.7	115.5	67	4.5	1
2	11/22/71	30	1,800	124.4	119.5	67	4.1	2
3	11/23/71	20	1,800	121.1	117.5	93	5.0	3
4	11/24/71	30	1,800	121.0	116.2	100	4.1	4
5	11/26/71	25	1,800	120.4	116.8	100	3.1	5
6	11/26/71	20	2,000	121.4	116.6	100	4.2	6
7	11/29/71	25	2,000	120.6	115.6	100	4.3	7
8	11/29/71	30	2,000	120.9	116.7	100	3.6	8
9	11/30/71	35	2,000	119.1	114.1	100	4.4	9
10	12/ 2/71	30	2,200	124.0	118.6	100	4.5	10
11	12/ 3/71	35	2,200	115.7	110.6	100	4.6	11
12	3/17/72	35	2,200	117.58	111.84	100	5.14	12
13	3/20/72	25	2,200	117.42	112.36	100	4.5	13
14	3/21/72	20	1,600	112.70	107.45	100	4.9	14
15	3/21/72	25	1,600	108.46	103.8	100	4.5	15
16	3/22/72	25	1,800	131.6	121.3	100	8.4	16
17	3/23/72	25	2,000	131.97	123.0	100	7.30	17
18	3/28/72	20	1,800	124.10	115.70	100	7.3	1
19	3/29/72	25	1,800	127.81	120.53	100	6.0	2
[1]20	4/13/72	30	1,800	132.62	128.83	112	2.9	3
21	4/17/72	20	2,000	131.48	123.66	112	6.31	4
22	4/17/72	25	2,000	131.89	123.35	112	6.91	5
23	4/18/72	30	2,000	133.59	125.81	112	6.2	6
24	4/19/72	35	2,000	132.55	125.23	100	5.8	7
25	4/20/72	20	1,800	133.38	124.57	100	7.1	8
26	4/21/72	25	2,200	135.0	125.5	100	7.58	9
27	4/24/72	30	2,200	129.12	123.0	100	4.97	10
28	4/25/72	35	2,200	137.07	127.85	100	7.21	11
29	5/ 2/72	30	2,200	132.21	124.73	100	6.0	12
30	5/12/72	25	2,000	126.15	118.17	100	6.75	13
31	5/16/72	30	2,200	136.70	126.62	100	7.90	14
32	5/17/72	25	2,000	132.39	122.32	100	8.23	15
33	5/18/72	25	2,000	129.03	121.39	100	6.29	16
34	5/19/72	25	1,800	137.4	126.3	100	8.79	17

See footnotes at end of table.

TABLE 4. - Radmark stowing test, Spokane Mining Research Center--Continued

Test	Date	Target distance, feet	Blower speed, rpm	Density, pcf		Feed rate, tph	Moisture, pct	Times used
				Wet	Dry			
NEW SAND-GRAVEL MIX--EQUAL PORTIONS OF FINE SAND, COARSE SAND MINUS 3/4-INCH PLUS 1/4-INCH GRAVEL AND MINUS 1-1/2-INCH PLUS 3/4-INCH GRAVEL--Continued								
35	5/23/72	20	1,800	129.3	122.2	100	5.83	1
36	5/24/72	20	(2)	-	-	-	-	-
37	5/25/72	30	2,000	135.4	125.4	100	7.95	2
38	5/30/72	30	1,800	140.9	129.6	100	8.67	3
39	5/31/72	30	2,000	133.2	123.2	100	8.08	4
40	6/ 1/72	35	2,200	137.7	128.0	100	7.58	1 and 2
41	6/ 2/72	25	2,200	133.7	125.9	100	6.20	3
42	6/ 5/72	35	2,000	130.4	121.1	100	7.66	4
43	6/ 6/72	30	2,000	126.5	122.0	100	3.70	1
44	6/ 7/72	20	1,800	133.2	126.8	100	5.07	2
45	6/21/72	25	1,800	133.2	125.3	100	6.27	3
46	6/22/72	15	1,800	132.3	124.4	100	6.60	4
MINUS 3/4-INCH PLUS 1/4-INCH FINE, CRUSHED ROCK								
47	10/27/72	25	2,000	109.4	105.2	90	4.0	1
48	11/ 7/72	25	1,800	103.1	101.1	100	2.0	2
49	11/ 8/72	20	1,800	102.2	100.1	100	2.2	3
50	11/ 9/72	30	1,800	109.0	106.1	100	2.8	4
51	11/10/72	35	2,000	110.7	109.1	100	1.4	5
52	11/13/72	30	2,000	110.3	107.0	100	3.1	6
53	11/14/72	25	2,000	115.0	112.2	100	2.5	7
54	11/20/72	20	2,000	105.1	102.1	100	2.9	8
55	11/22/72	25	2,200	111.7	107.5	100	3.9	9
56	3/30/73	20	1,800	119.3	116.6	100	2.3	10
57	4/ 4/73	20	2,000	115.2	110.6	100	4.2	11
MINUS 1-1/2-INCH CRUSHED QUARTZITE								
58	4/ 5/73	25	2,200	114.7	109.8	100	4.5	1
59	4/ 6/73	25	2,000	119.0	113.4	100	4.9	2
60	4/ 9/73	25	1,800	124.4	118.25	100	5.2	3
61	4/10/73	30	2,200	117.8	113.7	100	3.6	4
NEW MINUS 1-1/2-INCH CRUSHED QUARTZITE								
62	4/11/73	30	2,200	109.0	105.2	100	3.6	1
63	4/18/73	30	2,000	112.9	106.1	100	6.4	2
64	4/20/73	30	1,800	120.7	113.7	100	6.2	3
65	4/23/73	35	2,000	119.4	115.1	100	3.7	4
66	4/24/73	35	2,200	117.9	114.7	100	2.8	5
67	4/27/73	20	1,800	110.8	106.9	100	3.7	1
68	5/ 1/73	20	1,800	116.7	111.6	100	4.6	2
69	6/ 6/73	25	1,800	116.4	112.7	100	3.3	3
70	6/ 7/73	30	1,800	118.2	116.5	100	1.4	4

See footnotes at end of table.

TABLE 4. - Radmark stowing test, Spokane Mining Research Center--Continued

Test	Date	Target distance, feet	Blower speed, rpm	Density, pcf		Feed rate, tph	Moisture, pct	Times used
				Wet	Dry			
NEW MINUS 1-1/2-INCH CRUSHED QUARTZITE--Continued								
71	6/ 8/73	25	2,000	117.2	112.9	100	3.8	5
72	6/11/73	20	2,000	121.8	117.3	100	3.9	6
73	6/12/73	30	2,000	124.2	120.5	100	3.1	7
74	6/12/73	25	2,200	120.9	114.5	100	5.6	8
75	6/13/73	30	2,200	121.8	118.8	100	2.5	9
76	6/14/73	35	2,200	121.5	117.9	100	3.0	10
77	6/14/73	35	2,000	120.2	115.7	100	3.9	11
MINUS 3-INCH PLUS 1/4-INCH CRUSHED LIMESTONE								
78	6/20/73	25	1,800	122.34	119.55	50±	2.4	1
79	6/20/73	20	1,800	122.78	120.49	50±	1.9	2
80	6/21/73	25	2,000	125.6	121.9	100	3.0	3
81	6/27/73	25	2,200	125.0	122.3	100	2.2	4
82	6/28/73	30	2,000	126.9	123.3	100	2.5	5
83	6/29/73	30	2,000	127.3	123.3	100	3.3	6
84	7/ 2/73	35	2,200	123.0	120.0	100	2.6	7
85	7/ 3/73	25	2,200	128.4	126.5	100	1.5	8
86	7/ 3/73	35	2,000	123.7	118.7	100	4.2	9

[1] Test 20 was void; the moisture was obviously wrong. If moisture was 6 pct dry γ = 125.0; at 7 pct dry γ = 124.0.

[2] No good.

TABLE 5. - Bulk density, various materials, by moisture content and compressor speeds

Test material	Moisture, pct	Bulk density, pcf			
		1,800 rpm	2,000 rpm	2,200 rpm	Average
1 Sand-gravel mix 1:1:1:1[1]..........	5-8	125.7	122.9	127.0	124.1
2 .....do...........................	5-8	122.9	122.9	125.8	
3 .....do...........................	3-5	117.8	117.2	113.4	116.4
4 Minus 3-inch limestone............	2-3	120.5	123.2	121.7	121.6
5 Minus 3-inch limestone (first use)	2-3	119.6	-	-	119.6
6 Minus 1-1/2-inch crushed quartzite	2-6	114.8	114.4	113.5	114.3
7 Minus 3/4-inch crushed quartzite..	2-6	106.0	107.7	107.0	107.0

[1] 1:1:1:1 mix--one part each of fine sand, coarse sand, minus 3/4-inch plus 1/4-inch gravel, and minus 1-1/2-inch plus 3/4-inch gravel.

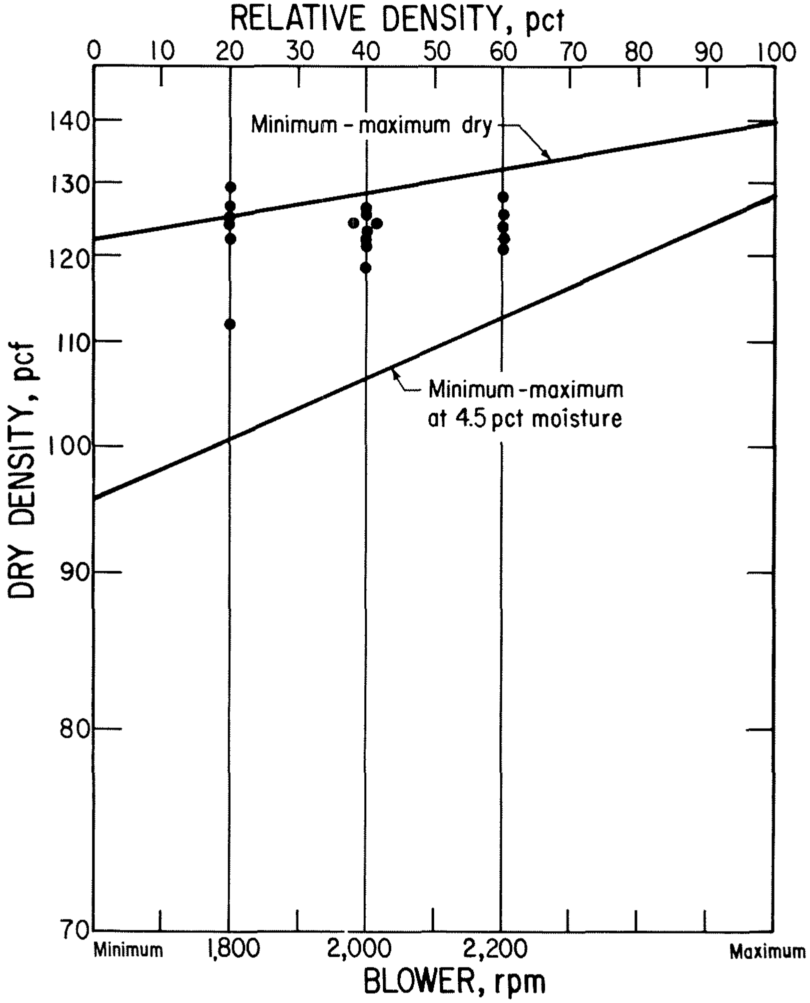


FIGURE 14. - Density minus 1-1/2-inch and minus 3/4-inch crushed quartzite at various blower speeds.

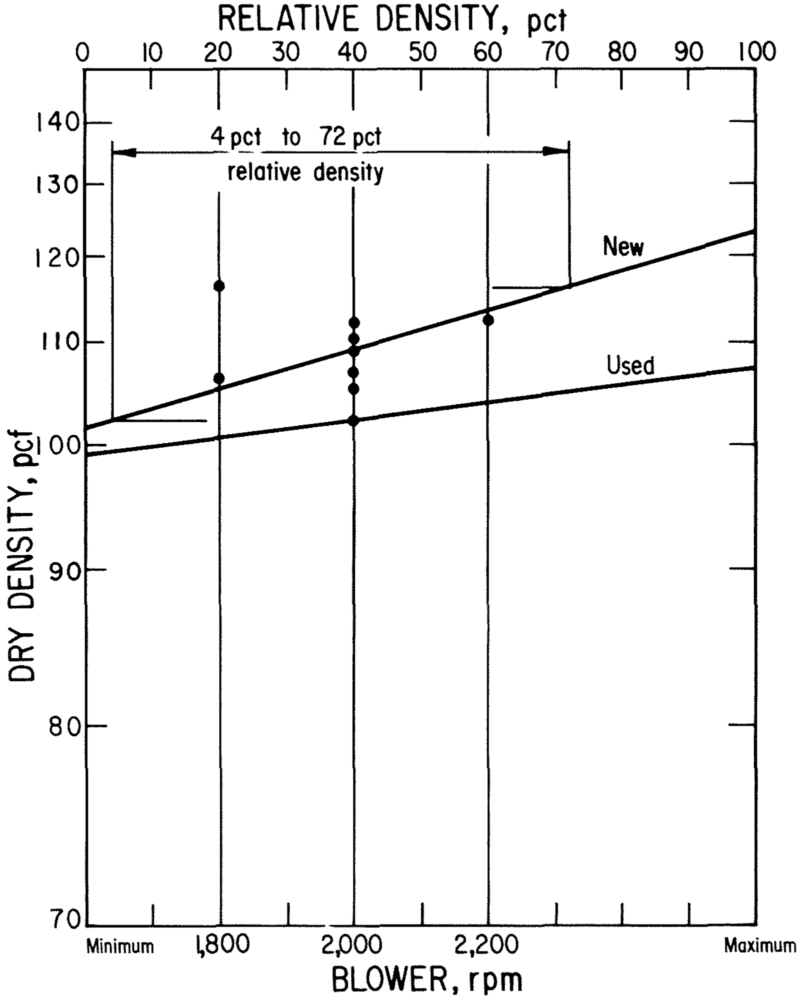


FIGURE 15. - Density minus 3/4-inch crushed quartzite at various blower speeds.

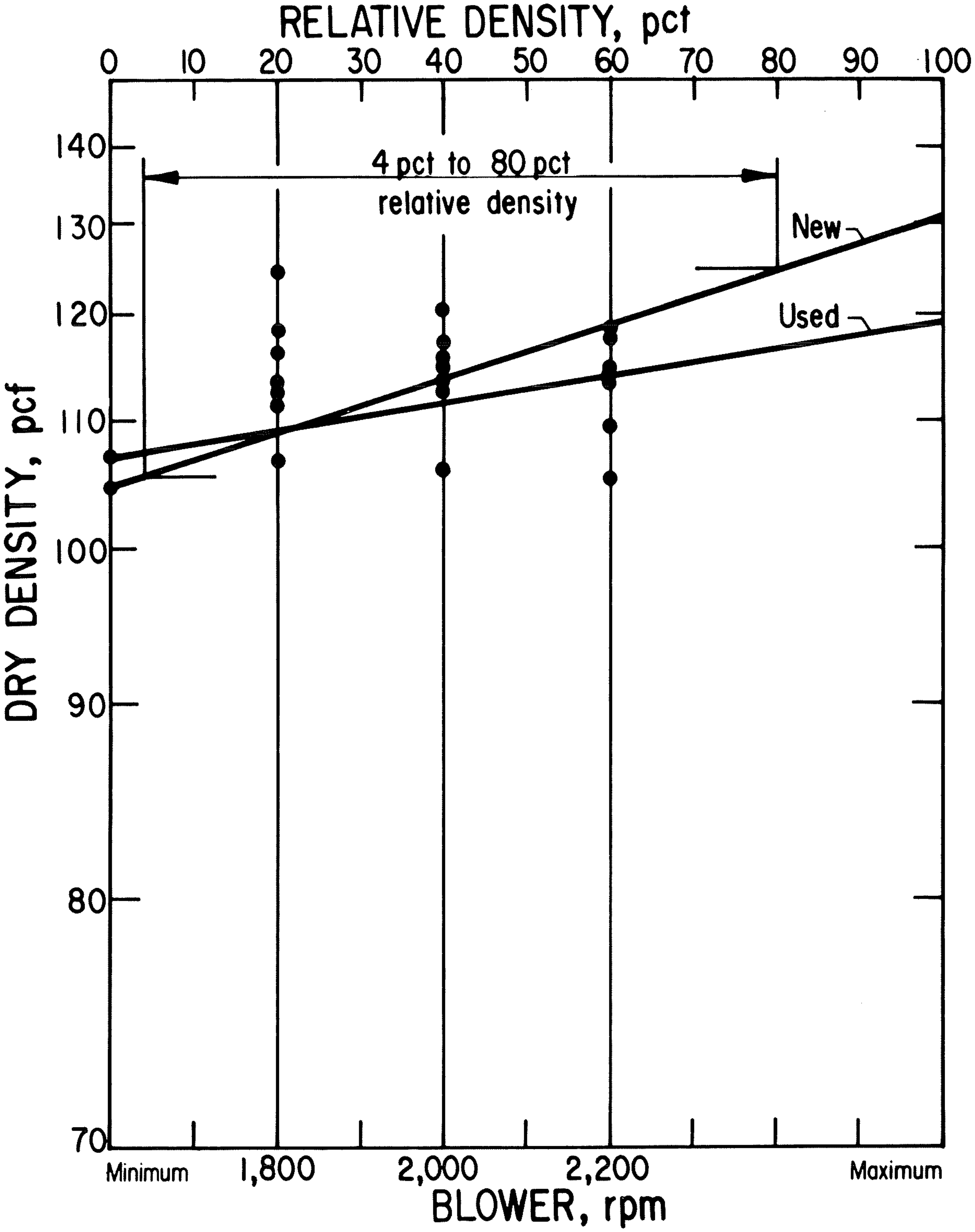


FIGURE 16. - Density minus 1-1/2-inch crushed quartzite at various blower speeds.

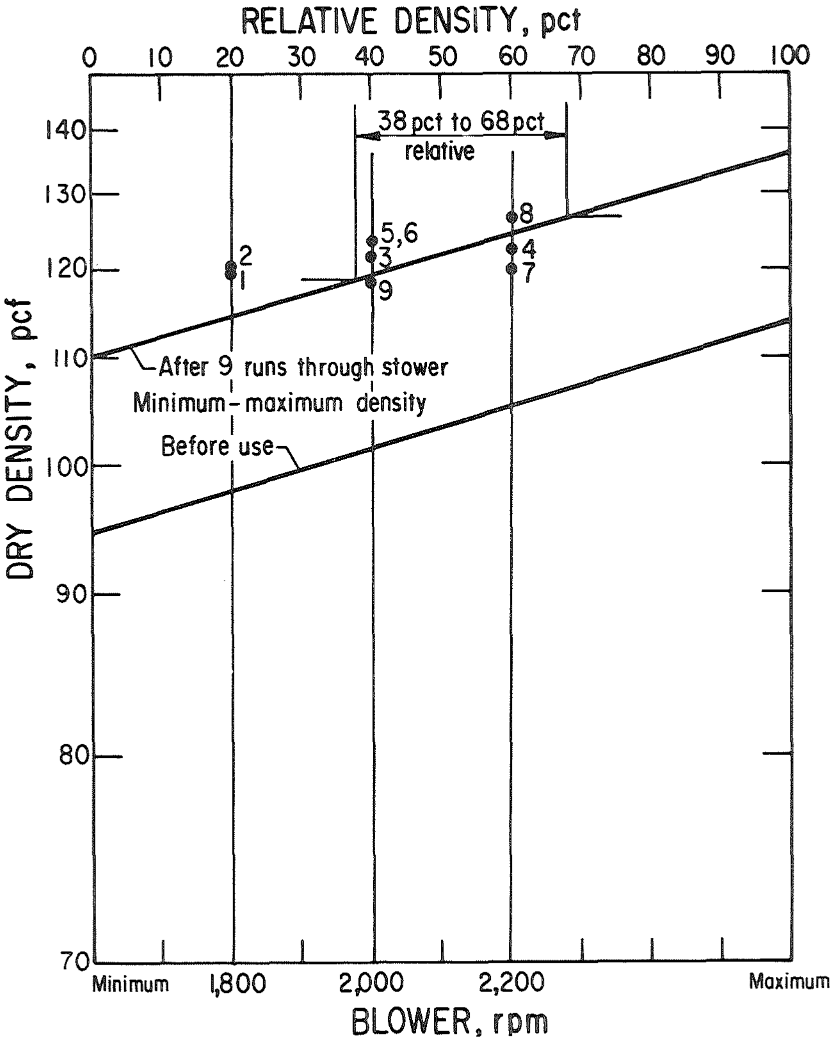


FIGURE 17. - Density minus 3-inch crushed limestone at various blower speeds.

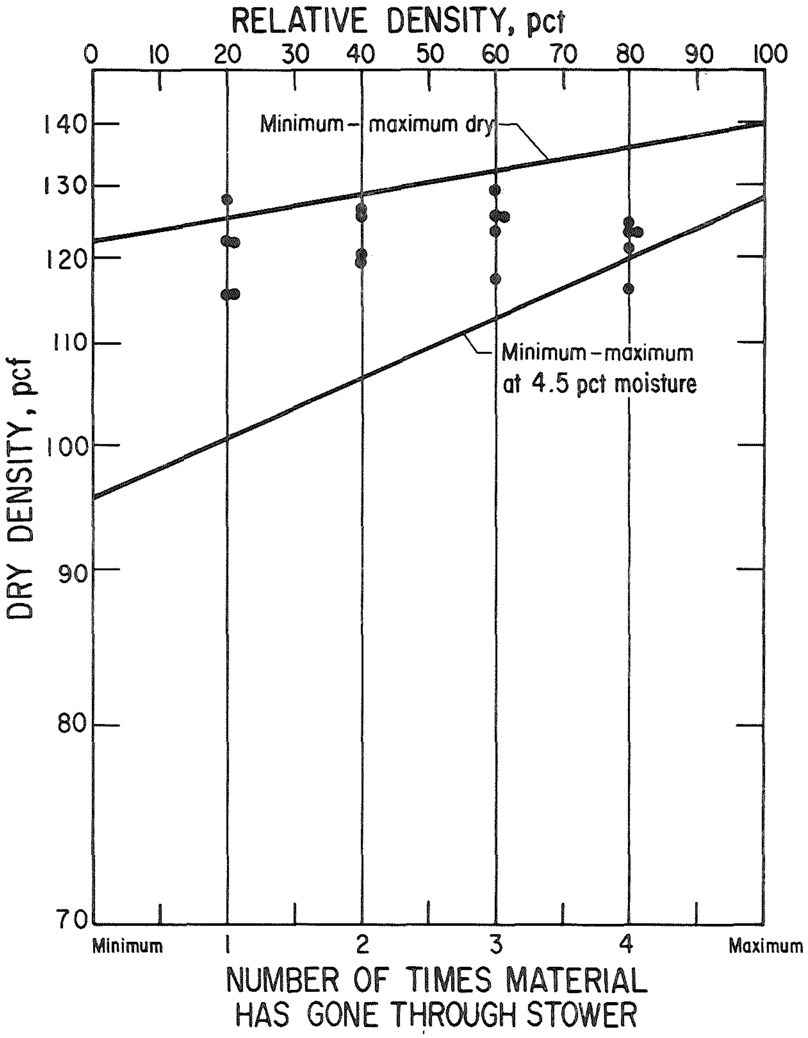


FIGURE 18. - Density 1:1:1:1 mix at 1 to 4 times through stower.

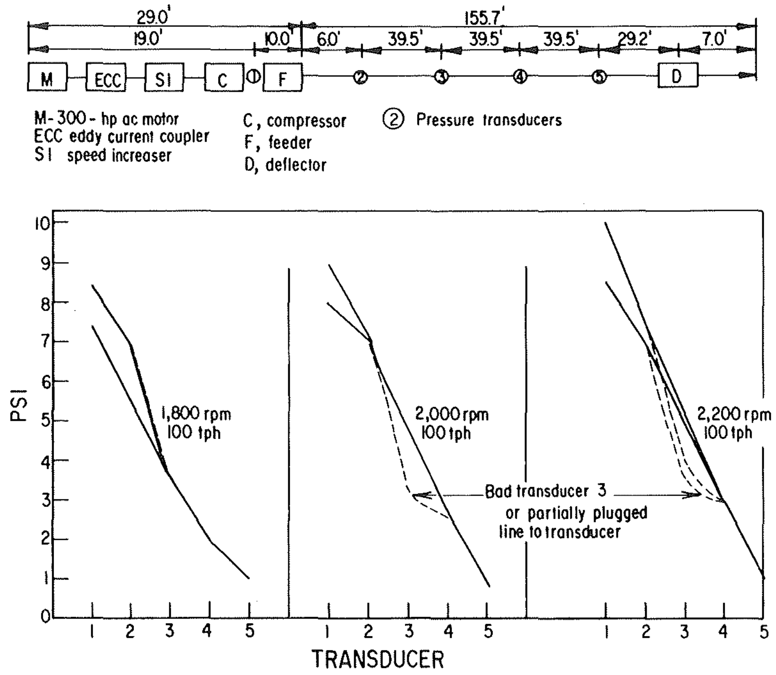


FIGURE 19. - Stower, pipe, and transducer layout and pressure drop in stowing line.

This project was terminated before another 100 tests or more could be run to fill gaps where most important information was needed.

The two most important test findings were that (1) water is very critical where fines are present, and (2) a well-graded mix with a high percentage of coarse material will have a higher density than a finer, well-graded mix for two obvious reasons: (a) a 3-inch piece of solid rock has less voids than the same volume of sand or 1/2-inch pieces, and (b) if the kinetic energy of the large particle is greater than the shear strength of the previously placed material, compaction will occur, whereas the energy of the smaller particles is insufficient to accomplish additional compaction.

Figure 19 shows the dimensions of the power train and the locations of the pressure transducers in the stowing line. The graphs in figure 20 show the pressures at each transducer.

Figures 21-23 showing density for various blower speeds and throw distances indicate that 1,800 rpm at 25-foot distance was about equal to 2,000 rpm at 30 feet and 2,200 rpm at 30 or 35 feet. This might indicate that there is an optimum impact velocity above and below which the density is less.

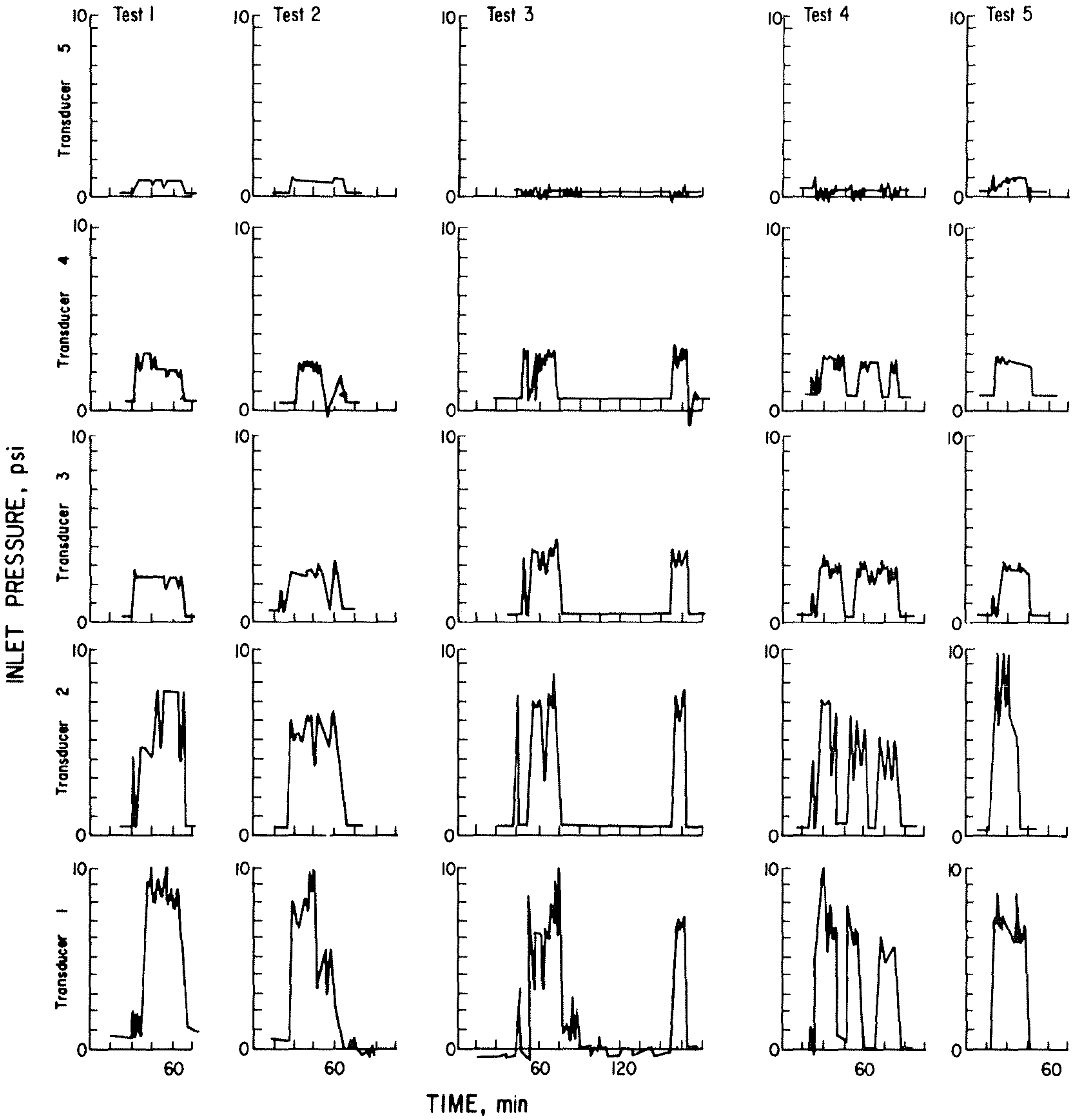


FIGURE 20. - Transducers 1-5 pressure drop.

Particle Velocity

Figures 24-25 show the particle velocity after discharging into free air at 1,800 and 2,200 rpm of the blower. A large number of counts were made from high-speed photos (200 frames per second) of individual particles for travel distances of 5 to 15 feet. Most of the good photos were of the minus 3-inch limestone that had very little fines, so individual particles could be traced continually for frame after frame and were not obscured by fines, dust, or fog. The small and medium particles were arbitrarily set at 3/4-inch to 1-inch sizes and the large particles were 2-1/2 inches to 3-1/2 inches. Particle velocity was measured in the center of the stream and 1-1/2 to 2 feet above and below the center. There was a wide range of velocities in each area

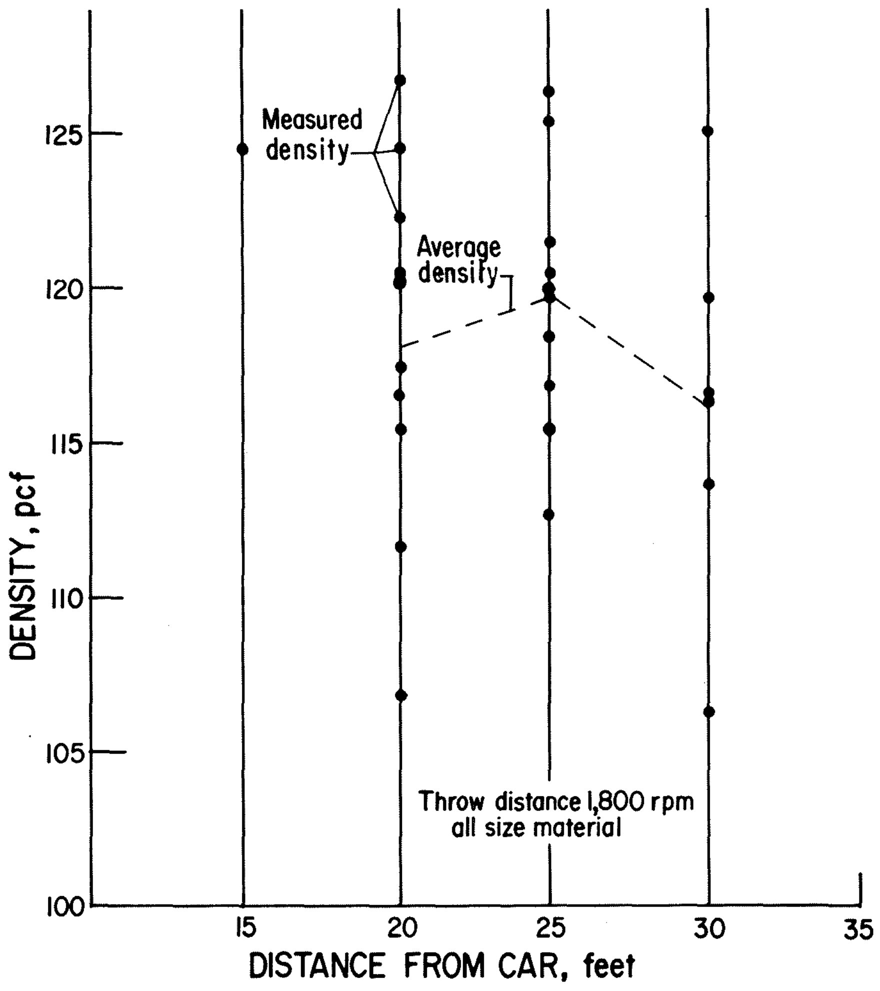


FIGURE 21. - Density versus throw distance at 1,800 rpm.

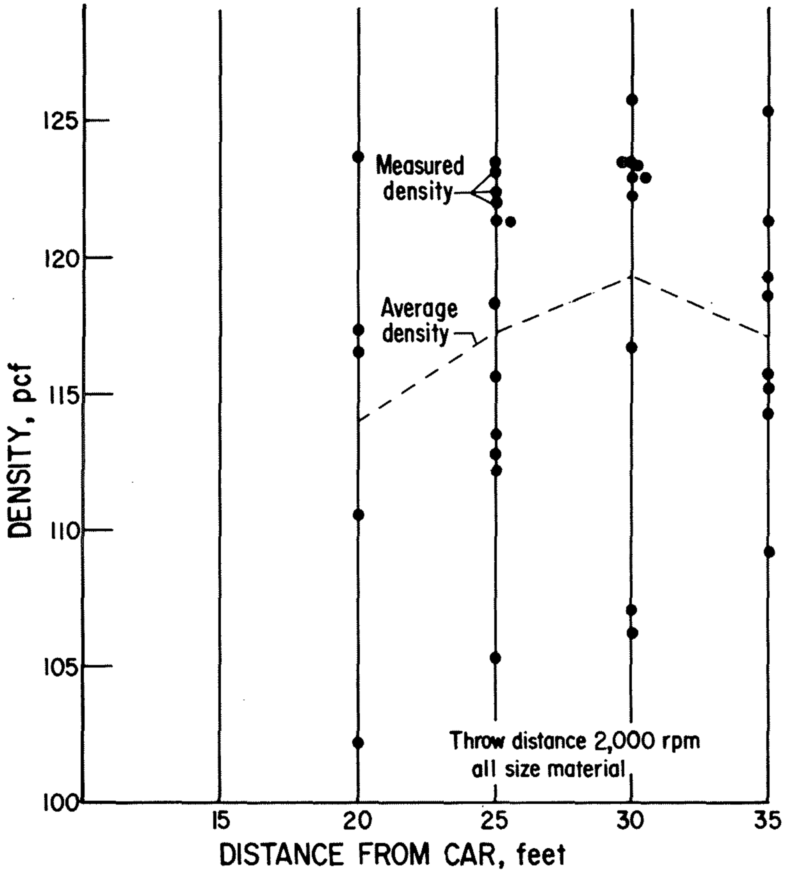


FIGURE 22. - Density versus throw distance at 2,000 rpm.

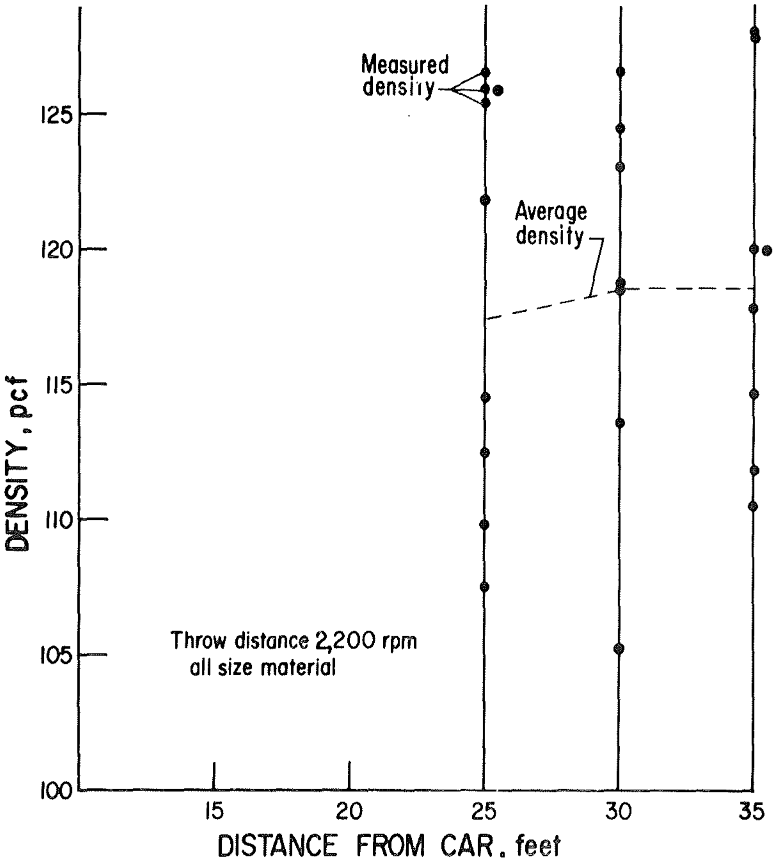


FIGURE 23. - Density versus throw distance at 2,200 rpm.

measured, but the averages set a definite pattern with the highest velocity in the center and successively lower velocities above and below the center, respectively. These velocities do not agree with some of the formulas published of theoretical velocity (9) under these same test conditions of air velocity, material size and density, and distance traveled, etc.

Definite clusters of material could be seen in the photos of the granular material, which seemed to travel long distances as clusters with no adhesion. At 1,800 rpm, the clusters had about the same speed as the average individual particles; while at 2,200 rpm, they definitely lagged behind the average about 10 fps. Coal shale could not be measured as individual particles, but many clusters were measured that had a higher velocity than the higher specific gravity granular material. The difference of 0.65 to 0.75 in specific gravity would account for this difference.

The exit velocity was checked for coal shale of specific gravity of 1.9 to 2.0 at 2,200 rpm, with a feed pressure of 5 psi and feed rate of 50 tph. The velocity of the air as it discharged from the nozzle was about 206 fps, and the particles that came out in slugs had a velocity of 160 fps for the first 10 feet of travel. As these "slugs" traveled in free air, the velocity dropped to 155 fps at 16 to 28 feet from the nozzle.

The coal shale velocity was checked also when the blower speed was 1,800 rpm and the feed pressure was 5 psi. The air velocity at discharge was 162 fps. The particle velocity for the first 10 feet after discharge was 130 fps and from 16 to 28 feet after discharge was 125 fps. This is considerably below the velocity of 2,220 rpm. Some of the fines that were dry or nearly dry at the time of discharge were blown into and out of the car and completely

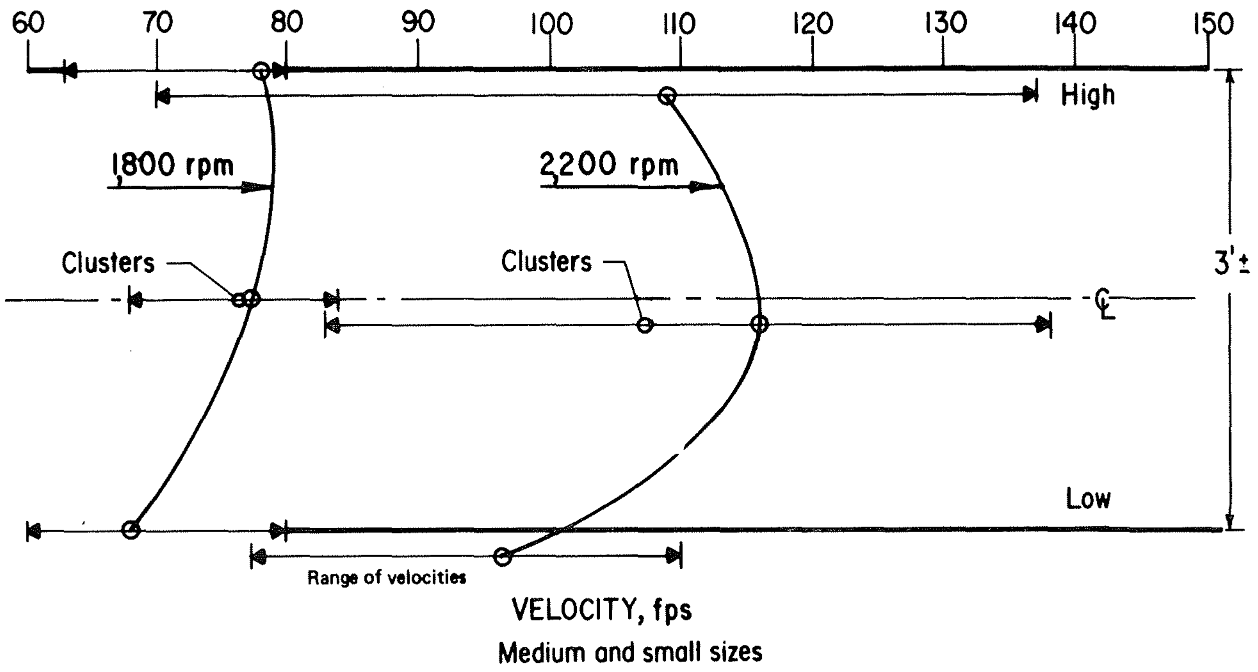


FIGURE 24. - Particle velocity after discharge, medium and small size.

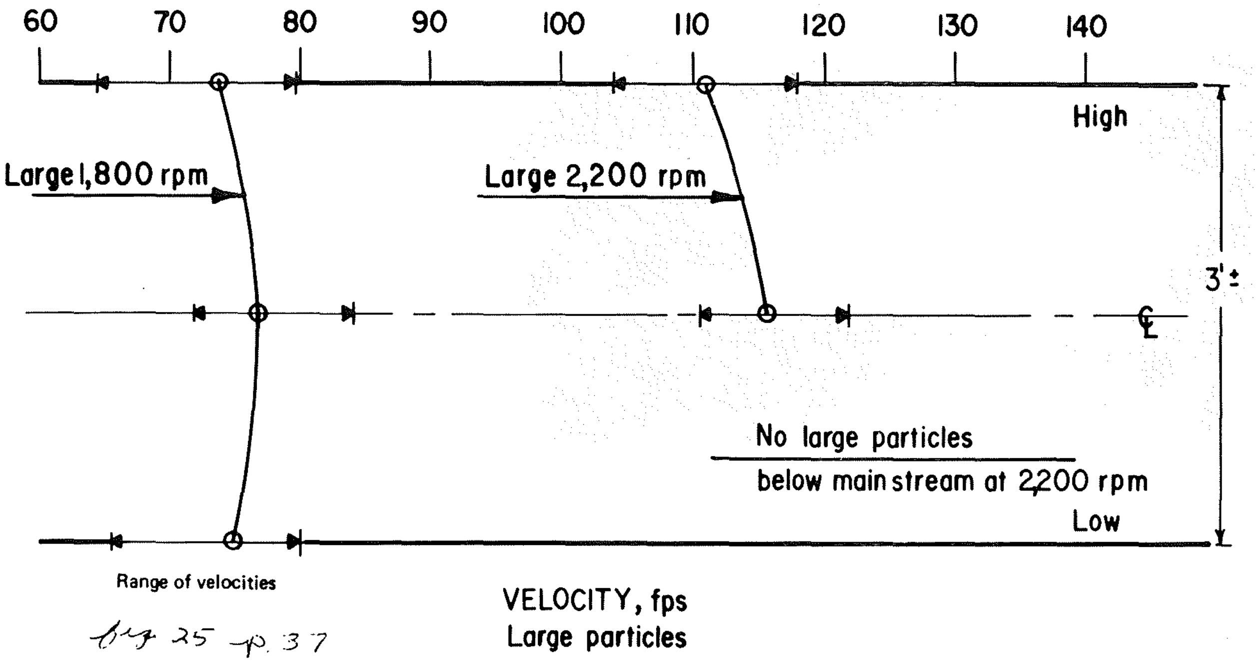


fig 25 - p. 37

FIGURE 25. - Particle velocity after discharge, large size.

out of the building because of the high velocity and volume of air in the discharge pipe. Visual observations indicated that individual small particles lost velocity very rapidly, and some particles were picked up from the surface of the stowed material by impacting air and blown around. To reduce the volume of air against the stowed material, a vented nozzle was built to release some of the air 18 inches from the end of the nozzle. There were eight vents, 2 inches by 2 inches, which took about one-third of the air from the pipe and dissipated it out the side at 160° angle from the direction of the flow. This slightly decreased the particle velocity at the nozzle but did not necessarily decrease the compaction. Compaction was essentially constant for particle velocities ranging from 130 to 160 fps.

Initially, a straight piece of 8-inch pipe was used for a nozzle. Because the scatter spread rock around the test area outside the car, a nozzle tapered from 8 inches to 7-3/8 inches ID in the last 18 inches was fabricated (fig. 26). This choke taper gave a much smaller pattern to the rock impacting on the pile, hopefully improving the density, reducing the cleanup from the scatter, and making it easier to place the rock where it was directed to create a smooth, even face that was easier to measure. A tapered nozzle would be an asset in full-scale operations in most conditions.

Most of the stowing with material of 2.65 to 2.70 specific gravity was at 100 tph, and the particle velocity was measured when the feed was also at this rate. The gage pressure at the stower (loaded) at 2,200 rpm was 8 to 10 psi, and while empty was about 2 psi, indicating air velocities of 200 fps and 215 fps, respectively. While running essentially empty (with only an occasional few rocks), the velocity of the particles in the center of the path was found to be almost exactly the same as when loaded but had a wider variation of from 64 to 146 fps.

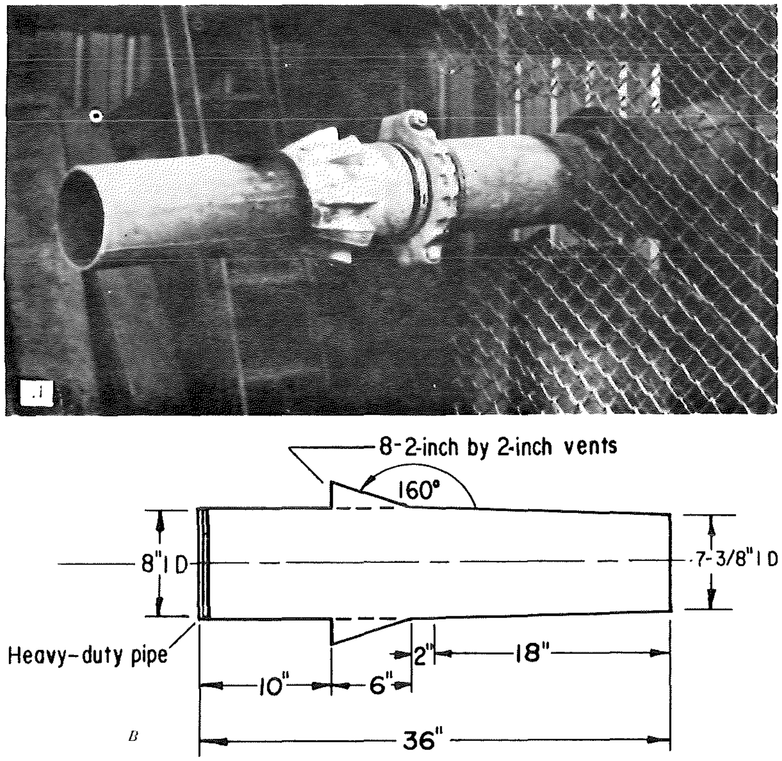


FIGURE 26. - Tapered nozzle. *A*, Tapered nozzle with air vents; *B*, tapered nozzle design.

Particle velocity near the start of the flight from the nozzle and near the end of the flight was measured to see if the particle was accelerating or decelerating. Individual particles could not always be identified the first 2 to 3 feet out of the nozzle, so most of the velocities from 3 to 8-1/2 feet were compared with the velocities from 8-1/2 to 14 feet. In some cases, velocities over five frames from 4 to 6-1/2 feet were compared with velocities over five frames from 9 to 11-1/2 feet from the nozzle. Of the 40 different particles measured, 22 seemed to increase after discharge while 7 remained the same and 11 decreased. There seemed to be no pattern as to the size or position of the particle in relation to the line of flight that might account for the differences in the velocities, and there was a wide range of velocities even in the same series of frames. In general, the particles high above the center line of the line of flight had a higher velocity than those well below the line of flight.

Line Pressure

Line pressures were measured with five pressure transducers, one between the blower and the feeder, one 6.0 feet beyond the feeder, and three more at 39.5 feet spacing along the line. Total length of pipe from blower to the end of the nozzle was 155.7 feet. A pressure gage on the operating panel was read on each test run to check with transducer 1, which measured the pressure at the same place. Transducers 2-5 had two outlets from the pipe so that if one became plugged with dust, the other might stay open. This also meant that with two openings, cleaning was required less frequently. Figures 19-20 show the operating pressures at 1,800, 2,000, and 2,200 rpm at 100 tph feed. The fluctuations of the pressure in transducer 1 are caused by the feed entering the airstream intermittently with a load of 16.7 pounds each 1/3 second. By the time the stream reaches the second transducer, which is 6 feet in front of the feeder, the fluctuation is less and is not noticeable at the third transducer. After plotting the pressure curves, it becomes apparent that transducer 3 was plugged or malfunctioning in some way as shown by the jog in the line at this point on the 2,000- and 2,200-rpm plots.

Transport Distance, Pipe Wear, and Cost

No attempt was made to determine how far the 300-hp motor and power train would deliver a given load for transport alone without regard for velocity of the material as it was discharged. Neither was the effect of 90° elbows in the line measured. This has been covered quite well by British National Coal Board Translation A. 261 (17). Operating cost and pipe wear were not evaluated because of the limited nature of the test facilities, intermittent operation, and short testing time. Some information on cost is contained in the Cominco report (19).

Power Requirements

The modified power train used on this project was different from what would normally be expected in a commercial operation in that the speed of the blower could be precisely varied from 1,600 to 2,200 rpm. This was essential for test purposes to get the optimum compaction for each material. The Westinghouse 300-hp, 3-phase, 60-cycle, 480-volt, 326-amp, 1,765-rpm motor was direct-connected to an eddy current coupler and a mechanical speed increaser to the blower. The power consumption at a feed rate of 100 tph at 1,800, 2,000, and 2,200 rpm was calculated as 173, 199, and 225 hp, respectively.

$$\text{Horsepower} = \text{amps} \times \text{volts} \times \sqrt{3} \times \text{power factor} \times \text{efficiency} \times 1.341;$$
$$\text{Power factor} = 0.9;$$
$$\text{Efficiency} = 0.9. \qquad (1)$$

The power usage for the 300-hp motor at 1,800, 2,000, and 2,200 rpm was 143.4, 165.0, and 186.4 kwhr, respectively.

$$\text{Killowatt-hours} = \text{amps} \times \text{volts} \times \sqrt{3} \times \text{power factor } (0.9). \qquad (2)$$

At 100 tph rate of stowing, this is less than 2 kwhr/ton at the highest speed. A 15-hp motor on the hydraulic pump powered the motor on the feeder. Power consumption on this motor or on the 2-hp motor on the conveyor was not measured.

The demand charge would be about equal to the power cost if the equipment was operated 5 hours per day and 20 days a month, or 10,000 tons per month. At this rate, the total power cost would be less than 4 kwhr per ton for transporting the material 157 feet with a vertical rise of 6 feet in this distance.

SECTION III--DISCUSSION OF RESULTS

Density

It is very fortunate that the authors had the experience with stowing and taking density samples of the in-place material in the Tertiary gravels at Badger Hill in California before building the test facilities at the auxiliary laboratory. Measuring and weighing the entire box to obtain average density was considerably more accurate and faster than obtaining individual samples. Some of the improvements in density were due to technique, a straight discharge line and some due to the tapered nozzle. The tapered nozzle concentrated the stream flow after discharge into free air into an area of 3± feet in diameter even at distances of 15 to 20 feet, allowing the material to be placed where desired. The rebound was less, probably because many particles that started to rebound were pushed back into the face by successive particles that hit them. This is a very inefficient use of energy when comparing the foot-pounds of energy delivered to the pile to produce the density as compared to the standard proctor test. Operator experience and training does improve the in-place density of the fill. By starting along the floor and directing the material back and forth and gradually building up the pile, the face assumes a 45° to 50° angle. When the rebound collects at the toe of the pile, a sweep or two of the nozzle will pick this up and place it against the toe and improve the density. The bottom foot or more of the pile probably had the lowest density of any other part. If stowing is in an enclosed area, such as the box used here or in actual operation in a flat-bedded deposit, the back of the fill probably has the highest density of all. Driving the material into the narrow 45° to 50° angle against the back confines it, and most of the material does not have time to rebound as it does when striking at an obtuse angle in the middle of the face. This gives a high density with lower percentage of rebound. When continually moving the deflector horizontally and vertically during stowing, there is a big difference in density in various parts of the box. On one occasion, a large chunk of coal shale rolled out on the floor when the box was dumped and a 1-cu-ft block was sawed out of it which had a dry density of 90 pcf. This is well above anything attained in either the barrels or the box with this material.

Materials 1 and 5 as tested had 50 pct and more minus 1/4-inch material, and stowing results were very sensitive to water content for both materials. The crushed rock had no fines, so water for dust suppression only was needed for the minus 3-inch, minus 1-1/2-inch, and minus 3/4-inch materials.

Gradation was found to be very important. A maximum of minus 3-inch plus 1/2-inch, well-graded material with just enough minus 1/2-inch well-graded particles to fill the voids of the larger material should be ideal for high density fill. This material requires a minimum of water to eliminate the dust and keep the fines at a moisture content high enough to avoid bulking. This water could be added in the line just ahead of discharge to keep the transport load down. Table 5 shows bulk density of stowed material.

Bulking can be explained in the following way. If a damp sand is loosely deposited, for instance by dumping, the cohesion prevents the soil particles from settling into stable positions and reduces the intergranular bearing capacity of the sand almost to zero. The volume of such a sand may exceed that of the same sand in a loose dry state by 20 to 30 pct.

If this project had not been stopped prematurely, the material described would have been tested to see what improvement could be made by adding a small amount of fines to fill up the void in material 4 (coarse limestone). As noted in the minimum-maximum density before use and after nine uses, the material seemed to make its own fines (fig. 7) and 38- to 68-pct relative density measured on the higher minimum-maximum line (fig. 17). The density achieved on even the first run with the minus 3-inch material was well above the maximum of that material before it had been used. This would indicate that a large amount of fines was produced during the first run, or the sample used for the "before" minimum-maximum test was not representative of the 3 tons actually stowed. A truly representative sample of this size material in a 0.1-cu-ft sample is very difficult to obtain, so both of these reasons may have contributed to the high relative density.

The coal shale was entirely different from the sand-gravel mix in that there were many more fines, especially where the material was used more than once. A standard and modified proctor test showed optimum moisture contents of 20 and 17 pct and a density of 84.5 and 89.0 pcf, respectively, so it was thought that the moisture should approach this figure. After some trial tests, 16 to 18 pct seemed to give the best results (figs. 27-29). One test had a higher moisture content and the material in the car seemed saturated. As soon as the stowing stopped, the entire front face slid out of the box. It was as if the force of the stowing material hitting the face had kept it in equilibrium, and when this force was removed, it slid.

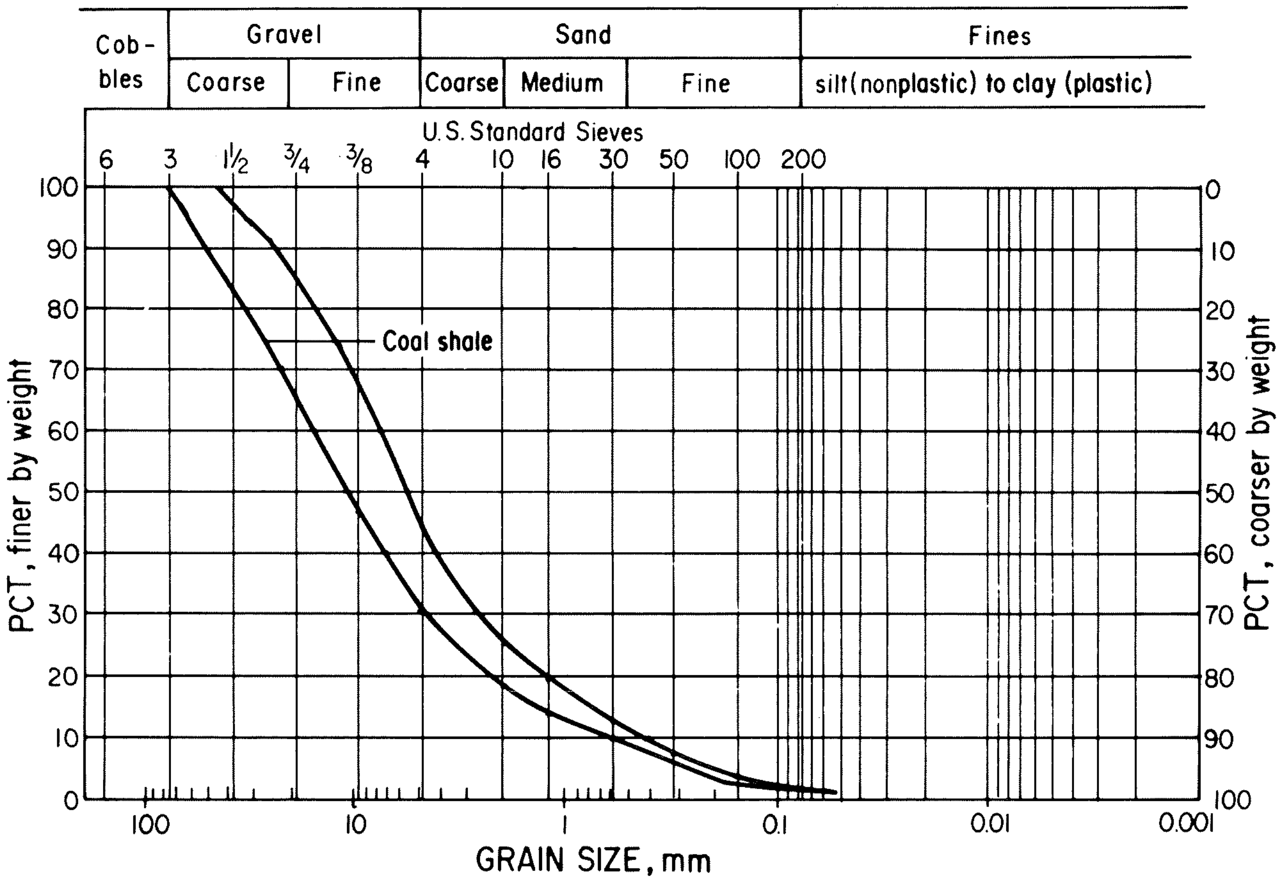


FIGURE 27. - Typical screen analysis, Black Diamond coal shale.

In August 1971, stowing tests were begun with the coal shale by blowing the material into 55-gal drums before the stowing box was completed (table 6). The best density obtained was 80.2 pcf, and 5 of 15 had densities above 79 pcf. This is better than any densities obtained in the large box later and can be attributed to the confined area being filled. The larger particles impacted the face and compacted the material, and much of it rebounded outside the barrel. The material left inside the barrel was compacted by approximately twice its own weight during this stowing time. When stowing in the large box (table 7), most of this rebound material stayed in the box and was compacted poorly by an occasional sweep of the material against it.

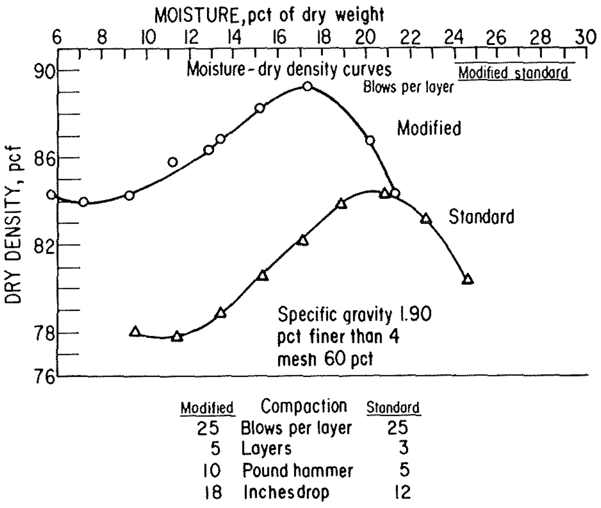


FIGURE 28. - Standard and modified proctor, Black Diamond coal shale.

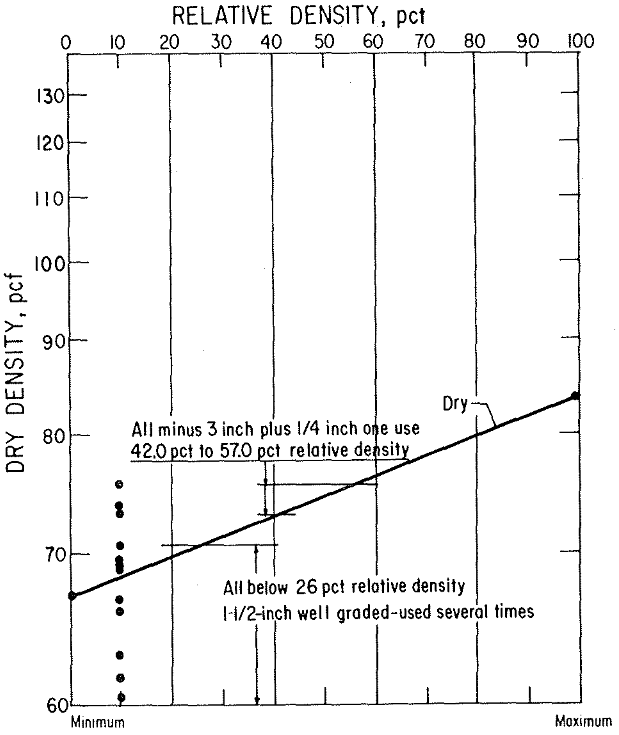


FIGURE 29. - Density Black Diamond coal shale.

TABLE 6. - Stowing coal shale in 55-gal drums

Date	Density, pcf	Order	Permeability, cfm/cu cm	Order	Moisture, pct
9/14................	80.2	1	5.5	6	19.2
9/22................	79.8	2	6.0	7	12.9
9/24................	79.8	3	8.3	9	16.3
9/17-20.............	79.5	4	47.4	15	19.0
9/13................	79.3	5	9.0	11	16.9
9/23................	79.0	6	5.0	5	15.0
9/14................	78.8	7	13.4	13	16.9
9/24................	78.8	8	3.6	4	14.4
9/23................	77.1	9	1.85	3	12.5
9/23................	77.0	10	6.5	8	15.0
9/17-20.............	76.9	11	1.7	2	15.8
9/22................	76.7	12	13.0	12	13.9
9/24................	75.4	13	14.0	14	13.3
9/24................	75.2	14	8.5	10	14.7
9/13................	72.2	15	1.7	1	15.7
Average........	77.7	-	-	-	-

TABLE 7. - Stowing data with coal shale, 1971 and 1972

Date	Stowing distance, feet	Back pressure	Feeder, rpm	Moisture, pct	Wet density, pcf	Dry density, pcf	Average
10/10/71.	26	5.5/6.0	25	15.0	81.2	70.6	65.2
10/14/71.	25	3.5/7.2	25	15.0	79.1	68.8	
10/15/71.	25	3.5/6.5	25	16.5	80.5	69.0	
10/19/71.	25	3.0/8.0	25	17.9	78.5	66.6	
10/21/71.	25	4.0/6.0	25	16.8	76.5	65.9	
10/26/71.	20	4.0/6.0	25	18.0	71.5	60.9	
10/27/71.	20	5.0/6.0	25	18.6	75.3	63.5	
10/28/71.	20	5.0/7.0	25	21.9	75.5	61.9	
10/29/71.	20	4.5	25	18.1	72.4	61.3	
11/4/71..	25	-	25	18.0	75.3	63.8	
7/6/72...	25	4/5	25	20.0	79.0	65.8	68.2
7/10/72..		-	25	19.0	73.4	61.7	
7/18/72..		4.5/5.3	25	20.35	83.17	69.1	
7/20/72..		7/8	25	18.0	71.3	60.4	
8/1/72...		6/7	25	15.1	72.48	62.97	
8/2/72...		5/6	25	20.0	83.43	69.5	
8/7/72...		-	25	17.0	78.14	66.79	
8/9/72...		5.5/6.0	25	17.17	80.81	68.97	
9/19/72[1].	25/18	6/7.5	25	16.4	82.22	70.6	
10/4/72[2].		5/6	25	16.36	86.04	73.94	
10/11/72[2]		6.5/7.5	16	15.64	84.65	73.2	
10/13/72[2]		5/6	20	16.98	88.67	75.8	

[1] Vented nozzle.
[2] Coarse shale used.

One of the disappointments of this series of tests was the difficulty or near impossibility of placing the low-gravity, friable, and fine coal shale and oil shale into a high-density pack. The fine material could not be moved and stowed without some dust. The free carbon and fines in the oil shale lost moisture rapidly during transit through the warm pipe and warm air that heated up after a few minutes of operation. Proctor tests show that the oil shale could be mechanically compacted to 96.6 pcf at 21 pct moisture, far above the 70 pcf attained with the stower (fig. 30). The moisture content was not near optimum in this limited test, so the density would have been a bit higher if additional tests could have been run at a high moisture content. The oil shale residue behaved like the coal shale, except that it was all minus 1/2-inch and would be even more difficult to attain a high relative density.

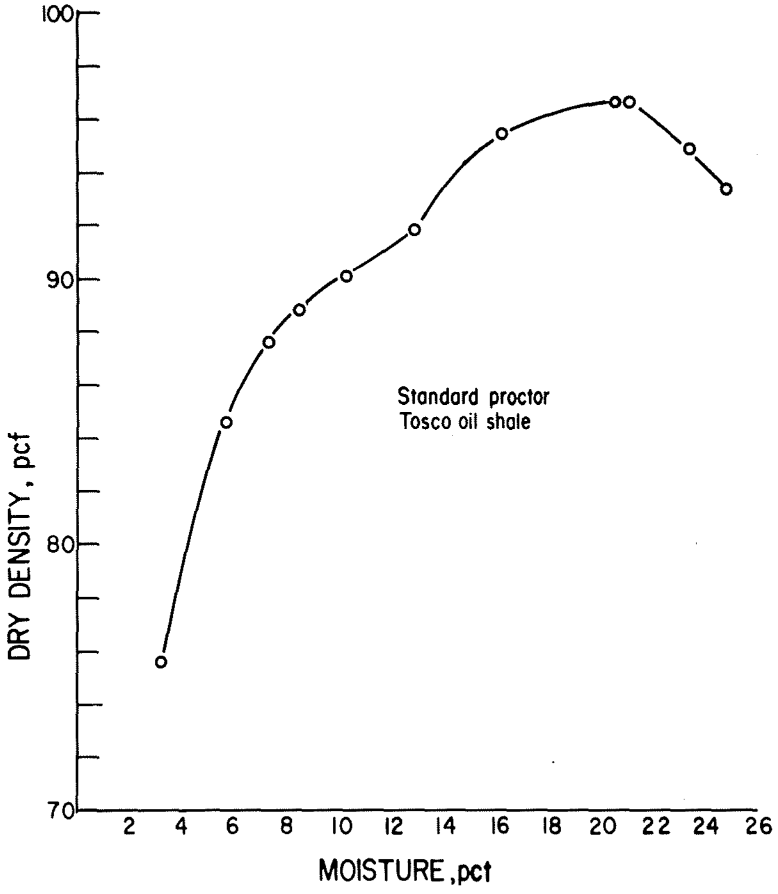


FIGURE 30. - Standard proctor, Tosco oil shale.

Conclusions

Advantages of Stowing

Pneumatic stowing has some advantages for stope filling or other special jobs of a similar nature:

1. It can produce a higher relative density fill than other methods normally used in mining but must be done with proper materials and under controlled conditions (fig. 31).

2. No large amount of water is used, thus reducing the water that must be pumped out of the mine.

3. There is less spill (and resulting cleanup) with pneumatic transport than with hydraulic transport.

4. Larger size material can be stowed with the present equipment, and it would be very easy to design and build a machine to handle material twice as large.

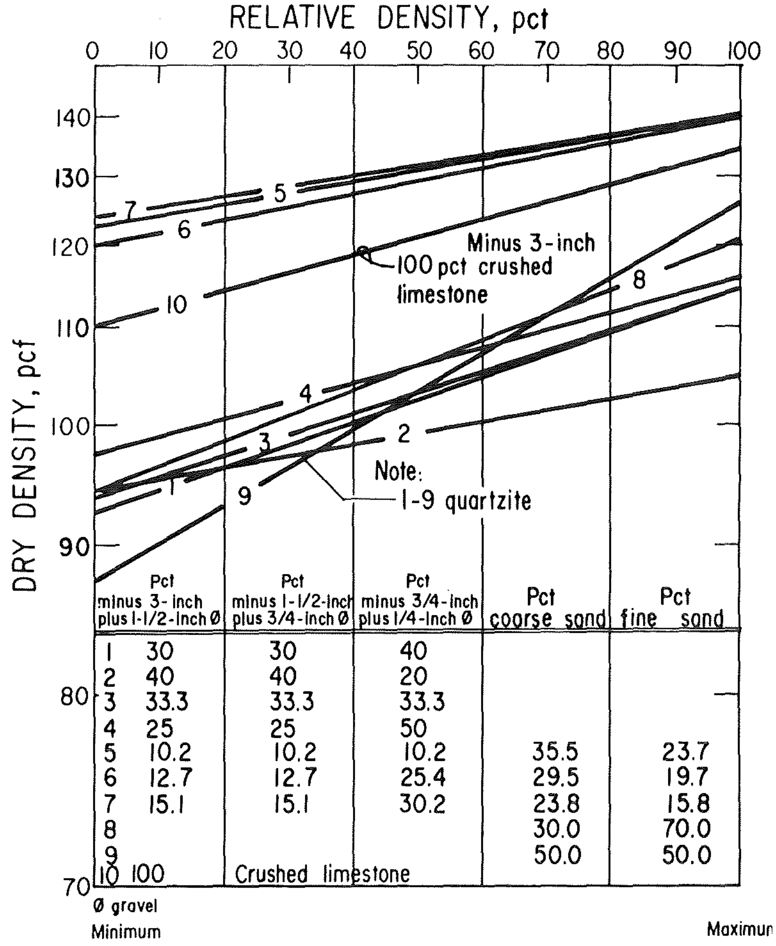
5. In flat-bedded deposits, the fill can be placed tight against the back.

6. Stowing in confined places, such as a drift with backfill for a blast, is ideal with the pneumatic stower. A saving in time, labor, and money is forecast.

7. Higher coal recovery may be possible with room-and-pillar mining using stowing for pillar recovery.

8. Surface subsidence could be lessened but not eliminated in both room-and-pillar and longwall mining systems.

9. Only bulkheads sufficiently constructed to confine stowed material are required and do not need to be watertight as with hydraulic fill.



	Pct minus 3-inch plus 1-1/2-inch Ø	Pct minus 1-1/2-inch plus 3/4-inch Ø	Pct minus 3/4-inch plus 1/4-inch Ø	Pct coarse sand	Pct fine sand
1	30	30	40		
2	40	40	20		
3	33.3	33.3	33.3		
4	25	25	50		
5	10.2	10.2	10.2	35.5	23.7
6	12.7	12.7	25.4	29.5	19.7
7	15.1	15.1	30.2	23.8	15.8
8				30.0	70.0
9				50.0	50.0
10	100	Crushed limestone			

FIGURE 31. - Minimum-maximum density, various materials.

10. The average of all the stowing tests (excluding coal) had a higher density than the six stope sand fills. The stowed material had just two-thirds the volume of voids of the sand fill (table 8).

Disadvantages

1. Pneumatic stowing is not well adapted to small cut-and-fill stopes for several reasons:

a. The number of elbows the material would have to traverse to get into the stope would result in low velocity and low density.

b. Short-time duration for filling in each stope as compared to the time and labor of preparing for the fill.

c. It could not produce a smooth, level fill for a working floor without a bulkhead or some such arrangement or be topped off with a cemented sand fill as is done at present. This would mean two fill systems for the mine, and unless pneumatic stowing made a denser fill and could help in ground support and reduce the rock burst, it would probably not be warranted.

2. Pipe wear is a big cost item when stowing an abrasive material in a long line.

3. For an underground installation in a small mine, the equipment is large and difficult to move and set up. A semipermanent setup is almost mandatory.

4. Getting enough material to keep the equipment working at capacity would be more difficult underground. For this reason, a surface installation (if possible) is ideal.

5. High initial capital cost, plus high operating cost (based on European experience).

6. High noise level.

TABLE 8. - Void ratio and percentage of voids of stowed material as compared with underground fill

Test material	Specific gravity	Density, pcf	Void ratio	Pct voids	Pct voids (average)
1 and 2.............	2.65	124.1	0.33	24.95	29.15
3..................	2.65	116.4	.42	29.61	
4..................	2.65	121.6	.36	26.46	
5..................	2.65	119.6	.38	27.67	
6..................	2.65	114.3	.45	30.88	
7..................	2.65	107.0	.55	35.30	
Sandfill 1..........	2.65	90.0	.84	45.47	44.07
Sandfill 2..........	2.65	100.0	.63	39.53	
Sandfill mine A.....	2.89	100.0	.80	44.55	
Sandfill mine B.....	2.83	104.0	.70	41.11	
Sandfill mine E.....	2.82	95.0	.85	46.03	
Sandfill mine H.....	2.96	97.0	.90	47.48	

NOTE.--1 and 2 have 56.6 pct of the void space of all sandfill averages.
4 has 60.0 pct of all sandfill averages.
5 has 62.8 pct of all sandfill averages.
All stowed material has 66 pct of all sandfill averages.

Current Applications

At the present time in the United States there is no stowing being conducted in coal mines, and as far as is known, there never has been except on a limited test basis. Pneumatic stowing could possibly be used in the near future as a way of eliminating all surface disposal by placing all coal washery wastes into the coal mines. This could be done through boreholes from the surface, piping mine wastes into abandoned workings through an entry, or hauling back into an operating mine and stowing into the mined area.

Crushed mine waste has been stowed into a large long-hole stope (17) for several years and has done quite well; whether it will become standard practice at this mine is not known, but it does place the fill where it is wanted and seems to serve the purpose very well.

Desert sand is being pneumatically placed into a southwest U.S. uranium mine (1, 17) from the surface vertically through a 1,450-foot borehole and horizontally several hundred feet with up to nine bends in the pipe. It costs less than the hydraulic fill used previously; the mine floor is dry, which is important for the rubber-tired equipment; the fill is tight against the back, confined laterally by light bulkheads that do not have to be watertight; and the filling rate is at the capacity of the feeder because of ample surface material close at hand. A tight fill was required to prevent subsidence, which could bring water into the mine from a water-bearing strata several hundred feet above. The high labor cost of moving the heavy double-wall pipe in the confined areas of the mine where equipment could not be used was corrected by using a lighter standard mild steel pipe. The lower labor cost will more than offset the added pipe cost. The mining system is being altered somewhat to make it more compatible with the pneumatic stowing. The rooms off the main drift are to be at about a 45° angle instead of 90°, and the mining sequence is to be changed. All this is to reduce the number and angle of bends in the system and minimize the amount of pipe to be moved with each fill. This could cut the labor cost more than 50 pct underground, which is now the main item of expense, even though the present total cost is less than it was with the fill used previously. Pneumatic stowing in this type of mine is ideal because of the flat terrain, the ease of drilling boreholes, the availability of fill material, and the relatively easy portability of the equipment on the surface. The most important item for a dense fill with this material is the water content, which must be high enough to be above the bulking moisture as stated previously.

Because of the potential of pneumatic stowing to provide a denser fill, eliminate problems of excess water, and handle coarse (minus 3-inch) waste material, the Bureau of Mines initiated this research to evaluate the support capability of pneumatically emplaced material. Limited field testing led to a controlled test series to determine the effect of various parameters on the quality (density) of the placed fill.

The stower is being experimentally used for hoisting coal shale from an abandoned mine to the surface through 90 feet of vertical pipeline in Bruceton, Pa. The blower and motor are on the surface and the feeder is underground. The front-end loader picks up the shale and delivers it to a crusher and from there to the feeder (3).

Coal and rock were hoisted vertically a distance of 1,300 feet at a rate of 45 and 25 tons per hour, respectively, in the South Durham area of England (3).

The stower was used to backfill many thousand feet of drifts and crosscuts for a large powder blast of a copper ore body in the Southwest recently for in-place leaching. It was a big labor saver and much cheaper than other possible methods (17).

It has been used with a tunnel boring machine for muck disposal up to 1,000 feet horizontally and less than 100 feet vertically where the bore was too small for efficient rail haul (1). Here the air supply came down from the surface to the boring machine. Two telescoping pipes for incoming and outgoing air allowed continuous movement of the machine with pipe changes each 10-foot advance.

There is some interest in the machine for hoisting ore and waste from a mine. One company has purchased a machine and plans to test it for hoisting 25 tph a vertical distance of 1,000 feet. Another mine was considering using it as a method to hoist ore from small offshoots of the ore body that is relatively near the surface but isolated from the main part of the mine.

There will undoubtedly be many more places where this equipment can be used to advantage in the future, but the largest use could possibly be to dispose of coal waste as described earlier.

Future Research

Further research must be done before the ultimate uses and limitations of stowing are known:

1. Testing to see how far a given material could be transported with given air pressure and volume.

2. Vertical pneumatic hoisting for all phases but especially power use, pipe wear, ultimate height per lift, and feeder design change if necessary.

3. Complete the original series of tests of material that had been planned for this project.

4. The effect of 90° elbows in the line with close monitoring with pressure transducers.

5. A detailed study of power usage.

6. An operational cost analysis.

BIBLIOGRAPHY

1. Ball, D. G. Pneumatic Backfilling. Paper pres. at Northwest Min. Assoc., Spokane, Wash., December 1969, 6 pp.; available on request from Radmark Engineering, Ltd., 245 Fell Ave., North Vancouver, British Columbia, Canada.

2. Ball, D. G., and J. E. Powell. High Capacity Pneumatic Conveyor Transport in Underground Mines. Royal School of Mines, London, England, 1972, 16 pp.

3. Ball, D. G., and D. H. Tweedy. Pneumatic Hoisting From Underground. CIM Bull., January 1975, 10 pp.

4. Bennett, H. B. Pneumatic Stowing From the Surface at Donisthorpe Colliery. Inf. Bull. No. 55/154, Nat. Coal Board Production Dept., publ. in TIME, v. 114, pt. 7, 1955, 17 pp.

5. Bild, A. Pressure Drop During Pneumatic Conveying of Materials in Pipes. Transl. No. 2427, Nat. Coal Board Transl. A 2395 from Stojirenstvi, v. 13, No. 3, 1963, pp. 187-192.

6. Brewer, H. Stowing Under Improved Dustiness and Throughput Conditions at Reduced Operating Cost. Transl. No. 1900, Nat. Coal Board Transl. A 215/FWH, from Glukhauf, year 98, No. 10, 1962, pp. 541-552.

7. Chumachenko, O. M. Technical and Economical Aspects of Total Stowing. Transl. No. 3167, Nat. Coal Board, Transl. A2383/JG, from Ugol, year 38, No. 6, June 1963, pp. 9-11.

8. Crow, L. J., and D. Cummins. Methane in Gob. 1973 Internal project report, Spokane Mining Research Center, BuMines, 26 pp.; available for consultation at Spokane Mining Research Center, 315 E. Montgomery St., Spokane, Wash.

9. IIT Research Institute. Feasibility of Pneumatic Stowing for Ground Control in Coal Mines. Res. Rept. No. D6068, January 1972, 128 pp.; available for consultation at Bureau of Mines libraries in Spokane, Wash., Denver, Colo., Twin Cities, Minn., and Pittsburgh, Pa.

10. Hill, J. R. M., M. M. McDonald, and L. M. McNay. Support Performance of Hydraulic Backfill. BuMines RI 7850, 1974, 12 pp.

11. Kealy, C. D., D. R. Corson, and R. A. Busch. Pneumatic Stowing Evaluation Test. Internal project report, Spokane Mining Research Center, BuMines, October 1968, 4 pp.; available for consultation at Spokane Mining Research Center, 315 E. Montgomery St., Spokane, Wash.

12. Konroth, W. Pressure Losses in Transport Pipes of Pneumatic Conveying Systems. Transl. No. 1896, Nat. Coal Board Transl. A 1965/ME, from Fardern & Hebeni, No. 8, 1960, pp. 609-613.

13. Lundsidle, C. A., J. C. Hartley, and J. W. Buch. Anthracite Mechanical Mining Investigations. Progress Report 6: Preliminary Report of Brieden Pneumatic Packing Machine. BuMines RI 4978, 1953, 13 pp.

14. McNay, L. M., and D. R. Corson. Hydraulic Sandfill in Deep Metal Mines. BuMines IC 8663, 1975, 63 pp.

15. National Coal Board, Great Britain. Hardened Steel Pipes for Pneumatic Stowing. Inf. Bull. No. 58/201, 1958, 7 pp.

16. National Coal Board Production Department, Great Britain. Pneumatic Stowing From the Surface at Crookhill Colliery. Inf. Bull. No. 58/158, 1958, 3 pp.

17. Peter, G. Measurements Made on Pneumatic Stowing Machines for Determining the Relationship Between Stowing Efficiency, Length of Pipe, Quantity of Air and Pressure. Transl. 3780, Nat. Coal Board Transl. A. 261, from Glukhauf, year 88, v. 33/34, Aug. 16, 1952, pp. 807-819.

18. Peter, G., and H. Horturg. A Motor Driven Pneumatic Stowing Machine With Stowing Lines Operating on Motor Pressure. Transl. No. 1848, Nat. Coal Board Transl. A 2113/FWH, from Glukhauf, year 97, No. 15, July 19, 1961, pp. 88-893.

19. Reynolds, J. W. Pneumatic Backfilling With Crushed Rock at the Sullivan Mine. Ann. General Meeting of CIMM, Ottawa, Canada, Apr. 9-12, 1972, CIMM Bull., v. 65, No. 723, July 1972, pp. 31-36.

20. Rommler, E., J. Schmidt, and H. Mareyeu. Results of Tests of Pneumatic Conveying Carried Out With the Moller Pump at the Schwarze Pumps Combine, East Germany. Transl. No. 2567, Nat. Coal Board Transl. A. 2283/A1, from Freiberger Farschunshefle, No. A 243, June 1962, pp. 66-79.

21. Schuchart, P. Resistance Law for the Hydraulic and Pneumatic Transport of Solids in Straight Horizontal Pipes. Transl. No. 3744, SMRE Transl. No. 5849, 1970, 11 pp.

22. Soderberg, R. L. Study of Backpacking Techniques. 1970 Internal project report, Spokane Mining Research Center, BuMines, 14 pp.; available for consultation at Spokane Mining Research Center, 315 E. Montgomery St., Spokane, Wash.

24. Weber, M., and N. Schanki. Pneumatic and Hydraulic Conveying. Transl. No. 3750, Nat. Coal Board Transl. A. 269/J.C., from Aufferitunga-Technik, No. 10, October 1967, pp. 549-556.

25. Whaite, R. H. Anthracite Mechanical Mining Investigations. Second Testing of Brieden Pneumatic Packing Machine. BuMines RI 5273, 1956, 22 pp.

26. Wittwer, G. H. Experience With Fixed Stowing Machines in Working Several Strip or Semi-strip Seams. Transl. No. 3748, Nat. Coal Board Transl. A/1589/JG., from Glukhauf, year 94, No. 11/12, Mar. 15, 1958, pp. 396-403.

27. Zasaditch, B. I. Pneumatic Transport for Underground and Surface Conveying. Transl. No. 2528 by H. Wulff, August 1963, from Ugol, v. 37, No. 11, 1962, pp. 38-41.

☆U.S. GOVERNMENT PRINTING OFFICE: 1976-703-001/15

INT.-BU.OF MINES,PGH.,PA. 21774