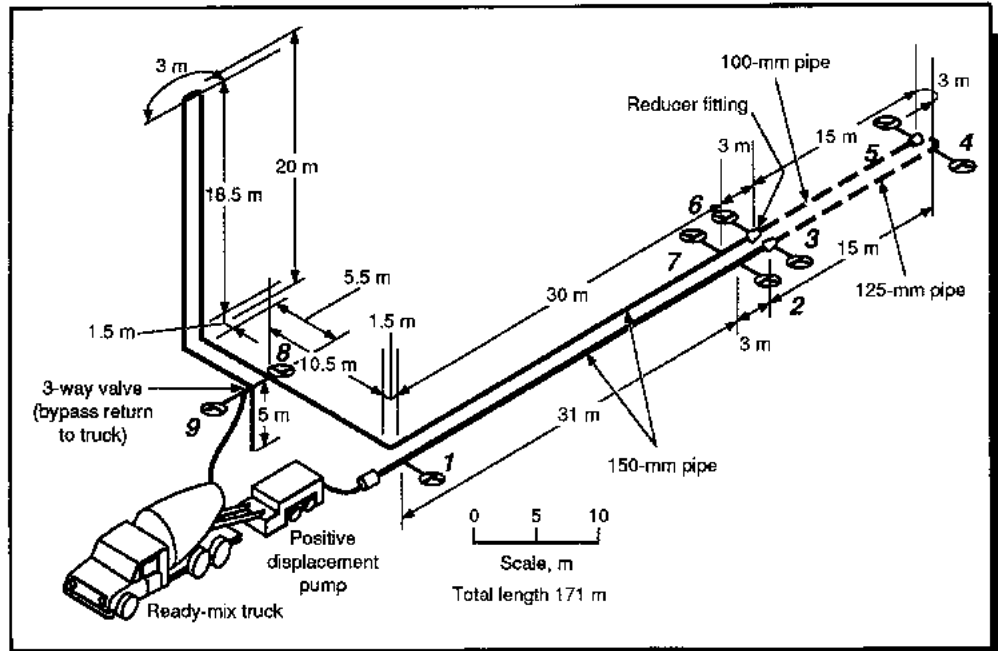


Transport of Total Tailings Paste Backfill: Results of Full-Scale Pipe Test Loop Pumping Tests

By C. C. Clark, J. D. Vickery, and R. R. Backer



UNITED STATES DEPARTMENT OF THE INTERIOR



UNITED STATES BUREAU OF MINES

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Cover: Total Tailings paste backfill full-scale pipe test loop.

Report of Investigations 9573

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Results of Full-Scale Pipe Test Loop
Pumping Tests**

By C. C. Clark, J. D. Vickery, and R. R. Backer

**UNITED STATES DEPARTMENT OF THE INTERIOR
Bruce Babbitt, Secretary**

**BUREAU OF MINES
Rhea Lydia Graham, Director**

International Standard Serial Number
ISSN 1066-5552

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UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

Metric Units

cm	centimeter		m ³ /h	cubic meter per hour
deg	degree		m ³ /min	cubic meter per minute
g	gram		min	minute
h	hour		mm	millimeter
kg	kilogram		m/m	meter per meter
kg/m ³	kilogram per cubic meter		MPa	megapascal
km	kilometer		m/s	meter per second
kPa	kilopascal		m/s ²	meter per second squared
kPa/m	kilopascal per meter		pct	percent
kW	kilowatt		t	metric ton
L	liter		t/d	metric ton per day
m	meter		t/h	metric ton per hour
m ³	cubic meter		°C	degree Celsius

U.S. Customary Units

ft	foot		lbf	pound of force
ft ³	cubic foot		lb/ft ³	pound per cubic foot
ft ⁴	foot to the 4th power		mi	mile
ft ³ /min	cubic foot per minute		psi	pound per square inch
ft/s	foot per second		psi/ft	pound per square inch per foot
gal	gallon		st	short ton
hp	horsepower		yd ³	cubic yard
in	inch		yd ³ /h	cubic yard per hour
lb	pound		°F	degree Fahrenheit

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

TRANSPORT OF TOTAL TAILINGS PASTE BACKFILL: RESULTS OF FULL-SCALE PIPE TEST LOOP PUMPING TESTS

By C. C. Clark,¹ J. D. Vickery,² and R. R. Backer³

ABSTRACT

This report presents the results of tests conducted by the U.S. Bureau of Mines to determine the transport characteristics of highly concentrated paste backfill mixes made from dewatered total mine tailings. The slurry concentrations of the mixes were all above 78 pct by weight with slumps ranging from 6.4 to 16.5 cm (2.5 to 6.5 in). Laboratory material property tests and full-scale pumping tests were conducted using pipe diameters of 102, 128, and 154 mm (4, 5, and 6 in) and a positive-displacement pump. The results indicated that the tested backfill mixes can be successfully transported as a stiff paste and identified the influence of and the interaction among the various mix and system variables.

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INTRODUCTION

ENGINEERED SUPPORT

The U.S. Bureau of Mines (USBM) has been active in mine waste management research for over 25 years. The goal of this research is to ensure that mine wastes are disposed of in a safe and environmentally benign manner. As part of its program in mine waste disposal technology, the USBM initiated projects to investigate the transport characteristics of high-concentration paste backfill formulated from total tailings. The results are described in this Report of Investigations (RI).

Many of the underground mines in the United States contain ore pillars that have been, and are being, left in place for mine stability. The use of engineered backfill having predictable structural properties may allow these valuable pillars to be recovered. Among the reported advantages of backfilling mined-out areas with engineered fill are the cost-effective use of mine waste, provision of uniform structural support, improvements in overall air quality, reductions in ventilation and refrigeration costs, and increased flexibility in mining technique because mining can take place above or below the fill (Cuerten, 1983; DeJongh and Morris, 1988; McKinstry and Laukkanen, 1989; Udd, 1989).

Underground mining methods that incorporate backfill as a primary support system are common in other parts of the world. Many of these mines use an engineered sandfill. While aiding in the support of mine structures, this type of fill requires special processing, transport, and placement techniques (Crandall, 1993; Kramers and others, 1989; Nicholson and Wayment, 1964). The sandfill is composed of partially classified tailings and water with added cement and is transported through boreholes and steel pipes using centrifugal pumps or gravity flow at slurry concentrations up to 75 pct by weight.

While simplifying transport, the use of large amounts of water causes many problems. The fill must be dewatered after it is placed so it will consolidate, a process which causes entrained fine material (fines)⁴ and cement to be flushed out along with the large volumes of excess water. This reduces the strength of the placed fill and deposits fines and cement in the lower workings, creating hazards for workers and increasing the need for maintenance of workings and equipment. These

⁴The terms coarse and fine are arbitrary and relative to the tailings grinds being compared. The percentage of material passing screen sizes of 0.074, 0.044, and 0.020 mm (0.0029, 0.0017, and 0.00146 in) is also used as a reference point, because materials of these sizes are missing from traditional classified sand-type tailings (Boldt and others, 1989; Brackebusch, 1992; Lidkea and Landriault, 1993; Vickery and Boldt, 1989). Material of minus 0.074 mm (200 mesh) is referred to as fines; material of minus 0.044 mm (325 mesh) is referred to as slimes.

factors limit the structural strength that can be obtained at low costs, since load-bearing capacity of the fill depends on cement content and void ratio (Landriault, 1987). There are also additional costs associated with pumping excess water to the surface; maintaining clogged bulkheads, ditches, and sumps; and repairing wear on pump components caused by the flushed cement.

In recent years, low-water-content, high-concentration paste backfills have been developed to reduce the problems associated with high-water-content slurry backfills. This type of fill provides better support and a safer working environment than does slurry sandfill because the excess water is eliminated, which allows greater strengths to be achieved and minimizes maintenance costs (Brackebusch, 1992).

The term "paste" refers to a class of backfills that have low water contents; high densities (>75 pct by weight); and consistency, transport, and deposition properties different from those of traditional low-concentration slurries or other types of high-concentration backfill (Aref and others, 1992; Putzmeister Thomsen, 1987, 1989a, 1989b; Verkerk and Marcus, 1988). Paste backfill has a homogenous appearance and a consistency that produces a measurable slump.⁵ The grain-size distribution of the backfill is such that, when placed, the fill is free standing and the contained water does not tend to be released. When transported through pipes, no minimum carrying velocity is required, and uncemented pastes can generally be remobilized if left stationary for many hours. The pressure gradients developed when high-concentration paste is transported are greater than those for low-concentration slurries.

The material used to make a paste fill is usually waste products from milling. Portland cement and other cementitious materials are commonly added to the backfill to increase support potential (Barrett, 1973; Thomas, 1973; Weaver and Luka, 1970). The advantages of using this type of fill are that it allows complete filling of stopes to the back, significantly increases strength gain rates for cemented fills, reduces void ratios in placed fills, lowers cement requirements for a given strength, and reduces the amount of fine material requiring surface disposal (Aref and Hassani, 1987; Robinsky, 1975; Verkerk and Marcus, 1988). Mining firms in South Africa, Germany, and Canada (Lerche and Renetzeder, 1984; Landriault and Goard, 1987; Verkerk, 1983) have reported success in developing new methods to batch, transport, and place paste backfill.

⁵Slump is a measure of the drop in height a material undergoes when it is released from a cone-shaped slip mold. It is more fully described in the section "Slump."

WASTE REDUCTION

The disposal of mill tailings has become a significant economic factor in domestic mine operations. Traditionally, the method of disposal was by impounding the waste material on the surface. Dams were constructed out of the coarse sand fraction of the mill tailings, and the fines were discharged behind the dams. Operating costs were low (Soderberg and Busch, 1977). Such impoundments were not only unsightly, but recent evaluations have indicated that many such structures could be unsafe during periods of heavy rain or during high winds, thus creating environmental and safety hazards as well as aesthetic liabilities to society (Khuntia and Pradhan, 1987; Vick, 1983). As a consequence, strict state and federal regulations now govern the construction and abandonment of tailings impoundments. The expense associated with compliance, in the form of permitting, construction of subsurface drainage systems and elaborate spillways, dust control, monitoring equipment, water treatment, etc., has had a major effect on the average cost of waste disposal using impoundments (Sauermaun, 1983; Verkerk and Marcus, 1988).

USE OF PASTE BACKFILL

Economic demands are forcing mines to recover as much ore as possible from underground workings. At the same time, mill operators are producing more finely ground tailings in an effort to extract greater percentages of minerals

from that ore. This puts greater demands on a mine's tailings disposal system and leaves less coarse sand available for low-concentration slurry backfill.

The result is that more attention is being paid to the development and use of highly concentrated paste backfill as a mine support system. This type of backfill is formulated from dewatered total tailings.⁶

The technology for transporting engineered high-concentration paste backfill made from tailings, additives, and water is just emerging in the United States (Boldt and others, 1989; Brackebusch, 1992; Crandall, 1993; Putzmeister Thomsen, 1989b; Vickery and Boldt, 1989). A principal requirement in designing a pipeline delivery system is determining the transport, or flow, properties of the material. These transport properties are a function of mix composition and system variables, some of which are relatively fixed because of the quantity of backfill needed and the desired load-bearing characteristics of the placed fill. It is, therefore, of great importance to be able to optimize flow conditions when given a particular set of variables (Gilchrist, 1988). This task is made difficult because of the wide variability in materials that can be used and the effect on transportability this variability produces. Thus, significant effort is required to achieve the desired goals. The development of techniques for logically determining how the transport characteristics of a particular paste backfill will change when mix and system variables are altered is needed if this technology is to be used by U.S. mining firms.

PASTE BACKFILL TRANSPORT TESTS

PIPE TEST LOOP

A fully instrumented pipe test loop (PTL) was constructed by personnel at the USBM's Spokane Research Center (SRC) to determine the transport characteristics of high-concentration paste backfill. The PTL is an instrumented, closed-circuit pipeline system powered by a diesel engine positive-displacement pump. The design and operation of the PTL are described in detail in appendix A. Test loops of this type provide essential engineering data (flow rate, pressure loss per unit length of pipe, shutdown and restart capabilities, and power consumption) needed to design full-scale pipelines. The USBM's PTL measures the gauge pressure associated with the flow of material through the pipeline at distinct points along its length.

TEST MIXES

Six high-concentration paste backfill mixes were developed and batched (table 1). The slump of the mixes was varied from 11.4 to 17.8 cm (4.5 to 7 in) with corresponding slurry

concentrations from 81 to 78 pct by weight. Two different mill grinds (figure 1) were used: four mixes were batched using coarse tailings and two using fine tailings. Descriptions of the tailings and physical property tests are found in appendix B. Tailings concentrations were varied from 94 to 100 pct of the solids weight. Cement was added in concentrations of 4 and 6 pct of the tailings weight. Strength tests on similar backfill mixes formulated from coarse and fine total tailings, cement, and water indicated that 28-day unconfined compressive strengths between 379 and 868 kPa (55 and 126 psi) can be obtained with these cement concentrations (figure 2) (Boldt and others, 1989, p. 13).

PUMPING TESTS

Pumping tests were conducted on the six backfill mixes using the PTL. Mixes 1 and 5 were tested using a 62-kW

⁶Total tailings include the full range of particle sizes from tailings produced by a mill, typically from 0.001 to 0.6 mm (0.00004 to 0.024 in).

Table 1.—Paste backfill mixes¹

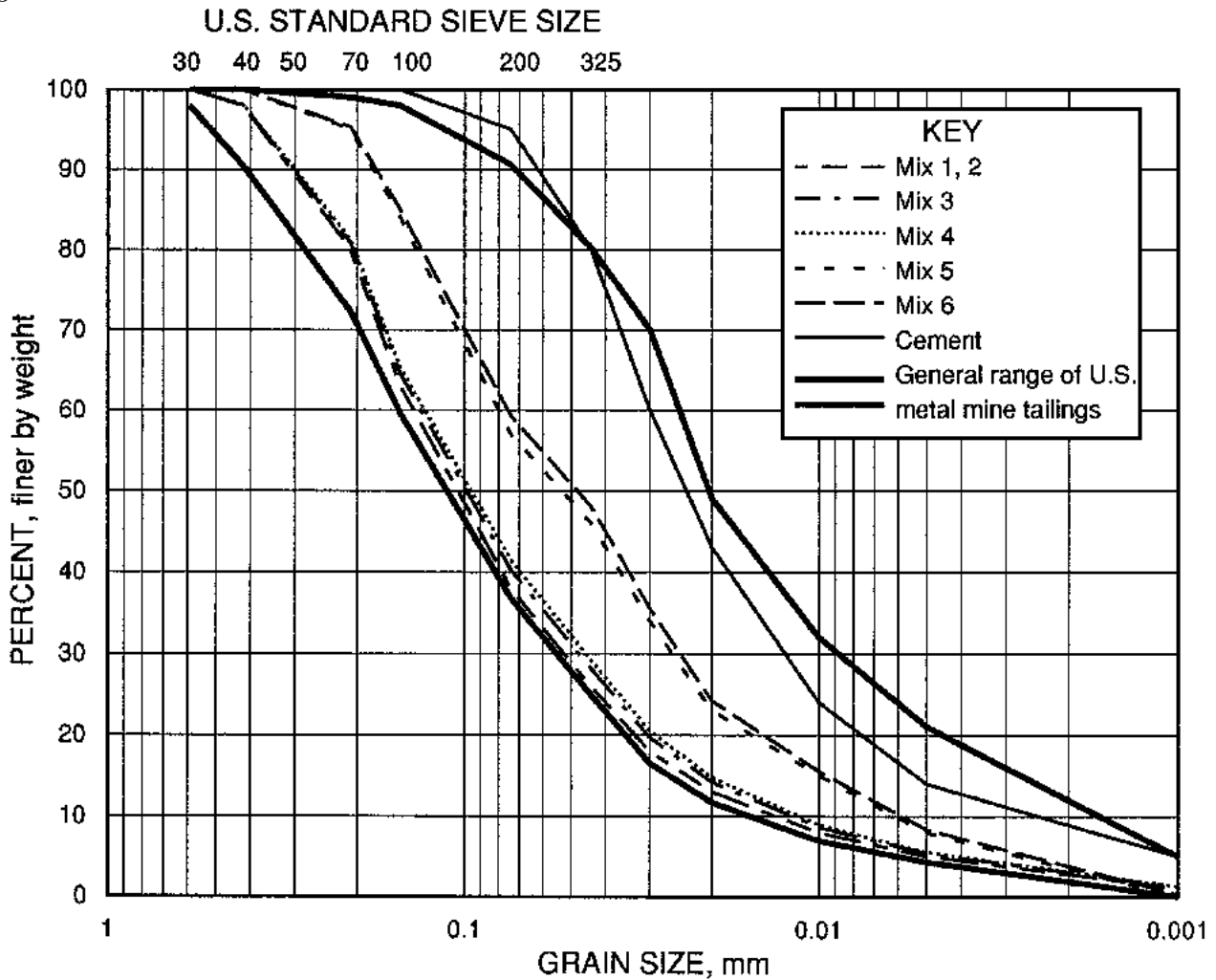
Mix	Tailings			Cement					C _t	G _s	Slump, cm		
	C _t	S _t	G _t	C _c	S _c	pct	G _c	w/c			Beg.	End ²	Diff.
Coarse tailings:													
1	80	100	2.8	0	0	0	3.15	0	81.3	2.06	11.4	NC	NC
2	80	100	2.8	0	0	0	3.15	0	80.7	2.06	16.5	NC	NC
3	77	96	2.8	3.2	3.8	4	3.15	6	79.6	2.06	17.8	15.4	-2.4
4	75	94	2.8	4.8	5.7	6	3.15	4	79.5	2.07	17.1	12.7	-4.4
Fine tailings:													
5	80	100	2.7	0	0	0	3.15	0	79.6	2.01	11.4	NC	NC
6	75	94	2.7	4.8	5.7	6	3.15	4	78.1	2.03	11.4	6.4	-5.0

C_t Concentration of tailings as a percentage of total slurry weight. C_t Concentration of solids as a percentage of total slurry weight.
 S_t Concentration of tailings as a percentage of dry solids weight. G_s Specific gravity of slurry.
 G_t Specific gravity of tailings. w/c Water-cement ratio.
 C_c Concentration of cement as a percentage of total slurry weight. Beg. Beginning.
 S_c Concentration of cement as a percentage of dry solids weight. Diff. Difference.
 G_c Specific gravity of cement. NC No change.

¹Initial mix batching was to 20 pct water, after which water was added to adjust mix to indicated slump.

²Slump was changed to these values by end of pumping test.

Figure 1

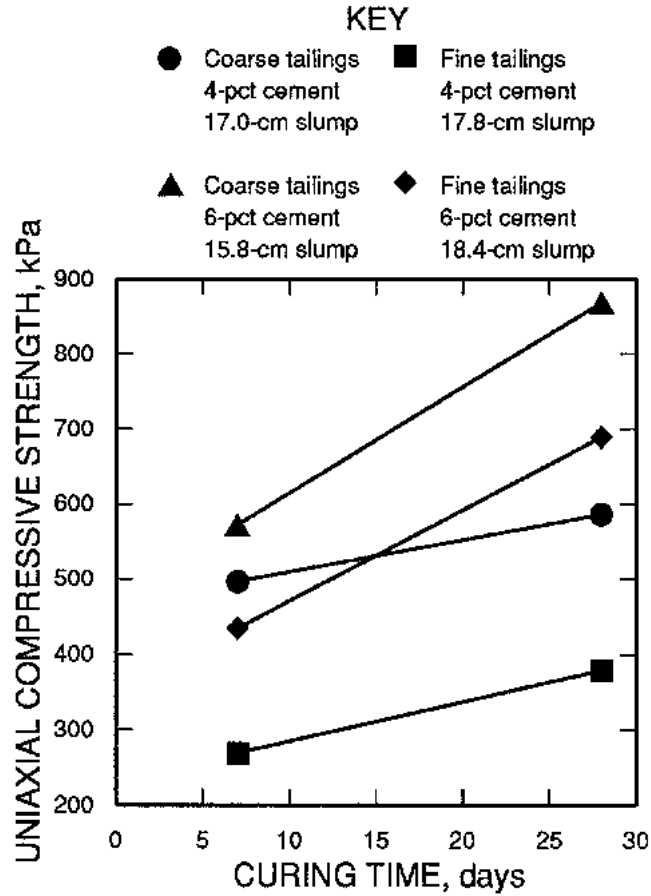


Particle-size distribution of uncemented and cemented total tailings.

(83-hp) pump and the pipe loop configuration shown in figure 3. Because of difficulties in obtaining sufficient pressure, the first pump was replaced with a more powerful 180-kW

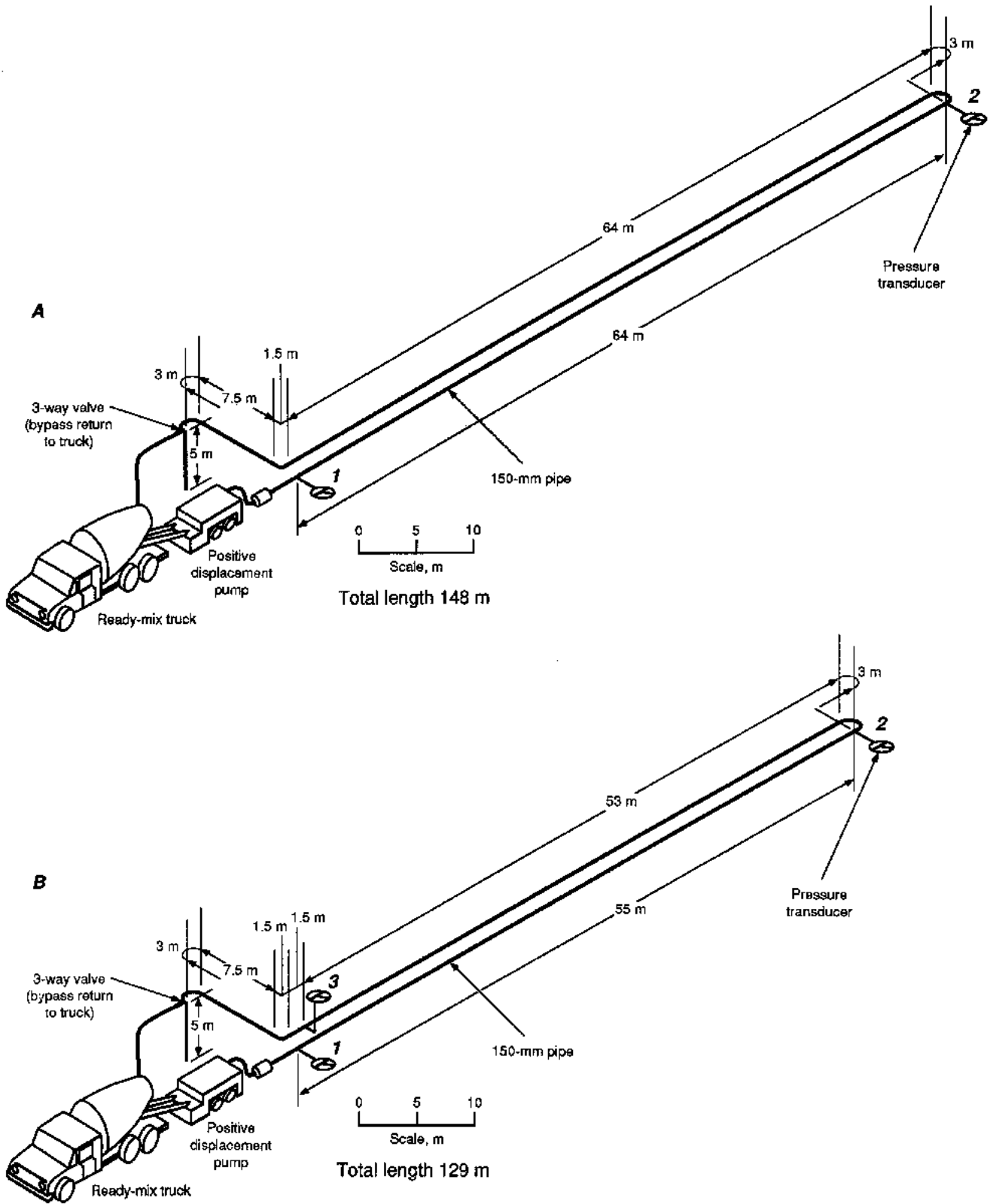
(240-hp) unit for the remaining tests. Mixes 2 and 6 were tested using the pipe loop shown in figure 3C, while mixes 3 and 4 were tested using the pipe loop shown in figure 3D.

Figure 2



Strength of coarse and fine cemented total tailings.

Figure 3



Pipe test loop configurations and pressure monitoring locations. A, Mix 5; and B, mix 1.

PASTE BACKFILL

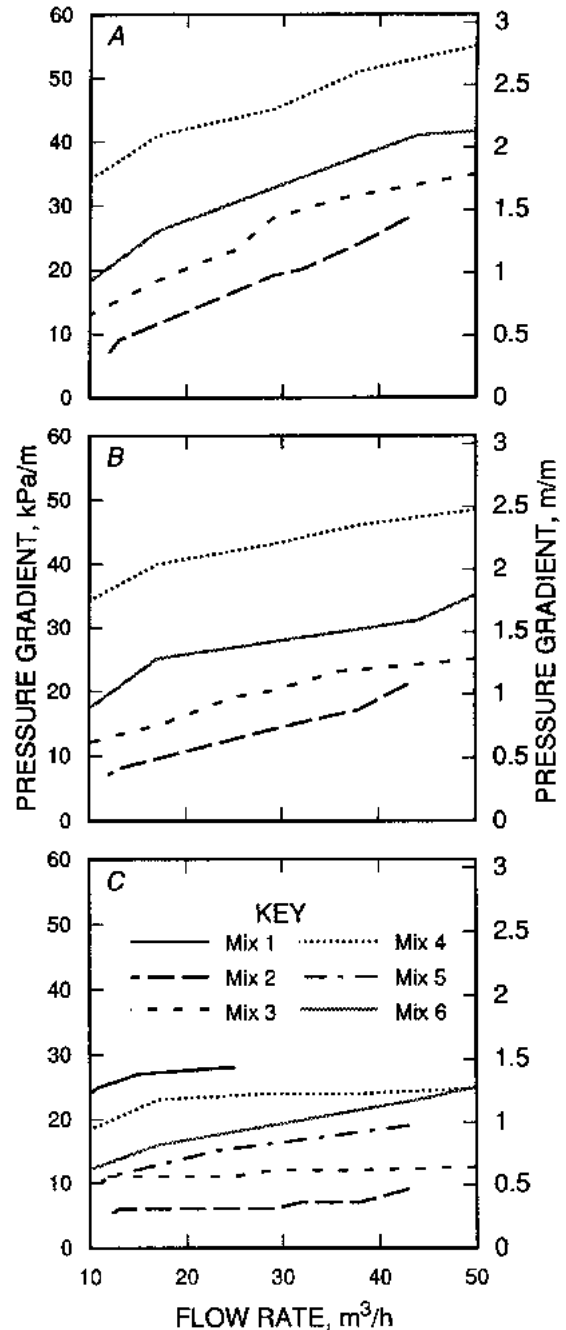
Pressure gradient was calculated to determine the transport characteristics resulting from different combinations of mix and system variables. The influence of each variable on transportability was determined by comparing the difference in pressure gradient among mixes that shared similar characteristics. Pressure gradient determines the system operating pressure required for pump specification and provides a good indication of how easily material will flow through pipes. The pressure gradient as a function of flow rate for each combination of mix and pipe size tested is shown in figure 4. A composite of the test results is shown in figure 5. The terms and equations used to reduce the pressure gradient data are described in appendix C.

SLUMP

Slump is a measure of the drop in height a material undergoes when it is released from a cone-shaped slip mold. Determination of slump provides a way of characterizing a material's consistency that can be related to transportability. Although pastes batched from dissimilar materials at the same slump will not flow in the same manner, comparisons can be made between mixes batched from similar materials at different slumps. A low-slump mix will flow less easily than a high-slump mix even if both are batched from the same material. A comparison of the pumping test results for mix 1 and mix 2 (figure 6) shows a substantial difference in pressure gradient associated with the 5-cm (2-in) change in slump. For a flow rate of approximately 25 m³/h (32 yd³/h), the pressure gradient changes from 28 to 6 kPa/m (1.2 to 0.3 psi/ft). This corresponds to a 78-pct decrease in pressure gradient for a 5-cm (2-in) increase in slump.

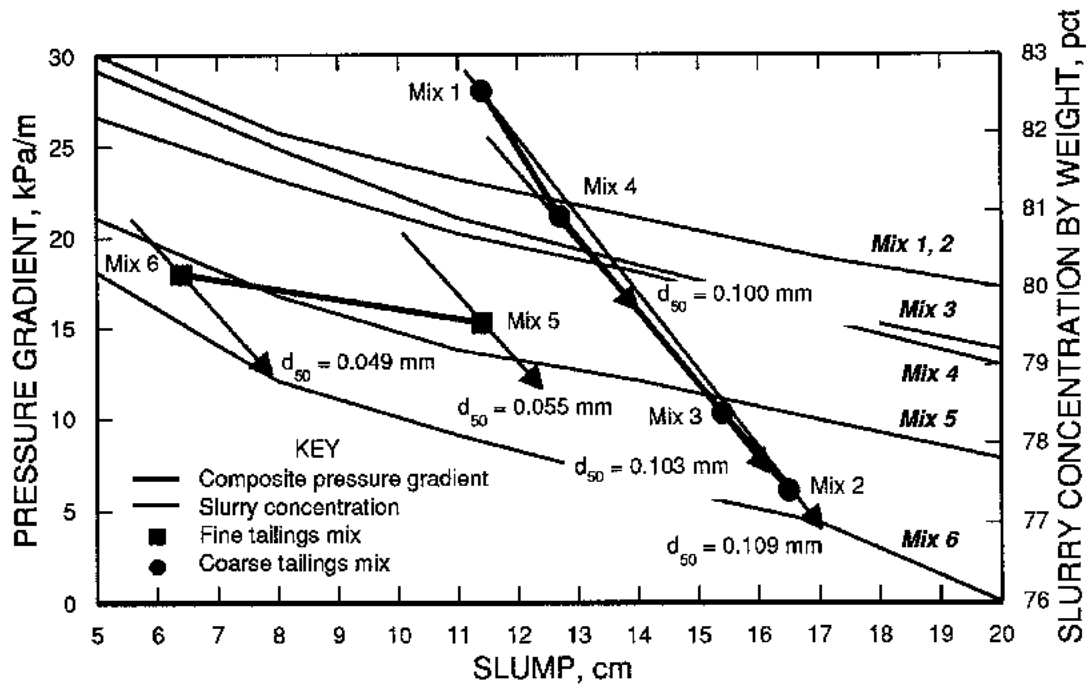
The effect on transportability produced by changes in mix variables can be determined by noting changes in slump. Changes in a specific mix will be manifested as changes in pressure gradient. This leads to the question as to whether the transportability of an untested mix can be determined from the material properties of its constituents and its slump without actually subjecting it to a pumping test. In short, if pressure gradient data produced from pumping tests exist for a particular mix, can the transportability of another untested mix be determined by comparing its proportion of these variables with the tested mix. The mixes are compared factor to factor in table 2. Note that in all cases, the mix with the smaller slump has

Figure 4



Pressure gradients for uncemented and cemented total tailings paste backfill. A, 102-mm-diam pipe; B, 128-mm-diam pipe; C, 154-mm-diam pipe.

Figure 5



Summary of test results for total tailings paste backfill at flow rate of $25 \text{ m}^3/\text{h}$ through 154-mm-diam pipe.

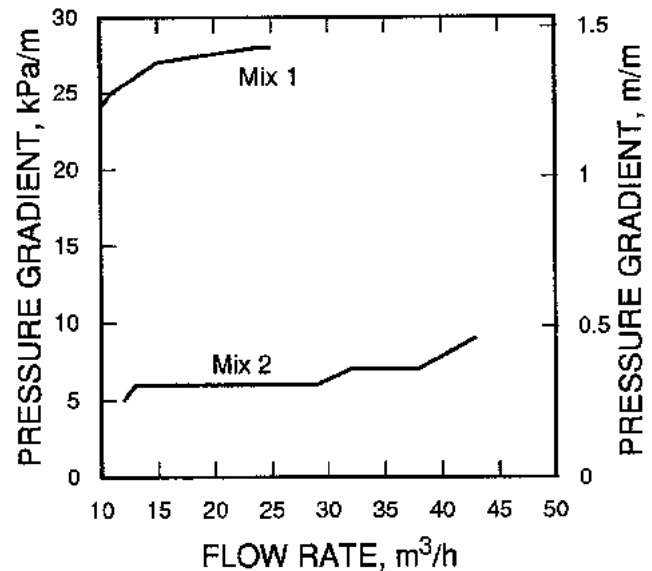
a greater pressure gradient—this is the most significant result of the investigation.⁷

SLURRY CONCENTRATION

Slurry concentration is the ratio of the weight of the solids in a mix to the weight of the total mix (water and solids). Slurry concentration is often used to compare the composition of mixes, particularly when batching. Although slurry concentration does not provide a direct indication of a material's consistency, in some cases it can be correlated to slump, which does.

⁷Although the following method was not used, it represents one of the most practical means of achieving the same mix consistency from batch to batch. Consistency can be measured by monitoring the electrical power used by a motor turning the paddles of a mixer. Batching is accomplished by carefully weighing the dry mix constituents into a container. The mixer is started and water is added until the power required by the motor corresponds to the target power for the mix consistency desired (Brackebusch, 1992; Lidkea and Landriault, 1993). Using this arrangement requires only that slump be correlated to consistency and consistency be correlated to power. It is also possible to predict what pressure gradient a mix will produce based on power once a correlation has been established between slump and pressure loss (figure 5).

Figure 6



Pressure gradients for total tailings paste backfill at different slumps through 154-mm-diam pipe.

Table 2.—Comparison of test results for total tailings paste backfill¹

Variable	First mix	Second mix	Difference
COMPARISON 1			
Mix	1	2	2-1
Tailings grind	Coarse	Coarse	NAP
Slump, cm	11.4	<16.5	5.1
Slurry concentration	81.3	>80.5	-0.8
d ₅₀ , mm	0.109	0.109	0
dP/dL, kPa/m	28.1	>6.1	-22.0
COMPARISON 2			
Mix	2	4	4-2
Tailings grind	Coarse	Coarse	NAP
Slump, cm	16.5	>12.7	-3.8
Slurry concentration	80.5	>80.4	-0.1
d ₅₀ , mm	0.109	>0.100	-0.009
dP/dL, kPa/m	6.1	<23.4	17.3
COMPARISON 3			
Mix	5	6	6-5
Tailings grind	Fine	Fine	NAP
Slump, cm	11.4	>6.4	-5.0
Slurry concentration	79.2	<79.5	0.3
d ₅₀ , mm	0.055	>0.049	-0.006
dP/dL, kPa/m	15.3	<18.0	2.71
COMPARISON 4			
Mix	1	4	4-1
Tailings grind	Coarse	Coarse	NAP
Slump, cm	11.4	<12.7	1.3
Slurry concentration	81.3	>80.4	-0.9
d ₅₀ , mm	0.109	>0.100	-0.009
dP/dL, kPa/m	28.1	>23.4	-4.7
COMPARISON 5			
Mix	2	3	3-2
Tailings grind	Coarse	Coarse	NAP
Slump, cm	16.5	>15.4	-1.1
Slurry concentration	80.5	>80.0	-0.5
d ₅₀ , mm	0.109	>0.103	-0.006
dP/dL, kPa/m	6.1	<11.3	5.2
COMPARISON 6			
Mix	3	4	4-3
Tailings grind	Coarse	Coarse	NAP
Slump, cm	15.4	>12.7	-2.7
Slurry concentration	80.0	<80.4	-0.4
d ₅₀ , mm	0.103	>0.100	-0.003
dP/dL, kPa/m	11.3	<23.4	12.1
COMPARISON 7			
Mix	1	3	3-1
Tailings grind	Coarse	Coarse	NAP
Slump, cm	11.4	<15.4	4.0
Slurry concentration	81.3	>80.0	-1.3
d ₅₀ , mm	0.109	>0.103	-0.006
dP/dL, kPa/m	28.1	>11.3	-16.8
COMPARISON 8			
Mix	1	5	5-1
Tailings grind	Coarse	Fine	NAP
Slump, cm	11.4	11.4	0
Slurry concentration	81.3	>79.2	-2.1
d ₅₀ , mm	0.1	>0.055	-0.054
dP/dL, kPa/m	28.1	>15.3	-12.9

dP/dL Pressure gradient.

NAP Not applicable.

¹Flow rate for all comparisons was 25 m³/h through 154-mm-diam pipe.

Note.—Bolded numbers indicate the most significant factor for determining which mix will have a greater pressure gradient.

In these tests, slurry concentration was correlated to slump using a series of mixes in which water was added in small increments. Slumps corresponding to the slurry concentrations were plotted on a graph to produce relational curves in which slump decreased as slurry concentration increased (figure 7). It should be noted that since the specific gravity of a material contributes to slurry concentration, pastes with identical particle-size distributions but different specific gravities will have different slurry concentrations at the same slump.

It was difficult to correlate slump with a particular slurry concentration because large changes in slump (consistency) corresponded to very minor changes in slurry concentration (in the range of 78 to 81 pct by weight). A comparison of mix 3 at slurry concentrations of approximately 79 and 81 pct by weight (figure 8) shows a large difference in slump, from 27 to 13 cm (10.5 to 5 in). Obviously, at these two slurry concentrations, this mix has different transport properties. A comparison of mix 1 and mix 2 (table 2) shows that slumps for these two mixes correspond to slurry densities of 81.3 and 80.5 pct by weight, respectively, which represents a 45-pct increase in slump for a 1-pct decrease in slurry concentration. This characteristic made trying to correlate slurry concentration to pressure gradient impossible. For example, comparing pressure gradients for mix 1 and mix 2 with changes in slurry concentration shows a 78-pct reduction in pressure gradient for a 1-pct decrease in slurry concentration.

Using slurry concentration as a basis for mixing a batch was also very difficult in terms of producing a material

with a repeatable consistency and, therefore, consistent transport characteristics from batch to batch. When batching by weight, it was impossible to control slurry concentration within ± 1 pct. For these reasons, mixing by slurry concentration is not recommended.

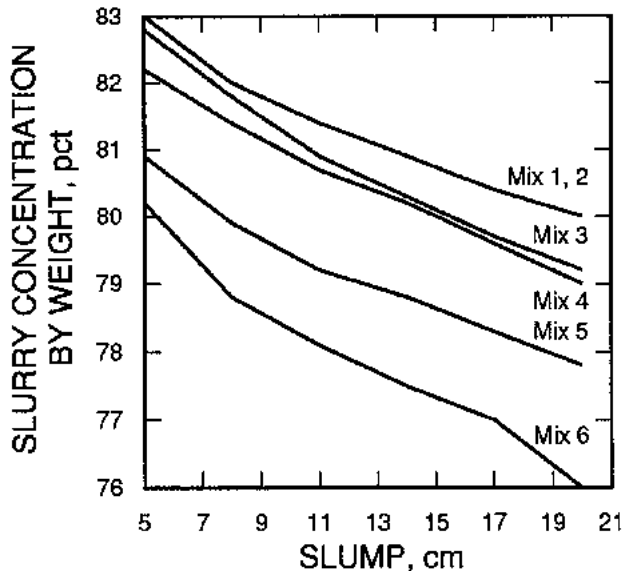
PARTICLE SIZE

The particle-size distribution curve shows the range of particle sizes present in a tailings sample and the type of

Figure 8



Figure 7



Slurry concentrations for coarse and fine total tailings paste backfill at different slumps.

Consistency of total tailings paste backfill at different slumps. A, 27-cm slump; B, 13-cm slump.

distribution of these particles. The diameter corresponding to 50 pct (d_{50}), finer by weight, in the particle-size distribution curve (figure 1) was used to identify differences among the tailings used. D_{50} is often used to characterize the particle-size distribution of tailings grinds and, as such, provides a reference point for comparing tailings (Sauermann, 1982; Verkerk and Marcus, 1988). The two tailings used in this investigation have a d_{50} of 0.109 (coarse tailings) and 0.055 mm (fine tailings) (140 and 270 mesh).

The relative proportions of fines and slimes present in the mixes have a significant influence on consistency. A threshold amount of fines and slimes, i.e., an amount that exceeds the pore volume of the large-sized particles, is required to present enough wetted surface area to trap the mix water in the paste matrix and prevent segregation of the particles and release of water, known as bleeding. It has been suggested that 15 pct, finer by weight, minus 0.020-mm (0.00079-in) material is sufficient to prevent bleeding (Brackebusch, 1992; Lidkea and Landriault, 1993). If sufficient amounts of fines and slimes are not present in the material, then water is released and a "sand-pack" forms in the pipeline if mix transport is stopped and restarted.

If there is an excess of fines and slimes in the mix in relation to the amount of water, then the increased wetted area presented by the fines and slimes leaves less free water, which then causes the formation of a more dense carrying medium with a higher resistance to flow (Verkerk and Marcus, 1988, p. 232).

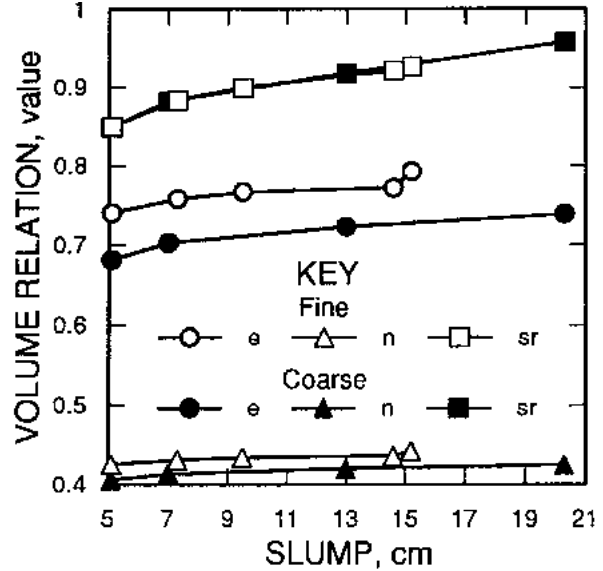
A comparison of the grain-size distribution for mix 1 and mix 5 (table 2) shows that while the mixes were batched to the same slump, differences in d_{50} particle size were substantial, i.e., diameters of 0.109 and 0.055 mm (140 and 270 mesh), respectively. The fine tailings [minus 0.074 mm (200 mesh) and minus 0.044 mm (325 mesh)], finer by weight, particle-size fractions are also greater than those of the coarse tailings. Because the finer material contains more particles per unit volume, there is more surface area to come into contact with water.

Comparisons of void ratio, porosity, and degree of saturation of the coarse and fine tailings mixes also show differences in surface area available for wetting. Fine tailings have a higher void ratio and porosity at nearly the same degree of saturation as coarse tailings. Thus, more water is required to wet the finer tailings, resulting in lower slurry concentrations for a given slump (figure 9). The degree of saturation indicates how much air is entrained in the mixes of lower slump. The air makes the mixes slightly compressible, which tends to dampen pressure surges (pressure surges are a consequence of the design of the positive-displacement pump). These surges were observed during pumping because the high-slump pastes caused tremendous hammering and resultant pressure spikes in the pipeline when the positive- displacement pump was used. Because of this, the use of twin-cylinder positive-displacement

pumps for pumping high-pressure, high-slump mixes is not recommended.

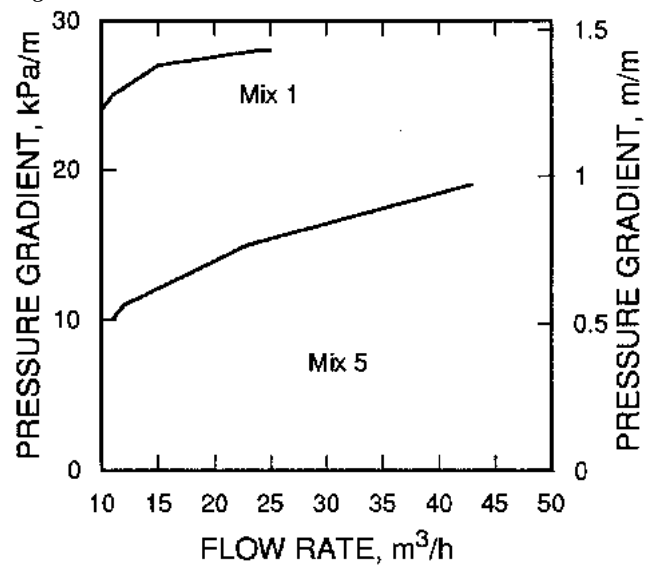
A comparison of the pumping test results for these mixes through 154-mm (6-in) diam pipe (figure 10) shows that although the mixes were batched to the same slump,

Figure 9



Volume relationships for coarse and fine total tailings at different slumps.

Figure 10



Pressure gradients for coarse and fine total tailings paste backfill through 154-mm-diam pipe.

there was a substantial difference in pressure gradient between the two mixes, even though the pressure gradients were very similar in slope and the relative difference in pressure gradient was constant. For a flow rate of approximately 25 m³/h (33 yd³/h), the pressure gradient changed from 28 to 15 kPa/m (1.2 to 0.7 psi/ft). This corresponded to a 46-pct reduction in pressure gradient as a result of reducing the d_{50} by half (figure 5). In terms of fines and slimes, a 46-pct reduction in pressure gradient results from increasing the quantity of minus 0.074-mm (200 mesh) material by 19 pct, finer by weight, and the quantity of minus 0.044-mm (325 mesh) material by 20 pct (figure 1).

During pumping tests, the mix material was observed to form a plug as it was discharged from the PTL into the pump inlet hopper. Inspection of the cross section of this plug revealed that fines and slimes in the mix had migrated toward the periphery of the plug during transport and had formed a thin layer of material at the interface between the pipe annulus and the plug. This layer of fine material appeared to lubricate the plug and could be responsible for the 50-pct reduction in pressure gradient. This phenomenon depended on there being a sufficient quantity of water in the mix to allow the fines and slimes to flow between the grains of the large-sized particles (Aref and others, 1992, pp. 272-273; Crandall, 1993, pp. 1776-1777).

The finer material was more transportable at a given slump. A comparison of the d_{50} lines for the mixes tested (figure 5) indicates that when well-graded total tailings mixes were batched by slump, the general trend was one of increasing pressure gradient with increasing d_{50} [the orientation of the lines was estimated using the two 0.109-mm (0.0043-in) data points for mixes 1 and 4].

ADDITION OF CEMENT

Cement addition or cement content is expressed as a percentage of tailings weight, as a percentage of solids weight, as a percentage of total slurry weight, or as a ratio of the weight of the water to the weight of the cement. In this investigation, cement was added as a percentage of tailings weight. The other expressions for cement content are included in table 1 for comparison.

The Type I-II portland cement used had a d_{50} diameter of 0.023 mm (minus 400 mesh) (figure 1). Although the particle size of this material is smaller in diameter than tailings particles, for the concentrations used to batch the mixes (4 and 6 pct of the tailings weight), the particle-size distribution was changed only slightly, as the percentage of fines and slimes increased and d_{50} decreased.

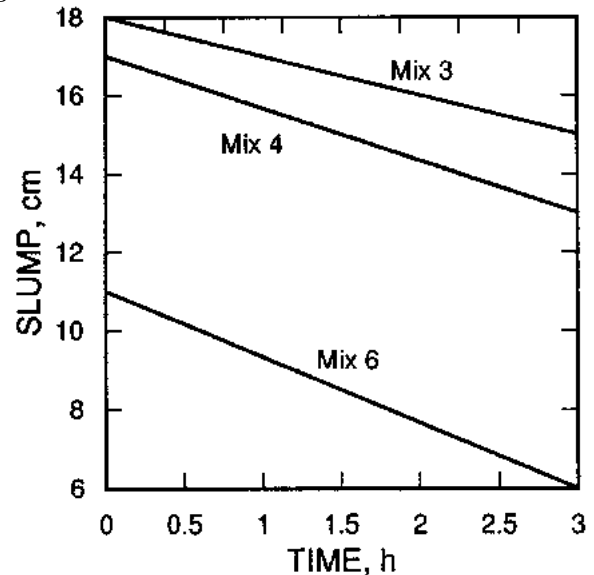
The slump of the cemented mixes decreased over the 3-h time required to complete the pumping tests (table 1).

A comparison of the reduction in slump over time for the mixes (figure 11) shows that the decrease in slump was related to the initial slump of the mix and the concentration of cement. Degree of agitation also influenced decreases in slump, but for the mixes tested, this variable was constant from mix to mix. Although slump can easily be increased by adding water, this practice is counterproductive to achieving high strengths with minimum cement consumption. Adding water can easily result in a mix in which the water-cement ratio is higher than optimal for strength. In these mixes, the cement becomes overhydrated, and cured strengths can be quite low for a given cement content.

The reduction in slump for cemented mixes is driven by the reaction between water and portland cement in the mix. When water is added to dry cement, an immediate and rapid reaction takes place, resulting in the formation of a supersaturated gel. The reaction then slows and a film of gel-like, microcrystalline calcium sulphoaluminate forms around the cement particles. This is followed by a period of slow reactions during which the amount of hydration products gradually builds up, forming loose bonds and slowly increasing the viscosity of the mix until there is a definite yield stress (a minimum shearing stress required to produce flow) (Lea, 1971, pp. 256-365).

The increase in viscosity results in a lower slump and, consequently, a higher pressure gradient. A comparison between the pressure gradients for the uncemented and cemented coarse and fine tailings mixes shows the effect

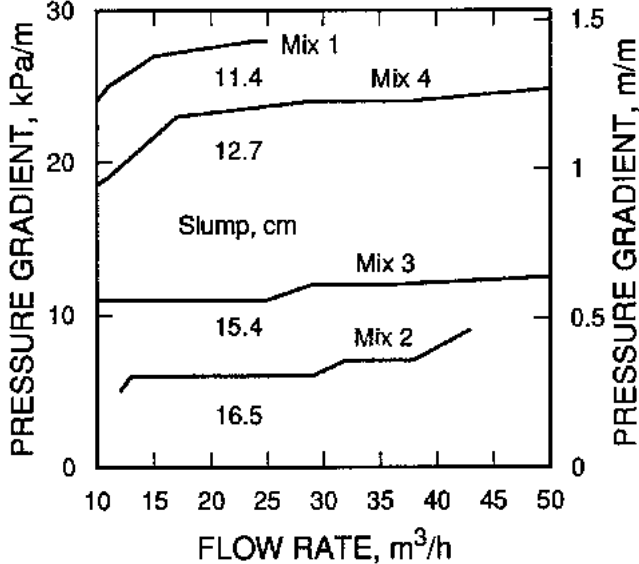
Figure 11



Decrease in slump over time for cemented total tailings paste backfill. Straight lines connecting data points were drawn for clarity and do not represent data points.

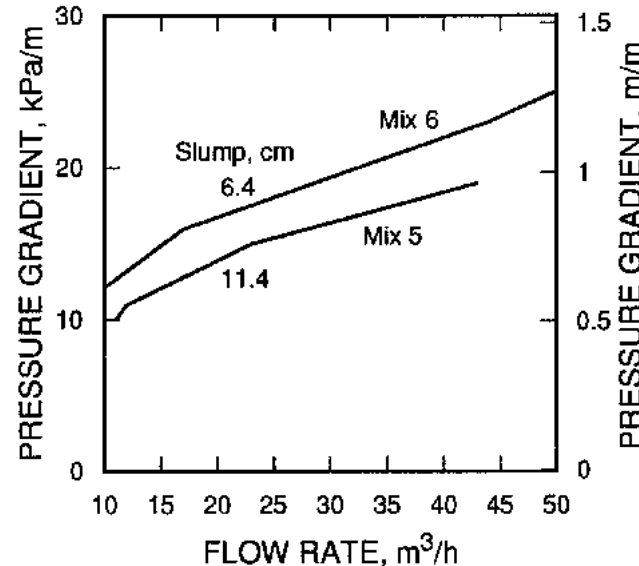
of this viscosity increase (figures 12 and 13). The differences in increases of pressure gradients for mixes 2, 3, and 4 and mixes 5 and 6 would suggest that incremental additions of cement were responsible. However, if the slump of the mixes at the time of testing is used as a basis for determining pressure gradient, the relationship between pressure gradient and

Figure 12



Pressure gradients for uncemented and cemented coarse total tailings paste backfill through 154-mm-diam pipe.

Figure 13



Pressure gradients for uncemented and cemented fine total tailings paste backfill through 154-mm-diam pipe.

slump developed previously is still valid. Note that uncemented mix 1 has a higher pressure gradient (and lower slump) than any of the cemented mixes, which confirms that pressure gradient increases as slump decreases regardless of the presence of cement.

The development of yield stress within the cemented mixes also influences transportability. When the bonds between the mix particles are strong enough to hold the mix together, but not strong enough to withstand small shearing forces, the paste exhibits a shear thinning behavior by becoming more fluid once mobilized. Shear thinning also implies that the structure of the paste recovers to its initial state if left standing, but considering the hydration process, this is not possible because as time passes, the material becomes stiffer and requires more pressure to overcome the yield stress and initiate flow.

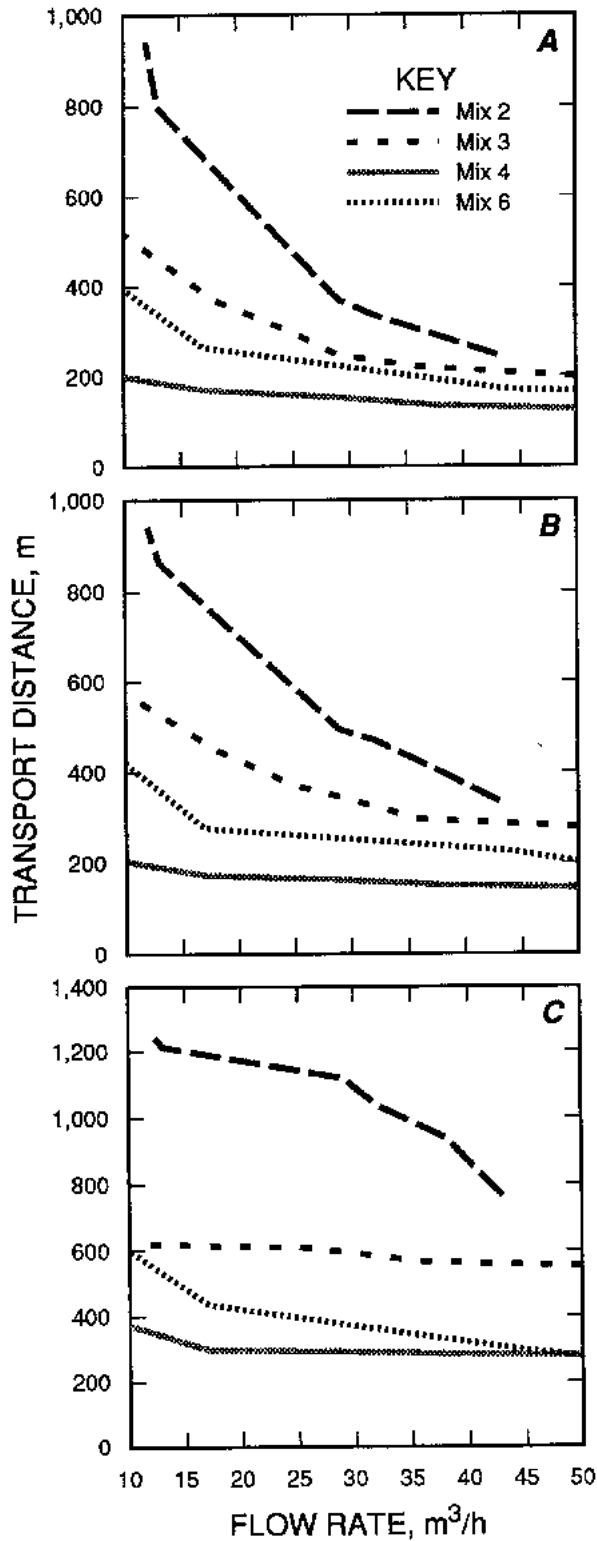
Because adding cement can cause slump to decrease with time and leads to the formation of yield stress within the mix, it may be necessary to batch the mix to the slump required at the time of transport and placement and to ensure that the batching and filling cycle be completed within a time frame dictated by the formation of structures within the material. If these time elements are known, then the transport characteristics of cemented mixes are the same as those for uncemented mixes with respect to slump, slurry concentration, and particle size.

These findings have important implications with respect to system design and testing, because the correlation between cement addition, decrease in slump over time, and formation of yield stress over time can be established in a laboratory. It becomes unnecessary to test cemented mixes to obtain flow properties; instead, flow properties can be determined using uncemented mixes with the proper slump applied. This not only reduces the amount of material necessary for testing (because the material can be used over and over again), but also eliminates disposal, cleanup, and machinery maintenance problems associated with cemented mixes.

PIPE DIAMETER

A comparison of pressure gradients through pipes of different diameters shows that increasing the pipe diameter can significantly lower pressure gradient for a given flow rate. Note as well that pressure gradients in different pipe diameters do not change linearly; that is, the rate of increase in pressure gradient as flow rate increases is greater in a 102-mm (4-in) pipe than in a 154-mm (6-in) pipe. A comparison of the maximum transport distances for the 102-, 128-, and 154-mm (4-, 5-, and 6-in) diam pipelines (figure 14) [using a pump outlet pressure of 6.9 MPa (1,000 psi) and the pressure gradients obtained for mixes 2, 3, 4, and 6 [mixes 1 and 5 were only tested in the 154-mm (6-in) diam pipeline] shows

Figure 14



Transport distance for total tailings paste backfill through pipelines of different diameters. Initial system pressure is 6.9 Mpa. A, 102-mm-diam pipe; B, 128-mm-diam pipe; C, 154-mm-diam pipe.

this relationship. Mix 2 had the lowest pressure gradient; a flow rate of 25 m³/h (33 yd³) through the 102-mm (4-in) diam line produced a maximum delivery distance of 476 m (1,567 ft). By increasing the line diameter to 154 mm (6 in), this distance increased to 1,141 m (3,743 ft).

FLOW RATE

A comparison of pressure gradient test results shows that pressure gradient increased with increasing flow rate for all the mixes tested. The rate of increase was greatest for flow rates below 15 to 20 m³/h (20 to 26 yd³/h), then became less as flow rate increased (figure 4). This pseudoplastic behavior is an important characteristic because it indicates that the flow rate of high-concentration paste mixes can be varied considerably without dramatically altering the pressure gradient. A comparison between pressure gradients for mix 6, a cemented total tailings paste backfill, and water at 25 °C (77 °F) through standard steel pipe (figure 15) can be used to generalize the marked contrast between this type of material and high-water-content, low-concentration slurries that behave like water. At low flow rates, the pressure gradient for water is also low. As flow rates increase, the rate of increase accelerates.

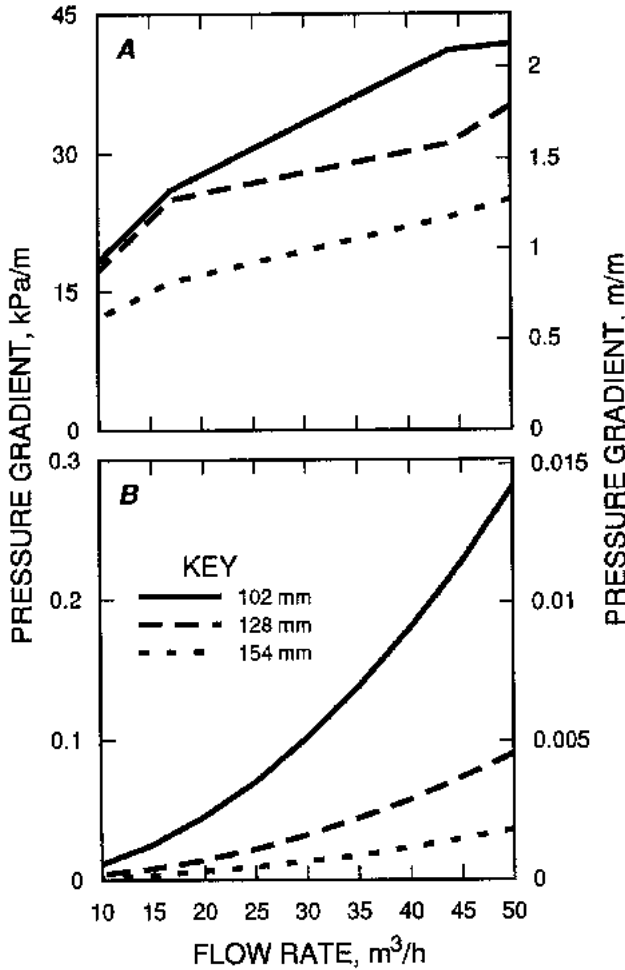
The noted pseudoplasticity of pastes in the tests could be partially explained by the possibility that the pump cylinders were not filling completely at higher pumping rates. If the cylinders are not completely filled, there will not necessarily be air entrained in the mix. A vacuum would be pulled when filling the piston, and the vacuum would collapse under positive pressure.

Changes in flow rate had the greatest influence on pressure gradient when flow rates were below 30 m³/h (39 yd³/h). The practical limit of flow rate for the mixes tested was a function of pressure gradients for the individual mixes and the line diameter. The maximum transport distance obtained, 1,257 m (4,124 ft), was for mix 2 through the 154-mm (6-in) diam pipeline at a flow rate of 12 m³/h (16 yd³/h).

VERTICAL FLOW

A comparison of pumping test results (figure 16) shows that the *net* pressure gradient in the combined vertical section of pipe (up plus down) was approximately equal to the pressure gradient in the horizontal section. Thus, other than the force of gravity, no additional forces were introduced in the vertical section. The pressure gradient decreased by an amount equal to the force of gravity exerted on the paste when the material was transported vertically downward. The gravity component of the pressure gradient was calculated to be between 20.1 and 20.6 kPa/m (0.88 and 0.91 psi/ft) depending on the unit

Figure 15



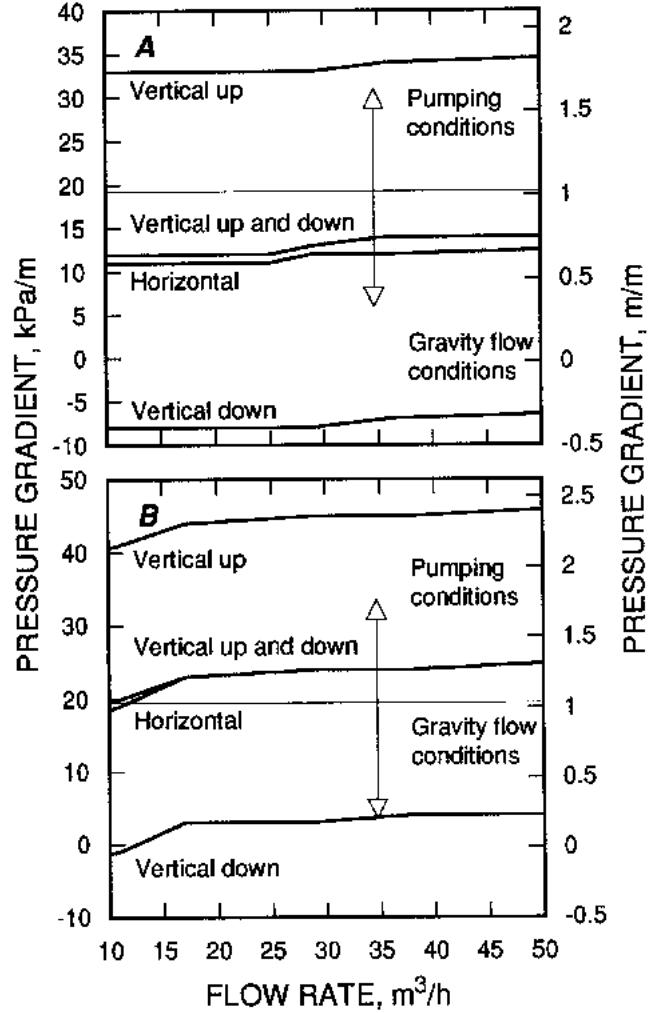
Pressure gradients through pipelines of different diameters. A, Water; B, mix 6.

weight of the mix. The frictional component of the pressure gradient for each mix remained the same regardless of the orientation of the pipe.

The significance of this finding is that, by using gravity to reduce the net pressure gradient for vertical flow, it becomes possible to deliver material horizontally with only the pressure generated by the standing column of material in the vertical section of the pipeline. The pressure gradients shown in figure 16 can be used to illustrate this concept.

In this figure, the line represents a pressure gradient of 1 m of slurry per 1 m of pipe. Pressure gradients below the line indicate that the mix will flow under its own weight; thus, the pressure generated by the standing column of material would be sufficient to deliver the material. Pressure gradients above the line indicate that the mix would have to be pumped. The

Figure 16

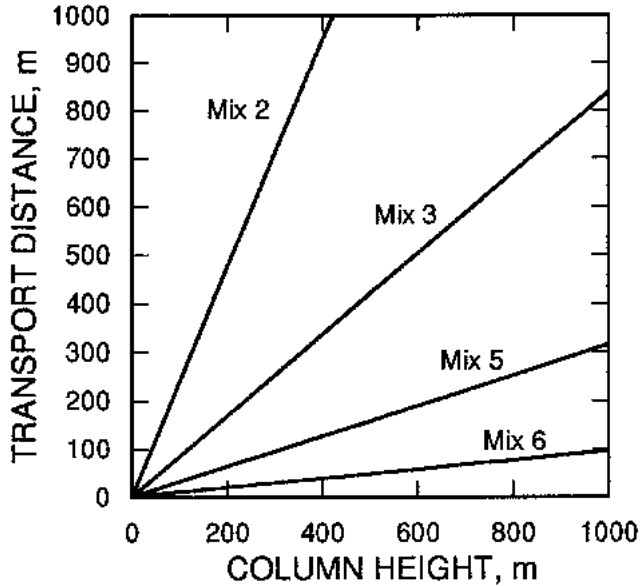


Pressure gradients for total tailings paste backfill through horizontal and vertical 154-mm-diam pipe. A, Mix 3; B, mix 4.

intercept of the line and the pressure gradients indicates the maximum flow rate that can be obtained by freefall in a vertical pipe.

Because the pressure at the base of the standing column of material available for transport is a function of the pressure imparted by gravity and the pressure lost through functional pressure gradient, it is possible to alter the distance the material can be transported horizontally by making changes either in the height of the standing column or in the diameter of the pipe. In figure 17, there is a tremendous difference in transport distance among the mixes at different column heights. Figure 18 shows pressure at the bottom of the standing column as a function of column height through different pipe diameters for mix 2. Note that a column height of 485 m (1,591 ft) in a 154-mm

Figure 17

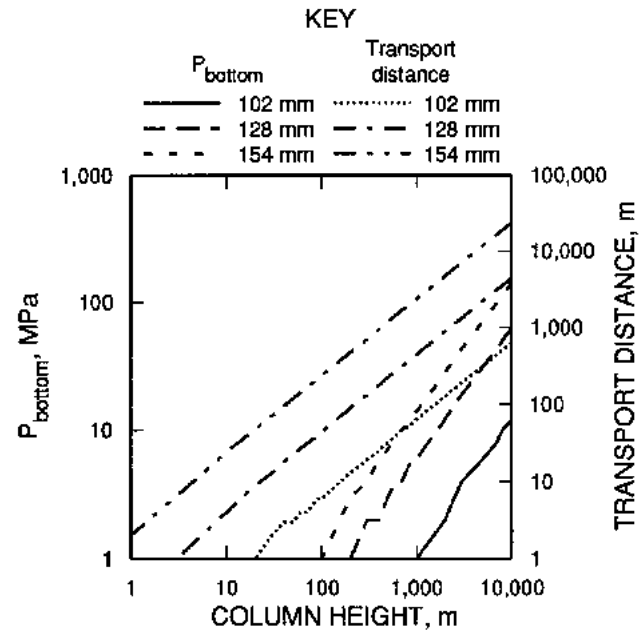


Transport distance for total tailings paste backfill at different column heights through 154-mm-diam pipe at flow rate of $25 \text{ m}^3/\text{h}$.

(6-in) diam pipe can produce the same amount of pressure [6.9 MPa (1,000 psi)] used in the previous examples for pump output pressure. The 128- and 102-mm (4- and 5-in) diam pipes require column heights of 1,110 and 5,700 m (3,642 and 18,700 ft), respectively, to produce the same pressure. Since the gravity pressure for a given column height is the same regardless of pipe diameter, the frictional difference between pipe diameters produces drastic differences in maximum transport distance. Using 300 m (984 ft) as the column height, the maximum transport distances for the 102- and 128-mm (4- and 5-in) diam pipes are 19 and 133 m (62 and 436 ft), respectively. By increasing the pipe diameter to 154 mm (6 in), a transport distance of 710 m (2,329 ft) is attainable.

Although the use of a vertical column to provide system pressure is well documented (Brackebusch, 1992; Crandall,

Figure 18



System pressure and transport distance for total tailings paste backfill at different column heights at flow rate of $25 \text{ m}^3/\text{h}$, mix 2.

1993, p. 1775; Lidkea and Landriault, 1993, pp. 337-347), there are several aspects unique to the type of system discussed here. (1) Care must be exercised in situations where the physical geometry of the mechanical system would allow the transported material to accelerate, which can occur when the line is first filled. In this situation, the impact on a pipe elbow at the bottom of the pipeline can be destructive. Such an impact can be alleviated by first filling the line with a less-dense material, such as water, and then pushing the water out of the line with the transported material. (2) If the feed to the vertical portion is intermittent, it is necessary to vent the top of the pipe to prevent generating a vacuum. The falling plug will actually be slowed and cushioned by the air trapped beneath it.

CONCLUSIONS

This investigation has focused on identifying the interaction among mix and system variables and the transportability of highly concentrated paste backfill. When pipe diameter, flow rate, and pipeline orientation are changed, the effects on transportability are predictable. The same cannot be said when changes are made in slurry concentration, particle size, and cement content. If the cumulative effects of all these

variables on pressure gradient are examined, a consistent relationship does emerge: That changes in mix variables affect slump and slump can be related to pressure gradient (figure 5). Thus, by examining slump, it is possible to determine the gross impact on pressure gradient that will be produced by changes in mix variables.

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APPENDIX A.—PIPE TEST LOOP

DESIGN

The choice of materials and equipment used to construct the PTL was based on what would typically be available to a commercial mining operation (Crandall, 1993; Lerche and Renetzeder, 1984; Putzmeister-Thomsen, 1987, 1989a, 1989b). The system was designed on the premise that a mine would need to fill a 300-m³ (392-yd³) void with dry solids within 7 h of an 8-h shift. Assuming slurry concentrations could vary from 75 to 85 pct by weight and the specific gravity of the solids could vary from 1.9 to 2.2, approximately 50 to 57 m³/h (65 to 75 yd³/h) of slurry would have to be transported and placed per shift (table A-1; figure A-1). Since paste backfill does not require a critical carrying velocity to remain in suspension, the velocity in the system pipeline was minimized to less than 2 m/s (6.6 ft/s) to reduce wear. The relationship between flow rate and velocity for selected standard steel pipe diameters is given in table A-2 and shown in figure A-2. Pipes with diameters of 102, 128, and 154 mm (4, 5, and 6 in) were selected on the basis of satisfactory combinations of adequate flow rate and low velocity. A list of the equipment used to construct the PTL is given in table A-3.

PUMP SELECTION

A trailer-mounted, positive-displacement concrete pump was used to pump the paste backfill mixture through the pipe loop circuit. The first pump used was powered by a 62-kW (83-hp) diesel engine with a maximum output of 50 m³/h

(65 yd³/h) and a maximum pressure of 4,826 kPa (700 psi). The pump consisted of a 0.40-m³ (0.52-yd³) receiving hopper, two 0.18 by 1 m (7 by 39 in) bore-and-stroke cylinders and pistons, and an S-shaped swing-tube valve. Flow rate was controlled by varying the cycling speed of the pump. A continuous flow of material through the delivery line was produced by sequencing the operation of the pump's two pistons with the swing tube. The swing tube allowed one cylinder to be open to the hopper on the suction stroke while the other cylinder discharged material through the swing tube outlet. At the end of piston travel, the direction changed and the swing-tube shifted to discharge material from the second cylinder (figure A-3) (Putzmeister-Thomsen, 1987, 1989b).

After two pump tests were completed, it was determined that the power output of the pump could not meet the desired pipe loop flow rate requirements, and the pump was replaced with a more powerful one. Although similar in design to the first pump, the second pump was powered by a 180-kW (240-hp) diesel engine. This pump had a maximum output of 74 m³/h (97 yd³/h) and a maximum pressure of 8,687 kPa (1,260 psi). The pump consisted of a 0.40-m³ (0.52-yd³) receiving hopper, two 0.18 by 2.1 m (7 by 83 in) bore-and-stroke cylinders and pistons, and an elephant-trunk-shaped swing-tube valve.

PIPE SYSTEM

The pipe loop used to test mix 5 consisted of 129 m (423 ft) of 154-mm (6-in) diam pipe arranged in a horizontal loop,

Table A-1.—Flow rates of total tailings paste backfill at different slurry concentrations

Slurry concentration, pct	Solids					Slurry			
	Specific gravity	m ³	m ³ /h	t/h	t/d	Specific gravity	m ³	m ³ /h	t/h
75	2.7	300	43	116	810	1.89	400	57	108
85	2.8	300	43	120	840	2.20	353	50	111

Table A-2.—Velocity through different pipe diameters at different flow rates, meters per second

Flow rate, m ³ /h	Pipe diameter, mm		
	102	128	154
50	1.70	1.08	0.75
51	1.73	1.10	0.76
52	1.77	1.12	0.78
53	1.80	1.14	0.79
54	1.84	1.17	0.81
55	1.87	1.19	0.82
56	1.90	1.21	0.84
57	1.94	1.23	0.85
58	1.97	1.25	0.86
59	2.01	1.27	0.88
60	2.04	1.30	0.89

Table A-3.—Equipment list

Item	Manufacturer	Model	Specifications
Concrete pump:			
	Thompson	TTS 2065	Trailer-mounted, positive-displacement pump powered by 62-kW (83-hp) diesel engine with a maximum output of 50 m ³ /h (65 yd ³ /h) and a maximum pressure of 4,826 kPa (700 psi). Pump consists of 0.40-m ³ (0.52-yd ³) receiving hopper, two 0.18 × 1 m (7 × 39 in) bore and stroke cylinders and pistons, and S-shaped swing-tube valve.
	Putzmeister	2100S	Trailer-mounted, positive-displacement pump powered by 180-kW (240-hp) diesel engine with a maximum output of 74 m ³ /h (97 yd ³ /h) and a maximum pressure of 8,687 kPa (1,260 psi). Pump consists of 0.40-m ³ (0.52-yd ³) receiving hopper, two 0.18 × 2.1 m (7 × 83 in) bore and stroke cylinders and pistons, and elephant-trunk-shaped swing-tube valve.
Pipeline:			
Sections	NA	Schedule 40	ASTM A-53 standard steel pipe.
Couplings	Victaulic	Style 77	Grooved end.
Elbows	Victaulic	Style 100	Grooved end, long radius, each elbow has 3-m (10-ft) straight pipe equivalent length. ¹
Recorder	Chessell	320	Six-channel strip chart.
Pressure transmitter:			
Mix 1	Bailey	KA14111-10k	0-4.13 MPa (0-600 psi), locations 1, 2.
Mix 2	Bristol-Babcock	2408-15B-621-110	0-5.17 MPa (0-750 psi), locations 3, 4.
	Bristol-Babcock	2408-15B-621-110	0-4.13 MPa (0-600 psi), locations 5, 6.
	Bailey	KA14111-10k	0-6.89 MPa (0-1,000 psi), location 7.
	Bailey	KA14111-10k	0-1.17 MPa (0-200 psi), location 9.
Mix 3, 4	Bailey	KA14111-10k	0-5.17 MPa (0-750 psi), locations 3, 4.
	Bristol-Babcock	2408-10B-611	0-4.13 MPa (0-600 psi), locations 5, 6.
	Bristol-Babcock	2408-10B-711	0-6.89 MPa (0-1,000 psi), location 7.
	Bailey	KA12111-10k	0-1.17 MPa (0-200 psi), location 8.
	Bristol-Babcock	2408-10B-711	0-6.89 MPa (0-1000 psi), location 9.
Mix 5	Bailey	KA14111-10k	0-4.13 MPa (0-600 psi), locations 1, 2.
	Bailey	KA121111-10k	0-1.38 MPa (0-200 psi), location 3.
Mix 6	Bristol-Babcock	2408-15B-621-110	0-5.17 MPa (0-750 psi), location 3.
	Bristol-Babcock	2408-15B-621-110	0-4.11 MPa (0-600 psi), location 4.
	Bristol-Babcock	2408-10B-611	0-3.45 MPa (0-500 psi), location 5.
	Bristol-Babcock	2408-15B-621-110	0-1.38 MPa (0-200 psi), location 6.
Pressure sensor, mixes 1-6	Hildebrandt	J-4	Diaphragm seal, locations-all.

¹Victaulic, 1981, pp. 19-22.

as shown in figure 3A. To test mix 1, the pipe loop length was increased to 148 m (486 ft) (figure 3B). To test mixes 6 and 2, the pipe loop length was reduced to 118 m (387 ft), and 102- and 128-mm (4- and 5-in) diam sections were added (figure 3C). To test mixes 4 and 3, the pipe loop length was increased to 172 m (564 ft); the loop was changed by adding a 20-m (66-ft) vertical section with 7-m (20-ft) horizontal legs of 154-mm (6-in) diam pipe to the existing 118 m (387 ft) of horizontal loop (figure 3D). This section was installed to evaluate the vertical flow characteristics of the mixes. A lateral pipe with removable cap was installed near the pump discharge point. This pipe was used to introduce water and air into the loop.

All of the test loop configurations were fabricated using 6-m (20-ft) lengths of grooved-end, standard steel pipe. The pipes were connected with grooved-end couplings and elbows with long radii (Victaulic, 1981). The pipe sections were assembled with tight connections to prevent line leaks, which can block

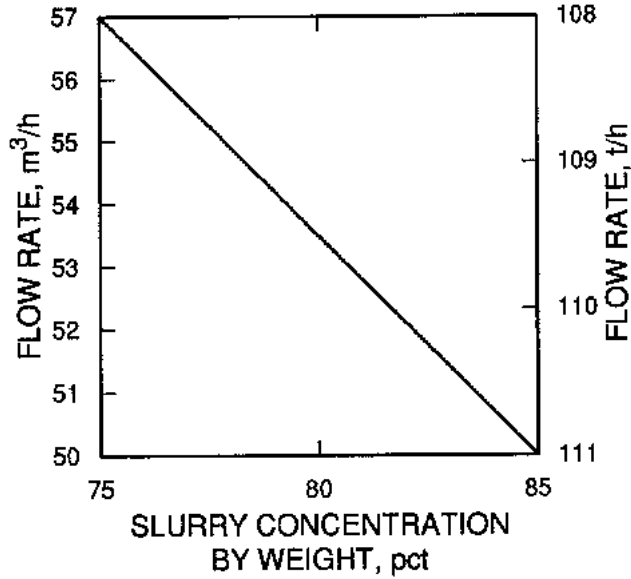
the line and cause extremely high wear. That is, a leak permits fines and water to pass through the mix matrix, abrading the area around the leak and leaving a coarse sand pack in the pipe.

MONITORING SYSTEM

The performance of the pipe test loop was monitored using two six-channel strip chart recorders. The chart recorder allowed the operator to determine visually when steady-state conditions were achieved after changing test parameters. Visual inspection was necessary because line pressures fluctuated very rapidly as a function of the twin-cylinder pump and were very difficult to capture with point readings unless the sampling rate was extremely high. And, if the sampling rate were increased, the large amount of data generated would make it extremely difficult to distinguish changes in system performance as a test was being run.

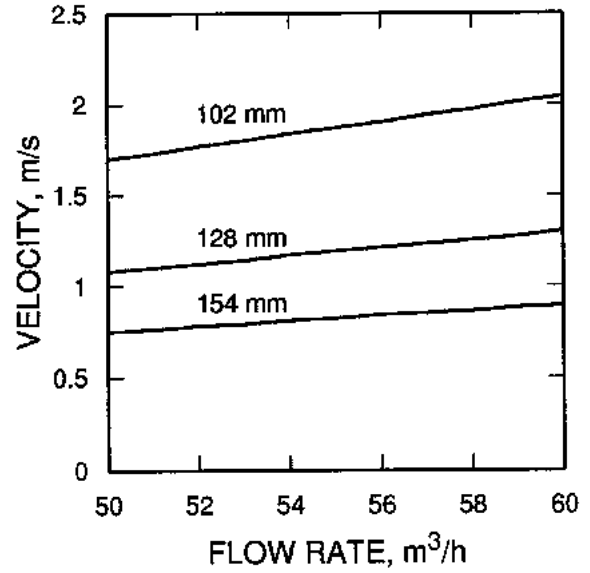
Strip chart recorders

Figure A-1



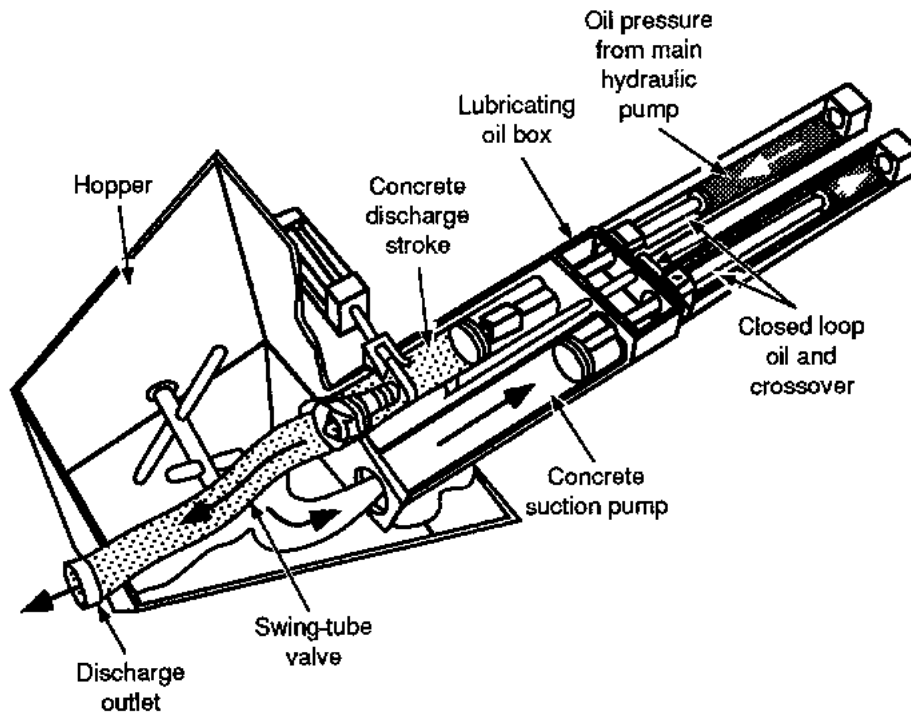
Flow rates for total tailings paste backfill at different slurry concentrations assuming placement of dry solids at $43 \text{ m}^3/\text{h}$.

Figure A-2



Flow velocities of water through different pipe diameters at different flow rates.

Figure A-3



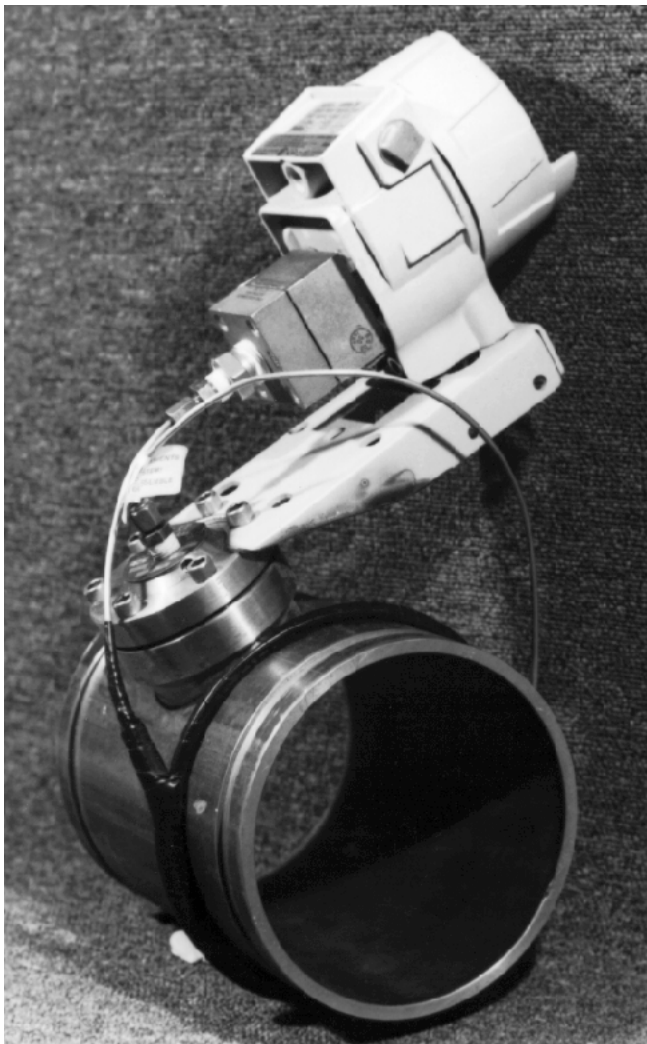
Positive-displacement pump mechanism.

were selected because they permit continuous real-time inspection of system performance, which is not possible with point-reading collection systems.

Pipeline gauge pressure was measured using electronic pressure transducers. The transducers were composed of pressure transmitters mounted to large oil-filled diaphragm seals of the type pictured in figure A-4. Figure A-5A shows a continuous pressure trace taken by the strip chart recorder. An analysis of a typical pressure pulse is shown in figure A-5B. Pressures were determined by taking the average peak pressure for each transducer. The cycling rate of the pump was determined by counting the spikes in the pressure readout from the strip chart.

Table A-4 describes the monitoring system setup on the pipe test loop for each mix tested.

Figure A-4



Electronic pressure transducer.

Table A-4.—Description of monitoring system on pipe test loop

Mix ¹	Length of pipe loop, ² m	Location of transducers (figure 3)	Distance between transducers, m	Additions
5 ...	129	1, 2	55	None.
		2, 3	56	None.
1 ...	148	1, 2	64	None.
6 ...	118	3, 4	15	128-mm section.
		5, 6	15	102-mm section.
		8, 9	12	Three elbows. ³
		3, 4	15	128-mm section.
2 ...	118	5, 6	15	102-mm section.
		7, 9	42	Three elbows. ⁴
		3, 4	15	128-mm section.
3 ..	172	5, 6	15	102-mm section.
		7, 8	42	154-mm-diam, 20-m-long vertical section; one elbow. ⁵
		8, 9	53	154-mm-diam, 20-m-long vertical section; four elbows. ⁶

¹Mixes are listed in order tested.

²All original pipe was 154 mm in diameter.

³Each elbow had a length equivalent to a 3-m-long straight section, for an adjusted length of 21 m between transducer locations 8 and 9.

⁴The adjusted length between transducer locations 7 and 9 was 51 m.

⁵The adjusted length between transducer locations 7 and 8 was 45 m.

⁶The adjusted length between transducer locations 8 and 9 was 65 m.

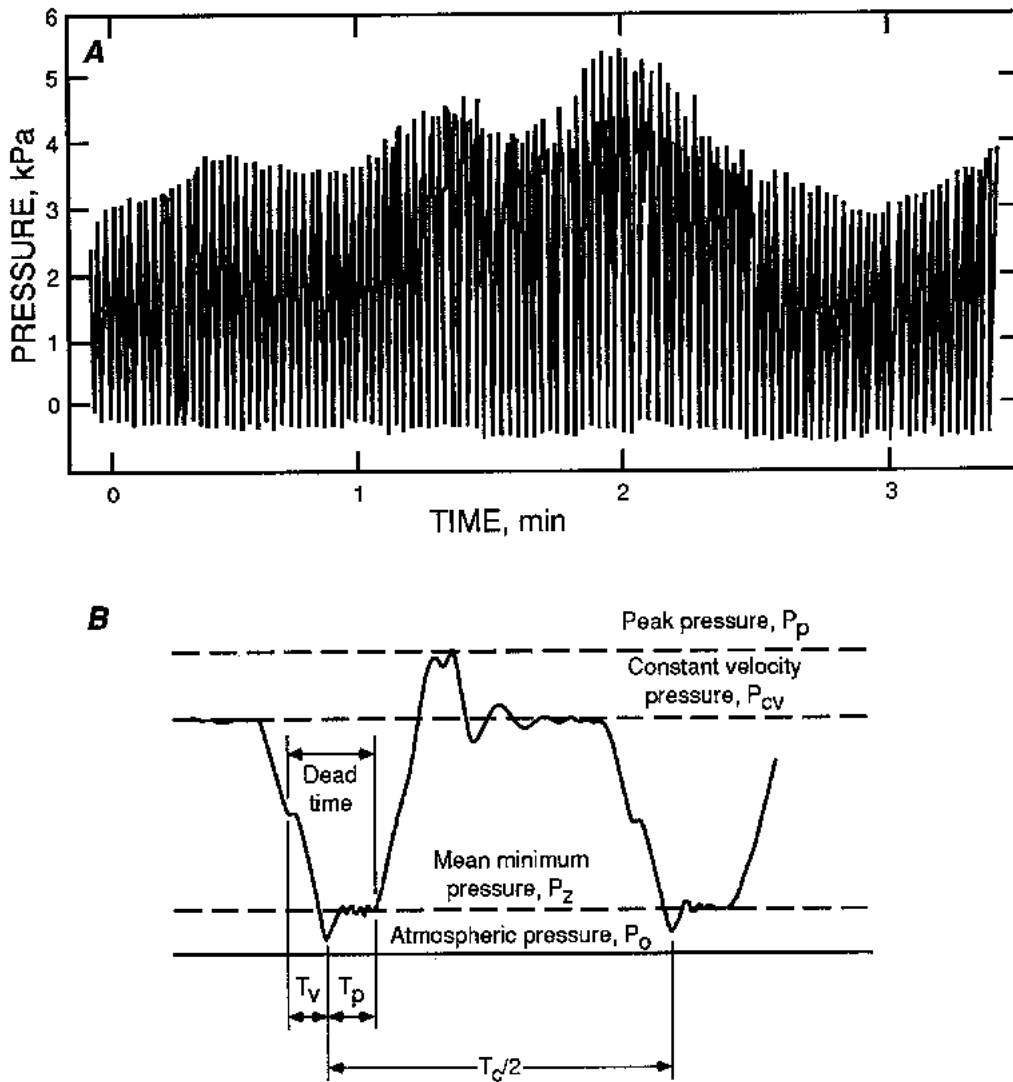
ADDITIONAL EQUIPMENT

Additional equipment for conducting the pumping tests included various drain and discharge valves; several 208-L (55-gal) barrels; a small 0.20-m³ (0.25-yd³) capacity front-end loader; a 1.5-t (1.65-st) capacity forklift with a barrel attachment; a 0-to-816-kg (0- to 1,800-lb) floor scale; a 25-m³/h (33-yd³/h) capacity loading conveyor; a 7.1-m³/min (250-ft³/min), 827-kPa (120-psi) air compressor; and a 7-m³ (9-yd³), high-discharge, ready-mix truck.

TEST PROCEDURE

The PTL required between 3 and 4 m³ (4 and 5 yd³) of paste mix for each pumping test. A ready-mix concrete truck was rented to mix and transport the paste mixes. Each test was started by filling the barrels with tailings using a small front-end loader. These barrels were then weighed. The water content of the bunker-stored tailings was used to determine the weight of the tailings. The tailings were transferred from the barrels to the loading conveyor with a forklift and loaded onto the concrete truck (figure A-6). Commercially available Type I-II portland cement was added in an amount equal to a percentage of the tailings weight and mixed in for several

Figure A-5



Data from strip chart recorder. A, Continuous pressure trace; B, analysis of pressure pulse.

minutes. Tap water was added until the mixture reached the target slump. The mixture was transported to the pumping facility 16 km (10 mi) away.

Before the truck arrived at the pumping facility, the pipe loop was filled with water, which was circulated through the loop for several minutes. A foam-rubber plug was inserted into the loop through the lateral pipe, and the compressor was connected to the cap using a 31.75-mm (1.25-in) diam air line with quick-connect end fittings. A three-way valve allowed water to be diverted from the main pipeline through an inclined pipe. Compressed air was used to force the plug and water around the test loop and push the water out the line and onto the ground. This water-flushing served several purposes: it served as a check for mechanical problems, it ensured that the

pipe loop was clear of debris, and it provided a lubricating film to prevent the test material from drying out when the pipe loop was loaded.

Once the truck arrived at the pump site, the mix slump was rechecked and adjusted by adding water until it reached the target consistency. This was necessary for all mixes because the slump would decrease during transport, which probably indicated that relatively dry tailings require a significant amount of time and agitation to become completely saturated. The transfer of the mix to the pump hopper was then begun (figure A-7). After the pump hopper was nearly filled with mix, the pump was started, mix was replenished using the truck discharge chute until the PTL was full, and paste was discharged on the return of the loop into the hopper.

Figure A-6

Conveyor system used to load dry total tailings into ready-mix truck.

The progress of the test material through the loop was monitored by tapping the pipe with a hammer and noting the sound. Blockages in the pipe could also be detected using this technique. The pump was run for several minutes to be certain that the paste was homogeneous. Large fluctuations in peak line pressure served as a good indicator that the paste was not of consistent thickness throughout the pipeline.

The transport tests consisted of pumping the mixes through the pipe loop at six flow rates while collecting pressure data from the transducers. Data were collected for 10 to 15 min at each flow rate to ensure that line pressures had stabilized.

The pipeline was emptied and cleaned at the end of each test. A three-way valve was used to divert material from the

Figure A-7

Transferring total tailings paste backfill from ready-mix truck to pump hopper.

main pipeline through an inclined pipe and back into the ready-mix truck to be transported back to SRC for disposal or reuse. The uncemented tailings were mixed with a larger quantity of unused material and reused on later tests of cemented tailings. Particle-size degradation was investigated and determined to be insignificant with the tailings used for testing. A foam-rubber ball was then inserted into the line through the lateral pipe. Compressed air was introduced to force the ball around the test loop and push the paste out the discharge section into the ready-mix truck. This process was repeated several times. Then, two balls with approximately 0.25 m (1 ft) of water trapped between them were inserted into the line and circulated to clean out the test loop.

APPENDIX B.—MATERIAL PROPERTIES

DESCRIPTION OF TAILINGS

The tailings used in the investigation were obtained from two underground metal mines with different mill grinds. The coarser tailings came from a silver mine and the finer tailings came from a copper-silver mine (figure 1). Bulk samples of the coarser tailings were obtained directly from the mine's batch plant storage facility, loaded into a dump truck, and transported to the USBM laboratory at SRC. These tailings had been slightly classified by the mine as part of its backfilling operation to facilitate handling by conventional means.

Bulk samples of the finer tailings (unclassified) were collected directly from the mine's tailings pond delivery line as a 40-pct-by-weight slurry. The barrels were then sealed and transported to SRC, where they were deposited in individual bunkers. In the bunkers, the tailings were spread and periodically mixed and respread to aid drying until they reached an as-stored moisture content of approximately 8.5 pct. These tailings were then used in the investigation.

PHYSICAL PROPERTY TESTS

Laboratory tests were conducted to determine the physical properties of the tailings and backfill mixes. Particle-size analyses were performed on the tailings according to American Society for Testing Materials (ASTM) standard D422-63 (1991b) using dry U.S. standard sieves for the particles above 0.074 mm (200 mesh). The minus 0.074 mm (minus 200 mesh) fraction was tested with a particle-size analyzer that operated on the principle of Stokes' law and utilized X-ray absorption.

The data from the particle-size analyses (table B-1) were combined with published data on particle sizes for cement and other tailings and plotted. Figure 1 and table B-1 show the particle-size distribution curves for the tailings, cement, and mix combinations. The particle-size distribution shows that these tailings fall within the range of grain sizes for most domestic metal mine tailings (Vick, 1983). Weight-volume relationships were also determined for the uncemented coarse and fine total tailings and water at different slumps (table B-2; figure 9). The fine tailings have greater void ratios and porosities than the coarse tailings at nearly the same degree of saturation.

The particle-size distribution curve for Type I-II portland cement shows that the cement is considerably finer ($d_{50} = 0.024$ mm) than either of the tailings. This is consistent with most present-day Type I-II portland cements, which average around 95 pct, finer by weight, than 0.088 mm (170 mesh) (Lea, 1971, p. 372). The effect of adding cement to either fine or coarse tailings is to decrease the d_{50} particle size. The compressive

strength values used to establish initial mix cement concentrations were obtained from published results of tests on laboratory-prepared 76.2- by 154-mm (3- by 6.06-in) samples of the same tailings material (table B-3; figure 2).

Table B-1.—Particle-size distribution of cemented and uncemented total tailings

Constituent	Cement, pct	d_{50} , mm	Particle size, pct, finer by weight	
			0.074 mm	0.044 mm
Coarse tailings:				
Lower limit of U.S. metal mine tailings.	NAP	0.117	38	24
Mix 1, 2		0.109	38	26
Mix 3	0	0.103	40.2	28.2
Mix 4	4	0.100	41.3	29.2
	6			
Fine tailings:				
Upper limit of U.S. metal mine tailings.	NAP	0.028	81	65
Mix 5		0.055	57	46
Mix 6	0	0.049	59.2	48
	6			
Portland cement, Type I-II.		0.024	94	80

NAP Not applicable.

Table B-2.—Volume relationships of total tailings paste backfill at different slumps

Volume relation, cm	Void ratio	Porosity	Degree of saturation
Mix 1, coarse tailings:			
5.1	0.682	0.406	0.849
8	0.703	0.413	0.883
13	0.723	0.420	0.919
20.3	0.739	0.425	0.958
Mix 5, fine tailings:			
5.1	0.740	0.425	0.850
7.3	0.756	0.431	0.884
9.5	0.767	0.434	0.900
14.6	0.772	0.436	0.922
15.2	0.793	0.442	0.927

The specific gravity of the tailings ranged from 2.7 to 2.8 (table 1). These values are typical for domestic metal mine tailings (Vick, 1983). The specific gravity of the cement used was 3.15 (Glover, 1992). The calculated specific gravity for the coarse tailings mixes ranged from 2.06 to 2.07 and for the fine tailings paste backfill mixes from 2.01 to 2.03.

Table B-3.—Average unconfined compressive strength of total tailings paste backfill, kilopascals

Cement, pct	Slump, cm	Compressive strength, kPa	
		7 day	28 day
Coarse tailings:			
4	19	496	586
6	16	572	868
Fine tailings:			
4	18	269	379
6	18	434	689

Slump was determined according to ASTM standard C143-90a (1991a) using a 31.75-cm (12.5-in) slump cone tester. The relationship between slump and slurry concentration was determined by preparing a series of mixes with slurry densities between 74 and 84 pct by weight. The slump corresponding to each slurry concentration was then measured. Specific gravity was determined according to ASTM standard D854-58 (1991c).

The relationship between slump and slurry concentration shows that large changes in slump corresponded to small

changes in slurry concentration (table B-4; figure 7). The slump of cemented paste backfill decreased over time as a function of initial slump and cement concentration (table 1; figure 11). Mix 6, with the lowest initial slump [11.4 cm (4.5 in)] and the highest cement concentration (6 pct), decreased in slump to 6.4 cm (2.5 in), which was the largest amount of decrease among the three mixes. Mix 4, with an initial slump of 17.1 cm (6.7 in) and 6-pct cement content, had the next largest decrease in slump, to 12.7 cm (5 in). Mix 3, with an initial slump of 17.8 cm (7.0 in) and 4-pct cement content, had the least amount of decrease in slump, to 15.4 cm (6.0 in).

Table B-4.—Slurry concentration of total tailings paste backfill at different slumps, percent

Mix	Cement, pct	Slump, cm					
		5	8	11	14	17	20
Coarse tailings:							
1, 2	0	83.0	82.0	81.4	80.9	80.4	80.0
3	4	82.8	81.8	80.9	80.3	79.7	79.2
4	6	82.2	81.4	80.7	80.2	79.6	79.0
Fine tailings:							
5	0	80.9	79.9	79.2	78.8	78.3	77.8
6	6	80.2	78.8	78.1	77.5	77.0	76.0

APPENDIX C.—TERMS AND EQUATIONS

WEIGHT-VOLUME

Mixes were batched using the percentage values from table 1 and a target volume of material. The weight of this volume of material was determined from the calculated specific gravity of the slurry. The weight-volume relationships used throughout this investigation were determined using the following equations and data from tables 1, C-1, C-2, C-3, and C-4. So that the text would be easier to follow, measurements are given in metric only.

Table C-1.—Water content of bunker-stored tailings

Sample	Weight, g		Water content, pct	Water concentration by weight, pct
	Sample	Oven-dried sample		
1	152.0	139.1	12.9	9.3
2	146.6	134.5	12.1	9.0
3	153.3	140.5	12.8	9.1
4	121.6	109.7	11.9	10.8
5	130.3	120.3	10.0	8.3
6	218.6	205.2	13.4	6.5
7	223.3	210.4	12.9	6.1
Average				8.5
				7.8

Concentration

Concentration as a percentage of the *total slurry weight*, C , is the ratio of the weight of constituent material to the weight of solids and water. Therefore,

$$C = \text{weight of constituent} \div (\text{weight of solids} + \text{weight of water}),$$

$$C_w = \text{concentration of water as a percentage of total slurry weight},$$

$$C_r = \text{concentration of solids as a percentage of total slurry weight},$$

$$C_t = \text{concentration of tailings as a percentage of total slurry weight},$$

and $C_c = \text{concentration of cement as a percentage of total slurry weight}.$

Concentration as a percentage of the *dry solids weight*, S , is the ratio of the weight of the additive to the weight of solids in the mix. Therefore,

$$S = \text{weight of additive} \div \text{weight of solids},$$

$$S_t = \text{concentration of tailings as a percentage of dry solids weight},$$

and $S_c = \text{concentration of cement as a percentage of dry solids weight}.$

Specific Gravity of Slurry

The specific gravity of slurry is the inverse sum of the concentration of water as a percentage of the total slurry weight and the ratio of the concentration of solids as a percentage of total slurry weight to the specific gravity of the solids. Therefore,

$$G_s = (C_w + C_s/G_r)^{-1},$$

where $G_s = \text{specific gravity of slurry},$

$$C_w = \text{concentration of water as a percentage of total slurry weight},$$

$$C_r = \text{concentration of solids as a percentage of total slurry weight},$$

and $G_r = \text{specific gravity of solids}.$

Specific Gravity of Solids

The specific gravity of solids is the sum of the products of the concentration of the tailings as a percentage of the dry weight of the solids multiplied by the specific gravity of the tailings plus the concentration of the cement as a percentage of the dry weight of the solids multiplied by the specific gravity of cement. Therefore,

$$G_r = (S_t \times G_t) + (S_c \times G_c),$$

where $G_r = \text{specific gravity of solids},$

$$S_t = \text{concentration of tailings as a percentage of dry solids weight},$$

$$G_t = \text{specific gravity of tailings},$$

$$S_c = \text{concentration of cement as a percentage of dry solids weight},$$

and $G_c = \text{specific gravity of cement}.$

Table C-2.—Weight-volume data for total tailings paste backfill

Sam- ple	Tailings		Cement		Water		Air		Total		Slurry		Slump,		Angle		Void		Poros- ity		Water		Concen- tration,		Degrees of sat.,		Sat. unit wt, m ³												
	kg	m ³	kg	m ³	kg	m ³	kg	m ³	kg	m ³	kg	m ³	kg	m ³	cm	R _i deg	ratio	pt	pt	pt	pt	pt	pt	pt	pt	pt	pt	pt											
MIXES 1 AND 2, COARSE, UNCEMENTED, SPECIFIC GRAVITY = 2.83																																							
1	13.6	0.0048	0.0	0.0	2.8	0.0028	0.00050	16.4	0.0081	14.3	2.025	5.1	78	0.68	0.41	0.20	0.83	0.85	2.086																				
2	13.6	0.0048	0.0	0.0	3.0	0.0030	0.00039	16.6	0.0082	14.3	2.025	7.0	72	0.70	0.41	0.22	0.82	0.88	2.073																				
3	13.6	0.0048	0.0	0.0	3.2	0.0032	0.00028	16.8	0.0083	14.3	2.026	13.0	65	0.72	0.42	0.23	0.81	0.92	2.060																				
4	13.6	0.0048	0.0	0.0	3.4	0.0034	0.00015	17.0	0.0084	14.4	2.032	20.3	50	0.74	0.42	0.25	0.80	0.96	2.050																				
MIX 3, COARSE, 4 pct CEMENT, SPECIFIC GRAVITY = 2.83																																							
1	13.2	0.0047	0.5	0.00017	2.8	0.0028	0.00049	16.5	0.0081	14.4	2.031	3.8	78	0.68	0.41	0.20	0.83	0.85	2.091																				
2	13.2	0.0047	0.5	0.00017	3.0	0.0030	0.00043	16.7	0.0083	14.3	2.021	7.3	72	0.71	0.42	0.22	0.82	0.87	2.073																				
3	13.2	0.0047	0.5	0.00017	3.2	0.0032	0.00032	16.9	0.0083	14.3	2.022	10.2	65	0.73	0.42	0.23	0.81	0.91	2.061																				
4	13.2	0.0047	0.5	0.00017	3.4	0.0034	0.00023	17.1	0.0085	14.3	2.018	15.6	50	0.76	0.43	0.25	0.80	0.94	2.045																				
5	13.2	0.0047	0.5	0.00017	3.6	0.0036	0.00021	17.3	0.0087	14.1	1.998	20.3	51	0.80	0.44	0.27	0.79	0.95	2.022																				
MIX 4, COARSE, 6 pct CEMENT, SPECIFIC GRAVITY = 2.83																																							
1	13.2	0.0047	0.8	0.00025	3.1	0.0031	0.00039	17.0	0.0084	14.4	2.035	6.4	78	0.70	0.41	0.22	0.82	0.89	2.081																				
2	13.2	0.0047	0.8	0.00025	3.3	0.0033	0.00031	17.2	0.0085	14.3	2.027	9.5	72	0.73	0.42	0.23	0.81	0.91	2.064																				
3	13.2	0.0047	0.8	0.00025	3.5	0.0035	0.00025	17.4	0.0086	14.3	2.017	15.6	65	0.76	0.43	0.25	0.80	0.93	2.046																				
4	13.2	0.0047	0.8	0.00025	3.7	0.0037	0.00024	17.6	0.0089	14.1	1.993	21.0	50	0.81	0.45	0.27	0.79	0.94	2.020																				
MIX 5, FINE, UNCEMENTED, SPECIFIC GRAVITY = 2.68																																							
1	13.6	0.0051	0.0	0.0	3.2	0.0032	0.00056	16.8	0.0088	13.5	1.900	5.1	78	0.74	0.43	0.23	0.81	0.85	1.964																				
2	13.6	0.0051	0.0	0.0	3.4	0.0034	0.00045	17.0	0.0089	13.5	1.904	7.3	72	0.76	0.43	0.25	0.80	0.88	1.954																				
3	13.6	0.0051	0.0	0.0	3.5	0.0035	0.00039	17.1	0.0090	13.5	1.906	9.5	65	0.77	0.43	0.26	0.80	0.90	1.949																				
4	13.6	0.0051	0.0	0.0	3.6	0.0036	0.00031	17.2	0.0090	13.5	1.912	14.6	50	0.77	0.44	0.27	0.79	0.92	1.946																				
5	13.6	0.0051	0.0	0.0	3.7	0.0037	0.00030	17.3	0.0091	13.5	1.903	15.2	51	0.79	0.44	0.27	0.78	0.93	1.935																				
MIX 6, FINE, 6 pct CEMENT, SPECIFIC GRAVITY = 2.68																																							
1	13.2	0.0049	0.8	0.00025	3.5	0.0035	0.00038	17.4	0.0090	13.7	1.929	5.7	78	0.75	0.43	0.25	0.80	0.90	1.971																				
2	13.2	0.0049	0.8	0.00025	3.7	0.0037	0.00032	17.6	0.0092	13.6	1.920	7.3	72	0.78	0.44	0.27	0.79	0.92	1.955																				
3	13.2	0.0049	0.8	0.00025	3.9	0.0039	0.00026	17.9	0.0094	13.5	1.909	11.4	65	0.81	0.45	0.28	0.78	0.94	1.937																				
4	13.2	0.0049	0.8	0.00025	4.2	0.0042	0.00018	18.1	0.0095	13.5	1.904	17.5	50	0.84	0.46	0.30	0.77	0.96	1.923																				
5	13.2	0.0049	0.8	0.00025	4.4	0.0044	0.00009	18.3	0.0097	13.4	1.899	21.3	51	0.87	0.47	0.32	0.76	0.98	1.908																				

Sat. Saturation.

Table C-3.—Results of pumping tests with horizontal sections only

D, mm	Stroke per min	Q, m ³ /h	v, m/s	Pressure, MPa		Length, m			In-out, No.	dP, kPa	Pressure gradient		Trans- port dist., m
				P _{in}	P _{out}	L _{in-out}	L _{fit}	L _{adj}			kPa/m	m/m	
MIX 1, COARSE, UNCEMENTED, SLUMP = 11.4 cm													
154	6.3	9	0.14	3.300	1.840	64.00	0.0	64.0	1-2	1,460	23	1.1	302
154	7.7	11	0.17	3.670	2.090	64.00	0.0	64.0	1-2	1,580	25	1.2	279
154	9.0	13	0.20	3.850	2.190	64.00	0.0	64.0	1-2	1,660	26	1.3	266
154	10.0	15	0.22	3.970	2.260	64.00	0.0	64.0	1-2	1,710	27	1.3	258
154	16.0	24	0.35	4.150	2.360	64.00	0.0	64.0	1-2	1,790	28	1.4	247
154	17.0	25	0.37	4.180	2.380	64.00	0.0	64.0	1-2	1,800	28	1.4	245
MIX 2, COARSE, UNCEMENTED, SLUMP = 16.5 cm													
102	3.7	12	0.39	0.500	0.390	15.00	0.0	15.0	5-6	110	7	0.4	941
102	4.3	13	0.45	0.530	0.400	15.00	0.0	15.0	5-6	130	9	0.4	796
102	9.2	29	0.97	0.720	0.440	15.00	0.0	15.0	5-6	280	19	0.9	370
102	10.3	32	1.09	0.767	0.460	15.00	0.0	15.0	5-6	307	20	1.0	337
102	12.1	38	1.27	0.890	0.530	15.00	0.0	15.0	5-6	360	24	1.2	287
102	14.0	43	1.46	0.970	0.550	15.00	0.0	15.0	5-6	420	28	1.4	246
128	3.7	12	0.25	0.650	0.540	15.00	0.0	15.0	3-4	110	7	0.4	941
128	4.3	13	0.29	0.670	0.550	15.00	0.0	15.0	3-4	120	8	0.4	863
128	9.2	29	0.62	1.000	0.790	15.00	0.0	15.0	3-4	210	14	0.7	493
128	10.3	32	0.69	1.010	0.790	15.00	0.0	15.0	3-4	220	15	0.7	470
128	12.1	38	0.81	1.170	0.910	15.00	0.0	15.0	3-4	260	17	0.9	398
128	14.0	43	0.94	1.300	0.990	15.00	0.0	15.0	3-4	310	21	1.0	334
154	3.7	12	0.17	0.370	0.090	42.00	9.0	51.0	7-9	280	5	0.3	1,257
154	4.3	13	0.20	0.390	0.100	42.00	9.0	51.0	7-9	290	6	0.3	1,213
154	9.2	29	0.43	0.420	0.105	42.00	9.0	51.0	7-9	315	6	0.3	1,117
154	10.3	32	0.48	0.450	0.110	42.00	9.0	51.0	7-9	340	7	0.3	1,036
154	12.1	38	0.56	0.490	0.115	42.00	9.0	51.0	7-9	375	7	0.4	938
154	14.0	43	0.65	0.580	0.120	42.00	9.0	51.0	7-9	460	9	0.4	765
MIX 3, COARSE, 4 pct CEMENT, SLUMP = 17.8 cm													
102	3.1	10	0.33	1.540	1.340	15.00	0.0	15.0	5-6	200	13	0.7	518
102	5.8	18	0.61	1.860	1.580	15.00	0.0	15.0	5-6	280	19	0.9	370
102	8.0	25	0.84	1.940	1.590	15.00	0.0	15.0	5-6	350	23	1.2	296
102	9.3	29	0.98	2.090	1.670	15.00	0.0	15.0	5-6	420	28	1.4	246
102	11.5	36	1.21	2.190	1.720	15.00	0.0	15.0	5-6	470	31	1.6	220
102	21.0	65	2.21	2.700	2.110	15.00	0.0	15.0	5-6	590	39	1.9	175
128	3.1	10	0.21	1.740	1.560	15.00	0.0	15.0	3-4	180	12	0.6	575
128	5.8	18	0.39	2.120	1.890	15.00	0.0	15.0	3-4	230	15	0.8	450
128	8.0	25	0.54	2.310	2.030	15.00	0.0	15.0	3-4	280	19	0.9	370
128	9.3	29	0.62	2.490	2.190	15.00	0.0	15.0	3-4	300	20	1.0	345
128	11.5	36	0.77	2.650	2.300	15.00	0.0	15.0	3-4	350	23	1.2	296
128	21.0	65	1.41	3.230	2.830	15.00	0.0	15.0	3-4	400	27	1.3	259
154	3.1	10	0.14	1.320	0.820	42.00	3.0	45.0	7-8	500	11	0.5	621
154	5.8	18	0.27	1.365	0.860	42.00	3.0	45.0	7-8	505	11	0.6	615
154	8.0	25	0.37	1.440	0.930	42.00	3.0	45.0	7-8	510	11	0.6	609
154	9.3	29	0.43	1.500	0.980	42.00	3.0	45.0	7-8	520	12	0.6	597
154	11.5	36	0.53	1.600	1.050	42.00	3.0	45.0	7-8	550	12	0.6	565
154	21.0	65	0.97	1.700	1.120	42.00	3.0	45.0	7-8	580	13	0.6	535
MIX 4, COARSE, 6 pct CEMENT, SLUMP = 17.1 cm													
102	1.7	5	0.18	2.420	1.950	15.00	0.0	15.0	5-6	470	31	1.5	220
102	3.6	11	0.38	2.840	2.310	15.00	0.0	15.0	5-6	530	35	1.7	195
102	5.5	17	0.59	3.430	2.820	15.00	0.0	15.0	5-6	610	41	2.0	170
102	9.3	29	0.98	3.570	2.890	15.00	0.0	15.0	5-6	680	45	2.2	152
102	12.3	38	1.30	3.720	2.950	15.00	0.0	15.0	5-6	770	51	2.5	134
102	17.0	53	1.79	3.860	3.020	15.00	0.0	15.0	5-6	840	56	2.8	123

See footnotes at end of table.

Table C-3.—Results of pumping tests with horizontal sections only—Continued

D, mm	Stroke per min	Q, m ³ /h	v, m/s	Pressure, MPa		Length, m			In-out, No.	dP, kPa	Pressure gradient		Trans- port dist., m
				P _{in}	P _{out}	L _{in-out}	L _{fit}	L _{adj}			kPa/m	m/m	
MIX 4, COARSE 6 pct CEMENT, SLUMP = 17.1 cm													
128	1.7	5	0.11	2.900	2.450	15.00	0.0	15.0	3-4	450	30	1.5	230
128	3.6	11	0.24	3.580	3.060	15.00	0.0	15.0	3-4	520	35	1.7	199
128	5.5	17	0.38	4.260	3.660	15.00	0.0	15.0	3-4	600	40	2.0	173
128	9.3	29	0.62	4.470	3.830	15.00	0.0	15.0	3-4	640	43	2.1	162
128	12.3	38	0.82	4.680	3.990	15.00	0.0	15.0	3-4	690	46	2.3	150
128	17.0	53	1.14	4.890	4.160	15.00	0.0	15.0	3-4	730	49	2.4	142
154	1.7	5	0.08	1.930	1.220	42.00	3.0	45.0	7-8	710	16	0.8	437
154	3.6	11	0.17	2.340	1.480	42.00	3.0	45.0	7-8	860	19	0.9	361
154	5.5	17	0.26	2.750	1.710	42.00	3.0	45.0	7-8	1,040	23	1.1	299
154	9.3	29	0.43	2.820	1.760	42.00	3.0	45.0	7-8	1,060	24	1.2	293
154	12.3	38	0.57	2.890	1.800	42.00	3.0	45.0	7-8	1,090	24	1.2	285
154	17.0	53	0.79	2.960	1.840	42.00	3.0	45.0	7-8	1,120	25	1.2	277
MIX 5, FINE, UNCEMENTED, SLUMP = 11.4 cm													
154	7.4	11	0.16	1.550	1.020	55.00	0.0	55.0	1-2	530	10	0.5	716
154	7.6	11	0.17	1.570	1.030	55.00	0.0	55.0	1-2	540	10	0.5	703
154	8.0	12	0.18	1.630	1.040	55.00	0.0	55.0	1-2	590	11	0.5	643
154	15.3	23	0.34	2.050	1.250	55.00	0.0	55.0	1-2	800	15	0.7	474
154	22.0	33	0.48	2.380	1.420	55.00	0.0	55.0	1-2	960	17	0.9	395
154	29.0	43	0.64	2.650	1.590	55.00	0.0	55.0	1-2	1,060	19	1.0	358
MIX 6, FINE, 6 pct CEMENT, SLUMP = 11.4 cm													
102	2.7	8	0.29	1.340	1.100	15.00	0.0	15.0	5-6	240	16	0.8	431
102	5.5	17	0.58	1.620	1.230	15.00	0.0	15.0	5-6	390	26	1.3	265
102	14.2	44	1.50	1.850	1.240	15.00	0.0	15.0	5-6	610	41	2.0	170
102	17.0	53	1.79	1.900	1.270	15.00	0.0	15.0	5-6	630	42	2.1	164
102	18.5	57	1.95	2.010	1.350	15.00	0.0	15.0	5-6	660	44	2.2	157
102	19.0	59	2.00	2.110	1.390	15.00	0.0	15.0	5-6	720	48	2.4	144
128	2.7	8	0.18	1.400	1.176	15.00	0.0	15.0	3-4	224	15	0.7	462
128	5.5	17	0.37	1.950	1.576	15.00	0.0	15.0	3-4	374	25	1.3	277
128	14.2	44	0.95	2.060	1.600	15.00	0.0	15.0	3-4	460	31	1.5	225
128	17.0	53	1.14	2.180	1.628	15.00	0.0	15.0	3-4	552	37	1.8	188
128	18.5	57	1.24	2.340	1.730	15.00	0.0	15.0	3-4	610	41	2.0	170
128	19.0	59	1.27	2.430	1.777	15.00	0.0	15.0	3-4	653	44	2.2	158
154	2.7	8	0.13	0.680	0.456	12.00	9.0	21.0	8-9	224	11	0.5	647
154	5.5	17	0.25	1.192	0.860	12.00	9.0	21.0	8-9	332	16	0.8	437
154	14.2	44	0.66	1.373	0.890	12.00	9.0	21.0	8-9	483	23	1.2	300
154	17.0	53	0.79	1.460	0.908	12.00	9.0	21.0	8-9	552	26	1.3	263
154	18.5	57	0.85	1.620	1.010	12.00	9.0	21.0	8-9	610	29	1.5	238
154	19.0	59	0.88	1.710	1.057	12.00	9.0	21.0	8-9	653	31	1.6	222

D Pipe diameter.

Q Flow rate.

v Velocity.

P_{in} Pressure at upstream transducer.P_{out} Pressure at downstream transducer.L_{in-out} Sum of length between upstream and downstream transducers.L_{fit} Equivalent length of added pipe.L_{adj} Adjusted length between upstream and downstream transducers.

dP/dL Pressure gradient.

Table C-4.—Results of pumping tests using 154-mm-diam pipe with vertical and horizontal sections

Stroke per min	Q, m ³ /h	v, m/s	Pressure gradi- ent, MPa		Length, m			In-out, No.	dP, kPa	dP/dL		Trans- port dist., m
			P _{in}	P _{out}	L _{in-out}	L _{tr}	L _{adj}			kPa /m	m/m	
MEASURED MIX 3, COARSE, 4 pct CEMENT, SLUMP = 17.8 cm												
Horizontal:												
3.1	10	0.14	1.320	0.820	42.00	3.0	45.0	7-8	500	11	0.5	1,765
5.8	18	0.27	1.365	0.860	42.00	3.0	45.0	7-8	505	11	0.6	1,747
8.0	25	0.37	1.440	0.930	42.00	3.0	45.0	7-8	510	11	0.6	1,730
9.3	29	0.43	1.500	0.980	42.00	3.0	45.0	7-8	520	12	0.6	1,697
11.5	36	0.53	1.600	1.050	42.00	3.0	45.0	7-8	550	12	0.6	1,604
21.0	65	0.97	1.700	1.120	42.00	3.0	45.0	7-8	580	13	0.6	1,521
Vertical up and down:												
3.1	10	0.14	0.820	0.033	53.00	12.0	65.0	8-9	787	12	0.6	1,619
5.8	18	0.27	0.860	0.060	53.00	12.0	65.0	8-9	800	12	0.6	1,593
8.0	25	0.37	0.930	0.120	53.00	12.0	65.0	8-9	810	12	0.6	1,573
9.3	29	0.43	0.980	0.150	53.00	12.0	65.0	8-9	830	13	0.6	1,536
11.5	36	0.53	1.050	0.170	53.00	12.0	65.0	8-9	880	14	0.7	1,448
21.0	65	0.97	1.120	0.190	53.00	12.0	65.0	8-9	930	14	0.7	1,370
CALCULATED MIX 3, COARSE, 4 pct CEMENT, SLUMP = 17.8 cm												
Vertical up:												
3.1	10	0.14	0.820	0.000	26.50	6.0	32.5	8-top	820	32	1.6	214
5.8	18	0.27	0.860	0.000	26.50	6.0	32.5	8-top	860	33	1.6	212
8.0	25	0.37	0.930	0.000	26.50	6.0	32.5	8-top	930	33	1.6	211
9.3	29	0.43	0.980	0.000	26.50	6.0	32.5	8-top	980	33	1.6	209
11.5	36	0.53	1.050	0.000	26.50	6.0	32.5	8-top	1,050	34	1.7	205
21.0	65	0.97	1.120	0.000	26.50	6.0	32.5	8-top	1,120	35	1.7	200
Vertical down:												
3.1	10	0.14	0.000	0.030	26.50	6.0	32.5	top-9	-30	-8	-0.4	1,820
5.8	18	0.27	0.000	0.060	26.50	6.0	32.5	top-9	-60	-8	-0.4	1,749
8.0	25	0.37	0.000	0.120	26.50	6.0	32.5	top-9	-120	-8	-0.4	1,732
9.3	29	0.43	0.000	0.150	26.50	6.0	32.5	top-9	-150	-7	-0.4	1,699
11.5	36	0.53	0.000	0.170	26.50	6.0	32.5	top-9	-170	-7	-0.3	1,606
21.0	65	0.97	0.000	0.190	26.50	6.0	32.5	top-9	-190	-6	-0.3	1,523
MEASURED MIX 4, COARSE, 6 pct CEMENT, SLUMP = 17.1 cm												
Horizontal:												
1.7	5	0.08	1.930	1.220	42.00	3.0	45.0	7-8	710	16	0.8	1,243
3.6	11	0.17	2.340	1.480	42.00	3.0	45.0	7-8	860	19	0.9	1,026
5.5	17	0.26	2.750	1.710	42.00	3.0	45.0	7-8	1,040	23	1.1	848
9.3	29	0.43	2.820	1.760	42.00	3.0	45.0	7-8	1,060	24	1.2	832
12.3	38	0.57	2.890	1.800	42.00	3.0	45.0	7-8	1,090	24	1.2	809
17.0	53	0.79	2.960	1.840	42.00	3.0	45.0	7-8	1,120	25	1.2	787
Vertical up and down:												
1.7	5	0.08	1.220	0.100	53.00	12.0	65.0	8-9	1,120	17	0.8	1,138
3.6	11	0.17	1.480	0.180	53.00	12.0	65.0	8-9	1,300	20	1.0	980
5.5	17	0.26	1.710	0.190	53.00	12.0	65.0	8-9	1,520	23	1.2	838
9.3	29	0.43	1.760	0.200	53.00	12.0	65.0	8-9	1,560	24	1.2	817
12.3	38	0.57	1.800	0.210	53.00	12.0	65.0	8-9	1,590	24	1.2	801
17.0	53	0.79	1.840	0.210	53.00	12.0	65.0	8-9	1,630	25	1.2	781
CALCULATED MIX 4, COARSE, 6 pct CEMENT, SLUMP = 17.1 cm												
Vertical up:												
1.7	5	0.08	1.220	0.000	26.50	6.0	32.5	8-top	1,220	38	1.8	184
3.6	11	0.17	1.480	0.000	26.50	6.0	32.5	8-top	1,480	40	2.0	171
5.5	17	0.26	1.710	0.000	26.50	6.0	32.5	8-top	1,710	44	2.2	158
9.3	29	0.43	1.760	0.000	26.50	6.0	32.5	8-top	1,760	44	2.2	156
12.3	38	0.57	1.800	0.000	26.50	6.0	32.5	8-top	1,800	45	2.2	154
17.0	53	0.79	1.840	0.000	26.50	6.0	32.5	8-top	1,840	45	2.2	152
Vertical down:												
1.7	5	0.08	0.000	0.100	26.50	6.0	32.5	top-9	-100	-3	-0.2	1,244
3.6	11	0.17	0.000	0.180	26.50	6.0	32.5	top-9	-180	-0	-0.0	1,027
5.5	17	0.26	0.000	0.190	26.50	6.0	32.5	top-9	-190	3	0.2	849
9.3	29	0.43	0.000	0.200	26.50	6.0	32.5	top-9	-200	4	0.2	833
12.3	38	0.57	0.000	0.210	26.50	6.0	32.5	top-9	-210	4	0.2	810
17.0	53	0.79	0.000	0.210	26.50	6.0	32.5	top-9	-210	5	0.2	788
D	Pipe diameter.		L _{in-out}		Sum of length between upstream and downstream transducers.							
Q	Flow rate.		L _{tr}		Equivalent length of added pipe.							
v	Velocity.		L _{adj}		Adjusted length between upstream and downstream transducers.							
P _{in}	Pressure at upstream transducer.		dP/dL		Pressure gradient.							
P _{out}	Pressure at downstream transducer.											

EXAMPLE

Test mix 3 is used as input for the following example, where $C_w = 0.2$, $C_t = 0.77$, and cement = 4 pct of tailings weight. Thus,

$$S_t = 1.0/1.04 = 0.96,$$

$$S_c = 0.04/1.04 = 0.038,$$

and $C_c = 0.80 \times 0.04 = 0.032$.

$$G_r = \frac{(0.77 \times 2.8) + (0.032 \times 3.15)}{0.802} = 2.81$$

or $G_r = (0.96 \times 2.8) + (0.038 \times 3.15) = 2.81$,

and $G_s = \left\{ \frac{0.2 + 0.77 + 0.032}{2.81} \right\}^{-1} = 2.06$.

Therefore, for a 4.85 m³ batch,

$$\begin{aligned} \text{target weight is } & 4.85 \text{ m}^3 \times 2.06 \times 1,000 \text{ kg/m}^3 \\ & = 10,000 \text{ kg,} \end{aligned}$$

$$\text{weight of water is } 0.2 \times 10,000 \text{ kg} = 2,000 \text{ kg,}$$

$$\begin{aligned} \text{weight of dry solids is } & 10,000 \text{ kg} - 2,000 \text{ kg} = \\ & 8,000 \text{ kg,} \end{aligned}$$

and weight of cement is $0.04 \times 8,000 \text{ kg} = 320 \text{ kg}$.

NOTE: Calculated slurry C differs from the value shown in table 1 because water was added to adjust the slump during batching, thus changing the target weight value of the water. The slurry C was determined from figure 7, which shows the relationship between slurry concentration and slump as determined from laboratory tests. The slurry C for a 17.8-cm slump = 79.6.

Water-Cement Ratio

The water-cement ratio, w/c, is the ratio of the weight of the water to the weight of the cement. Therefore, w/c = weight of water ÷ weight of cement.

Continuing with the example described above, w/c = 2,000/320 kg = 6.25.

Addition of Water

Water addition is expressed in two ways: as a percentage of the solids weight (water content), which is used by civil

and geotechnical engineers, and as a percentage of the total slurry weight (water, concentration by weight), which is used by metallurgical engineers.

Water Content

Water content as calculated by civil and geotechnical engineers is the ratio of the weight of the water to the weight of the solids given a unit volume of material. Therefore, water content = weight of water per unit volume ÷ weight of solids per unit volume.

Water content for the bunker-stored tailings was determined by taking samples of the tailings from different locations within the storage bunker and then averaging the values. The water contents of individual tailings samples were determined by subtracting the oven-dried weight of each sample from the original stored weight and dividing this value by the oven-dried weight. For example, the water content of tailings sample 1 (table C-1) was $12.9 \div 139.1 \text{ g} = 9.3 \text{ pct}$.

The water content for the bunker-stored tailings was determined by averaging the water content values of seven individual samples, so that $(9.3 + 9 + 9.1 + 10.8 + 8.3 + 6.5 + 6.1) \div 7 = 8.5 \text{ pct}$.

Because the stored tailings contained water, when batching, additional tailings samples were added to compensate for the weight lost to water. The water content of the tailings and the weight of the dry tailings were used to determine the amount of additional stored tailings and water needed for the mix.

The weight of the stored tailings needed to batch the mix was determined by multiplying the dry tailings weight by 1 and the water content, so that weight of stored tailings = weight of dry tailings $\times (1 + \text{water content})$, or $8,000 \text{ kg} \times (1 + 0.085) = 8,680 \text{ kg}$.

The amount of additional water needed for batching the mix was determined by subtracting the weight of the water contained in the stored tailings samples from the original weight of the water needed for the mix, or $2,000 \text{ kg} - (8,680 \text{ kg} - 8,000 \text{ kg}) = 1,860 \text{ kg}$.

Water, Concentration by Weight

Water, concentration by weight, as used by metallurgical engineers, is calculated as the ratio of the weight of the water to the weight of the total sample given a unit volume of material. Therefore, water, concentration by weight = weight of water per unit volume ÷ total slurry weight per unit volume.

Water, concentration by weight, for the bunker-stored tailings was determined by taking samples of tailings from different locations within the storage bunker and then averaging the values. The water, concentration by weight, of individual tailings samples was determined by subtracting the oven-dried weight of each sample from the original stored weight, and dividing this value by the original stored

weight. For example, the water, concentration by weight, of tailings sample 1 (table C-1) was $12.9 \text{ g} \div 152.0 \text{ g} = 8.5 \text{ pct}$.

The water, concentration by weight, of the bunker-stored tailings was determined by averaging the values of water, concentration by weight, of seven individual samples, so that $(8.5 + 8.3 + 8.4 + 9.8 + 7.7 + 6.1 + 5.8) \div 7 = 7.8 \text{ pct}$.

Because the stored tailings contained water, when batching, additional tailings samples were added to compensate for the weight lost to water. The water, concentration by weight, of the tailings and the weight of the dry tailings were used to determine the amount of additional stored tailings and water needed for the mix.

The weight of the stored tailings needed to batch the mix was determined by dividing the dry tailings weight by 1 minus the water, concentration by weight, so that weight of stored tailings = weight of dry tailings \div (1 - water, concentration by weight), or $8,000 \text{ kg} \div (1 - 0.078) = 8,680 \text{ kg}$.

The amount of additional water needed for batching the mix was determined by subtracting the weight of the water contained in the stored tailings samples from the original weight of the water needed for the mix, or $2,000 \text{ kg} - (8,680 \text{ kg} - 8,000 \text{ kg}) = 1,860 \text{ kg}$.

Void Ratio

The void ratio is the ratio of the volume of voids to the volume of solids per unit volume. The equation is given as

$$e = V_v/V_s,$$

where e = void ratio,

V_v = volume of voids occupied by air and water,
and V_r = volume of solids.

V_v , which is the sum of the volume of air and the volume of water per unit volume, is determined as

$$V_v = V_a + V_w,$$

where V_a = volume of air

and V_w = volume of water.

V_r , which is the sum of the volume of tailings and the volume of cement per unit volume, is determined as

$$V_r = V_t + V_c,$$

where V_t = volume of tailings

and V_c = volume of cement.

EXAMPLE

Using data from table C-2 for mix 3, sample 1,

$$V_v = 0.00049 \text{ m}^3 + 0.0028 \text{ m}^3 = 0.00329 \text{ m}^3,$$

$$V_r = 0.0047 \text{ m}^3 + 0.00017 \text{ m}^3 = 0.00487 \text{ m}^3,$$

and $e = 0.00329/0.00487 \text{ m}^3 = 0.68.$

Porosity and Degree of Saturation

Porosity is the ratio of the volume of voids to the total volume per unit volume, or

$$n = V_v/V,$$

where n = porosity,

V_v = volume of voids,

and V = total volume.

Degree of saturation is the ratio of volume of water to volume of voids per unit volume, or

$$S = V_w/V_v,$$

where S = degree of saturation,

V_w = volume of water,

and V_v = volume of voids.

EXAMPLE

Using calculations from the previous example,

$$n = 0.00329/0.0081 \text{ m}^3 = 0.41$$

and $S = 0.0028/0.00329 \text{ m}^3 = 0.85.$

PRESSURE GRADIENT

Pressure exerted on the walls of the pipeline by material being transported was measured using transducer pairs mounted in the pipeline. Differential pressure was determined by calculating the difference between upstream and down-stream pressure measurements. The pressure gradient as a function of flow rate, pipe size, and mix combination is shown

in figure 4. Pressure gradient relationships are determined from the ratio of the differential pressure difference to the pipe length between the transducer pairs according to the following equations.

$$dP/dL = (P_{in} - P_{out})/L_{adj}$$

where dP/dL = differential pressure \div differential length = pressure gradient,

P_{in} = pressure at upstream transducer,

P_{out} = pressure at downstream transducer,

and L_{adj} = adjusted length between upstream and downstream transducers.

L_{adj} is calculated according to the equation

$$L_{adj} = L_{in-out} + L_{fit}$$

where L_{in-out} = sum of length between upstream and downstream transducers

and L_{fit} = equivalent length of additional pipe resulting from line fittings.

EXAMPLE

Using mix 3 at a rate of 3.1 strokes per minute through 154-mm-diam pipe (table C-3),

$$L_{adj} = 42 \text{ m} + 3 \text{ m} = 45 \text{ m}$$

and $dP/dL = (1.32 \text{ MPa} - 0.82 \text{ MPa})/45\text{m} = 11 \text{ kPa/m}$.

The locations of the transducers in the pipeline and the distance between pairs were changed for each of the six loop tests, as shown in figure 3.

The vertical component of the pressure gradient was obtained by multiplying the unit weight of the mixes by acceleration due to gravity (9.81 m/s²).¹ Thus,

$$dP/dL_{grav} = \text{unit weight} \times g_a$$

where dP/dL_{grav} = vertical component of pressure gradient

and g_a = gravity.

EXAMPLE

Using the unit weight for mix 3 (table 1),

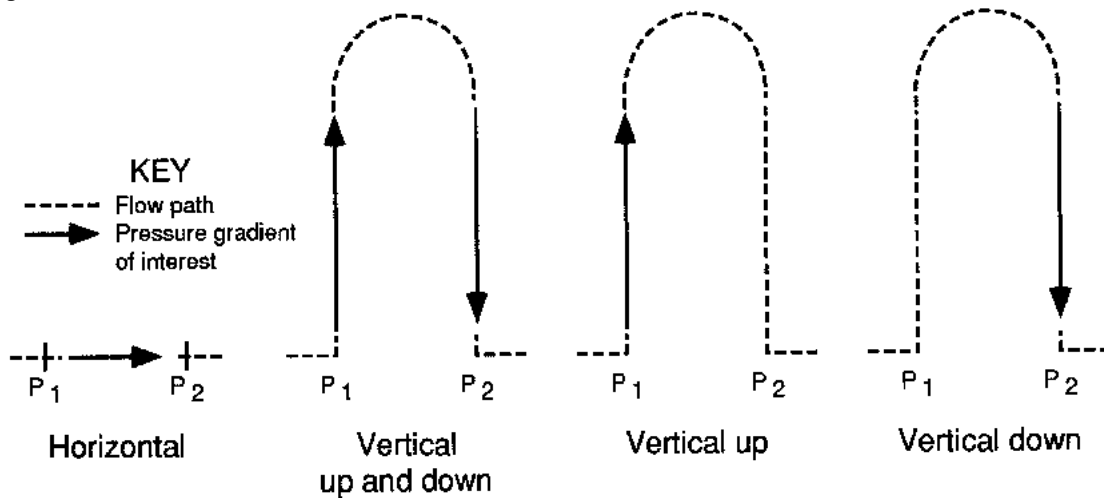
$$dP/dL_{grav} = 2,060 \text{ kg/m}^3 \times 9.81 \text{ m/s}^2 = 20.2 \text{ kPa/m}$$

The pressure gradient for vertical up or vertical down transport is obtained by adding or subtracting the vertical component plus the frictional component of the pressure gradient. The frictional component of the pressure gradient is the resistance present regardless of the orientation of the pipe (table C-4 and figure C-1).

Therefore,

¹In the U.S. customary system, the unit weight must first be converted to an equivalent force in slugs. Therefore, unit weight = 129 lb/ft³ \div 32.2 lb \cdot ft/lb \cdot s² = 3.88 lbf \cdot s²/lb \cdot ft⁴, so that $dP/dL_{grav} = 3.88 \text{ lbf} \cdot \text{s}^2/\text{lb} \cdot \text{ft}^4 \times 32.2 \text{ ft/s}^2 \div 144 \text{ in}^2/\text{ft}^2 = 0.90 \text{ psi/ft}$.

Figure C-1



Horizontal and vertical flow paths and pressure gradients.

$$dP/dL = \pm dP/dL_{grav} + dP/dL_{vert\ up\ and\ down}$$

where $dP/dL_{vert\ up\ and\ down}$ = frictional component of pressure gradient - pressure gradient for horizontal transport.

The following example shows the calculations involved in determining the vertical up and vertical down pressure gradients (table C-4).

EXAMPLE

Using mix 3,

$$dP/dL_{vert-up} = 20.2\text{ kPa/m} + 12\text{ kPa/m} = 32\text{ kPa/m}$$

and $dP/dL_{vert-down} = -20.2\text{ kPa/m} + 12\text{ kPa/m} = -8\text{ kPa/m}$.

Flow Rate

Flow rate is a function of pump speed and displacement and is determined by taking the product of the stroke rate per minute and multiplying this figure by the volumetric displacement of the pump cylinders. Thus,

$$Q = N \times V_p,$$

where Q = flow rate,

N = number of strokes per minute,

and V_p = volumetric displacement, cubic meters per stroke.

It was assumed that 100 pct of the material in each stroke was actually delivered because the material fed directly into the pumping cylinders and no air pockets were observed at the pump discharge point.

The equation for the volumetric displacement of the Thompson TTS 2065 pump is

$$V_{TTS\ 2065} = L_{TTS\ 2065} \times \pi \times (D_{TTS\ 2065})^2/4,$$

where $L_{TTS\ 2065}$ = stroke length of Thompson pump cylinder

and $D_{TTS\ 2065}$ = diameter of cylinders.

Therefore, $V_{TTS\ 2065} = 1\text{ m} \times \pi \times (0.1778\text{ m})^2/4 = 0.0248\text{ m}^3$ per stroke.

The equation for the volumetric displacement of the Putzmeister 2100S pump is

$$V_{2100S} = L_{2100S} \times \pi \times (D_{2100S})^2/4,$$

where L_{2100S} = stroke length of Putzmeister pump cylinder

and D_{2100S} = diameter of pump cylinders.

Therefore, $V_{2100S} = 2.1\text{ m} \times \pi \times (0.1778\text{ m})^2/4 = 0.0521\text{ m}^3$ per stroke.

EXAMPLE

Using data from table C-3 for mix 3 at a rate of 3.1 strokes per minute,

$$Q = 3.1\text{ strokes per minute} \times 60\text{ min/h} \times 0.0521\text{ m}^3\text{ per stroke} = 10\text{ m}^3/\text{h}.$$

Table C-5 lists the test results used to obtain figure 5 and is based on data from tables B-1, B-4, and C-3. Mixes are compared factor by factor in table 2. The most significant factor for determining which mix will have a greater pressure gradient is shown in bold. Pressure gradients for water and test mix 6 at different flow rates are given in table C-6 and shown in figures 15 and 16.

Table C-5.—Combined test results for total tailings paste backfill at flow rate of 29 m³/h

Mix	Slump, cm	Pressure gradient (dP/dL)		Slurry concentration, pct	Particle size, mm
		kPa	m/m		
Fine tailings:					
6	6	18.0	0.92	80	0.05
5,1	11	15.3	0.78	79	0.06
Coarse tailings:					
5,1	11	28.1	1.43	81	0.109
4	13	21.2	1.08	80	0.100
3	15	10.3	0.52	80	0.103
2	17	6.1	0.31	80	0.109

Horizontal Transport Distance

The horizontal transport distance generated by a standing column of material is obtained by dividing the frictional pressure gradient into the pressure at the bottom of the standing column. Thus,

Table C-6.—Pressure gradients for water and mix 6 at various pipe diameters

Flow rate (Q), m ³ /h	102 mm		128 mm		154 mm	
	v, m/s	dP/dL, kPa/m	v, m/s	dP/dL, kPa/m	v, m/s	dP/dL, kPa/m
WATER (25 °C)						
10	0.34	0.011	0.22	0.004	0.15	0.001
15	0.51	0.025	0.32	0.008	0.22	0.003
20	0.68	0.045	0.43	0.014	0.30	0.006
25	0.85	0.070	0.54	0.022	0.37	0.009
30	1.02	0.101	0.65	0.032	0.45	0.013
35	1.19	0.137	0.76	0.044	0.52	0.017
40	1.36	0.179	0.86	0.057	0.60	0.023
45	1.53	0.226	0.97	0.073	0.67	0.029
50	1.70	0.280	1.08	0.090	0.75	0.036
MIX 6						
10	0.34	18	0.22	17	0.15	13
15	0.51	24	0.32	23	0.22	15
20	0.68	27	0.43	26	0.30	17
25	0.85	30	0.54	27	0.37	17
30	1.02	33	0.65	28	0.45	19
35	1.19	36	0.76	29	0.52	21
40	1.36	39	0.86	30	0.60	22
45	1.53	41	0.97	31	0.67	24
50	1.70	42	1.08	35	0.75	25

dP/dL Pressure gradient.

v Velocity.

$$\text{transport distance} = P_{\text{bottom}} \div dP/dL,$$

where P_{bottom} = pressure at bottom of a standing column

and dP/dL = pressure gradient.

The pressure at the bottom of a standing column is obtained by taking the difference between the pressure imparted by gravity and pressure lost through frictional pressure gradient, so that

$$P_{\text{bottom}} = P_{\text{grav}} - P_{dP/dL},$$

where P_{grav} = column height $\times g_a \times$ unit weight

and $P_{dP/dL}$ = column height $\times dP/dL$.

The following example shows the calculations involved in determining system pressure and transport distance as given in tables C-7 and C-8.

EXAMPLE

For mix 2 (tables 1 and C-7), where column height is 300 m, pipe diameter is 154 mm, and flow rate is 29 m³/h,

$$P_{\text{grav}} = 300 \text{ m} \times 9.81 \text{ m/s}^2 \times 2,060 \text{ kg/m}^3 = 6.1 \text{ MPa},$$

$$P_{dP/dL} = 300 \text{ m} \times 6 \text{ kPa/m} = 1.8 \text{ MPa},$$

$$P_{\text{bottom}} = 6.1 \text{ MPa} - 1.8 \text{ MPa} = 4,263 \text{ kPa},$$

and transport distance = $4,263 \text{ MPa} \div 6 \text{ kPa/m} = 710 \text{ m}$.

Table C-7.—Transport distance of total tailings paste backfill through 154-mm-diam pipe at different column heights at flow rate of 29 m³/h, meters¹

	Mix 1	Mix 2	Mix 3	Mix 4	Mix 5	Mix 6
Column height, m:						
0	0	0	0	0	0	0
100	-28	237	84	-12	31	10
200	-56	474	167	-23	63	19
300	-83	710	251	-35	94	29
400	-111	947	335	-47	126	38
500	-139	1,184	419	-59	157	48
600	-167	1,421	502	-70	189	57
700	-195	1,658	586	-82	220	67
800	-223	1,894	670	-94	252	76
900	-250	2,131	753	-105	283	86
1,000	-278	2,368	837	-117	315	95
dP/dL, kPa/m	28	6	11	23	15	18

dP/dL Pressure gradient.

¹Negative numbers indicate no flow.

Table C-8.—System pressure of total tailings paste backfill through different pipe diameters at different column heights and flow rate of 29 m³/h, mix 2

Column height, m	P _{grav} ¹ , MPa	102 mm ¹		128 mm ²		154 mm ³	
		P _{bottom} ¹ , MPa	Transport dist., m	P _{bottom} ¹ , MPa	Transport dist., m	P _{bottom} ¹ , MPa	Transport dist., m
1	0.020	0	0.06	0.006	0.40	0.014	2.0
10	0.20	0.012	0.64	0.062	4.0	0.14	24
20	0.40	0.024	1.3	0.120	9.0	0.28	47
30	0.61	0.036	1.9	0.190	13	0.43	71
40	0.81	0.048	2.5	0.250	18	0.57	95
50	1.0	0.060	3.2	0.310	22	0.71	118
60	1.2	0.073	3.8	0.370	27	0.85	142
70	1.4	0.085	4.5	0.430	31	0.99	166
80	1.6	0.097	5.1	0.500	35	1.1	189
90	1.8	0.11	5.7	0.560	40	1.3	213
100	2.0	0.12	6.4	0.620	44	1.4	237
200	4.0	0.20	13	1.2	89	2.8	474
300	6.1	0.40	19	1.9	133	4.3	710
400	8.1	0.50	25	2.5	177	5.7	947
500	10	0.60	32	3.1	222	7.1	1,184
600	12	0.70	38	3.7	266	8.5	1,421
700	14	0.80	45	4.3	310	10	1,658
800	16	1.00	51	5.0	355	11	1,894
900	18	1.10	57	5.6	399	13	2,131
1,000	20	1.20	64	6.2	443	14	2,368
2,000	40	2.0	127	12	887	28	4,736
3,000	61	4.0	191	19	1,330	43	7,104
4,000	81	5.0	254	25	1,774	57	9,472
5,000	101	6.0	318	31	2,217	71	11,841
6,000	121	7.0	382	37	2,661	85	14,209
7,000	141	8.0	445	43	3,104	99	16,577
8,000	162	10	509	50	3,548	114	18,945
9,000	182	11	572	56	3,991	128	21,313
10,000	202	12	636	62	4,435	142	23,681

P_{grav} = Column height × gravity × unit weight.

P_{bottom} = Pressure at bottom of standing column.

¹Pressure gradient (dP/dL) = 19 kPa/m. ²dP/dL = 14 kPa/m. ³dP/dL = 6 kPa/m.