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Summary Memorandum

To:	Tailings Relocation Working Group: D. Hockley, C. Scott, M. Stepanek, V. Enns, J. Brodie	**Date:**	April 12, 2006
cc:	V. Chort	**From:**	Gordon Doerksen
Subject:	Tailings Relocation, 2005/06 Task 22a	**Project #:**	1CD003.79

In early 2005, SRK issued a draft report on tailings relocation under 2004/05 Task 16a, Develop Tailings Relocation Plan. The report, which covered most of the issues related to tailings relocation using hydraulic monitoring methods, was reviewed by various technical reviewers. As a result of their comments and discussions during subsequent planning meetings, the 2005/06 study requirements related to tailings relocation were directed to the following two main issues:

- Lime demand testing; and
- Hydraulic monitoring review.

The scope of the work on lime demand testing was described in a proposal from SRK issued initially in the middle of the 2005 field season. In December 2005, a technical memorandum summarizing the results of the lime demand testing was circulated to the Tailings Relocation Working Group.

The scope of the work on hydraulic monitoring review was based largely on a Tailings Relocation Working Group meeting held on September 29, 2005. It comprised incremental work on the following:

- Case Studies of Hydraulic Mining Operations
- Hydraulic Mining
- Slurry Handling
- Cost Estimation

SRK produced a technical memorandum for each of these topics between October 2005 and March 2006 under Task 22a.

This memorandum puts all of the techincal memoranda prepared under 2005/06 Task 22a in a single document. They are included with this document as Attachments A through F. A brief discussion of each of these memoranda is provided below.

Subsequent to the receipt of comments from the Tailings Relocation Working Group on each of these memoranda, it is SRK's intent that each of these memoranda be finalized. SRK understands that, as part of the work required during the 2006/07 period, the final version of each of these memoranda will be "rolled into" the draft report from 2004/05 Task 16a in order to produce an updated, more comprehensive report on tailings relocation.

1 Case Studies of Hydraulic Mining Operations – Attachment A

A review of six hydraulic mining operations was documented with five involved in the relocation of mine tailings. As expected, all the operations varied considerably in their applicability to the Faro project and in their success.

Only one operation, Giant Yellowknife, ran into problems with maintaining high slurry densities. The Yellowknife project struggled because the tailings were frozen and were not easily slurried unless they were thawed.

Los Bronces in Chile had problems when the distance from the mining face to the sumps became excessive and the gradient in the slurry ditches was no longer steep enough to support slurry flow. All of the projects described had solids of lower density than the Faro tailings. The case study solids densities were normally around 2.7 t/m3. The high density of the Faro tailings solids will likely mean that slope gradients from the mining face to the sumps will need to be higher than other operations, necessitating the need to keep the sumps as deep as possible and close to the mining faces. The actual gradient needed to keep the solids in suspension will be determined in the field.

2 Hydraulic Mining – Attachment B

Additional consideration was given to the hydraulic monitoring system. The proposed hydraulic mining system comprises a total of nine high pressure (24 bar) water monitors supplied with water from multi-stage vertical turbine pumps located in the Faro Pit. Six monitors at a time will operate using a total of 400 l/s of water. The monitors will give a solids mining rate of approximately 27,000 tonnes/day or 4,000,000 tonnes per 5-month operating year from a slurry containing 46% solids by mass.

The monitors can be arranged in two different mining configurations. In the first scenario, the monitors can be located on top of the tailings, pointing down to cut tailings at a 45o slope in front of the monitor. The second scenario has the monitors located below the top level of the tailings, washing a tailings face in front of the monitors. Both set-ups may have applications in the project depending on whether the tailings can support the weight of mobile equipment, the density of slurry produced and safety factors.

3 Slurry Handling – Attachment C

The proposed slurry handling system is made up of vertical cantilever sumps pumps, trash screens, a lime addition system, and a series of slurry booster pumps. The slurry pumping system will move slurried tailings from the mining area to the Faro Pit. The design of the slurry transport system completed for the 2004/05 draft report and subsequent costing efforts (Attachment F) was based on standard, proven equipment made of materials suitable for the application.

Lime addition calculations were conducted in a separate study (Attachment D). For the purpose of slurry system design, it was assumed that lime addition would occur prior to the booster pumps.

4 Lime Demand Testing – Attachment D

Samples selected for the verification testing mirrored the initial sample selection and included additional samples at depth that previously would have been considered to have no lime demand. Good reproducibility was observed for the coarse tailings from the original impoundment, which has the highest lime demand. There is some variability in the test results, most noticeably for the samples from the Original Impoundment fines, Secondary Impoundment fines, and the Secondary Impoundment Coarse samples.

These results were used to calculate the overall lime demand, as described in the 2004/05 assessment, by using the isopach estimates of tailings volume in each depth interval. It was further assumed that 30%, 20% and 25% of the tailings in the Original, Secondary and Intermediate Impoundments, respectively, are coarse.

The results are summarised in Table 1, below. As shown in Table 1, the verification tests indicate a marginally lower overall net lime neutralization cost than the initial tests.

Table 1: Estimated Overall Cost of Lime

Description	Original	Secondary	Intermediate	Total
Initial Estimate	$46,071,000	$50,793,000	$38,276,000	$135,140,000
Verification Tests	$41,579,000	$42,972,000	$24,447,000	$108,998,000
Average	**$43,828,000**	**$46,884,000**	**$31,364,000**	**$122,076,000**

5 Rheology Testing – Attachment E

Slurry pumping tests were conducted at the Saskatchewan Research Council (SRC) on representative tailngs samples collected during the 2005 field season. Sample location details are provided in Attachment E. The slurry pumping tests established deposition velocities associated with tailings slurries at a range of pulp densities. An interesting finding from the study was that the fines in the slurry mix were so dense that they did not help float coarser particles, as is normally the case. The SRC tests also showed that friction losses increased significantly with fluid velocity. This means that excessive velocities in the pipelines must be avoided to keep pumping pressure and operating costs down.

The high specific gravity (4.0) of the tailings solids, while manageable in the pipe /pump circuit, remains an unknown factor in the movement of slurry from the mining face to the sump pumps. Slurry will run from the face in ditches sloped towards the sumps. If depositional velocities in the SRC work hold true for open channel flow, then the gradients of the ditches have to be such that they maintain slurry flows above 1.0 m/s. Field tests will determine what these gradients are and may lead to slightly higher overall project costs due to increased road building to accommodate sump pump movement.

6 Cost Estimation – Attachment F

The cost estimation for the Faro Tailings Relocation project was re-done based on the revised mining and pumping systems and basin clean-up volumes described in the attached technical memos. The resultant cost estimation yielded the following results:

Table 2: Hydraulic Mining Cost Summary

Unit Operation	Tonnes (M)	Capital Cost		Operating Cost		Total Cost	
		Tot. Cost ($M)	Unit Cost ($/t)	Tot. Cost ($M)	Unit Cost ($/t)	Tot. cost ($M)	Unit Cost ($/t)
Hydraulic mining	49.0	$4.9	$0.10	$38.1	$0.78	$43.0	$0.88
Slurry pumping	49.0	$6.4	$0.13	$31.1	$0.64	$37.5	$0.77
Basin cleanup	12.0	$0	$0	$39.0	$3.25	$39.0	$3.25
Decommissioning	61.0	$0	$0	$0.5	$0.01	$0.5	$0.01
TOTAL	**61.0**	**$11.3**	**$0.19**	**$108.7**	**$1.78**	**$120.0**	**$1.97**

A simple sensitivity analysis was done of the main cost drivers in the project. Mining productivity was found to have the largest impact on the cost followed, to a much lesser degre, by basin clean-up cost, labour cost, capital cost and power cost.

Costs were generated from either supplier budgetary quotes or by experience with other similar projects and purchases. Electrical power costs were obtained from Yukon Energy Corp. (YEC) and were quoted at $0.1457/kWhr.

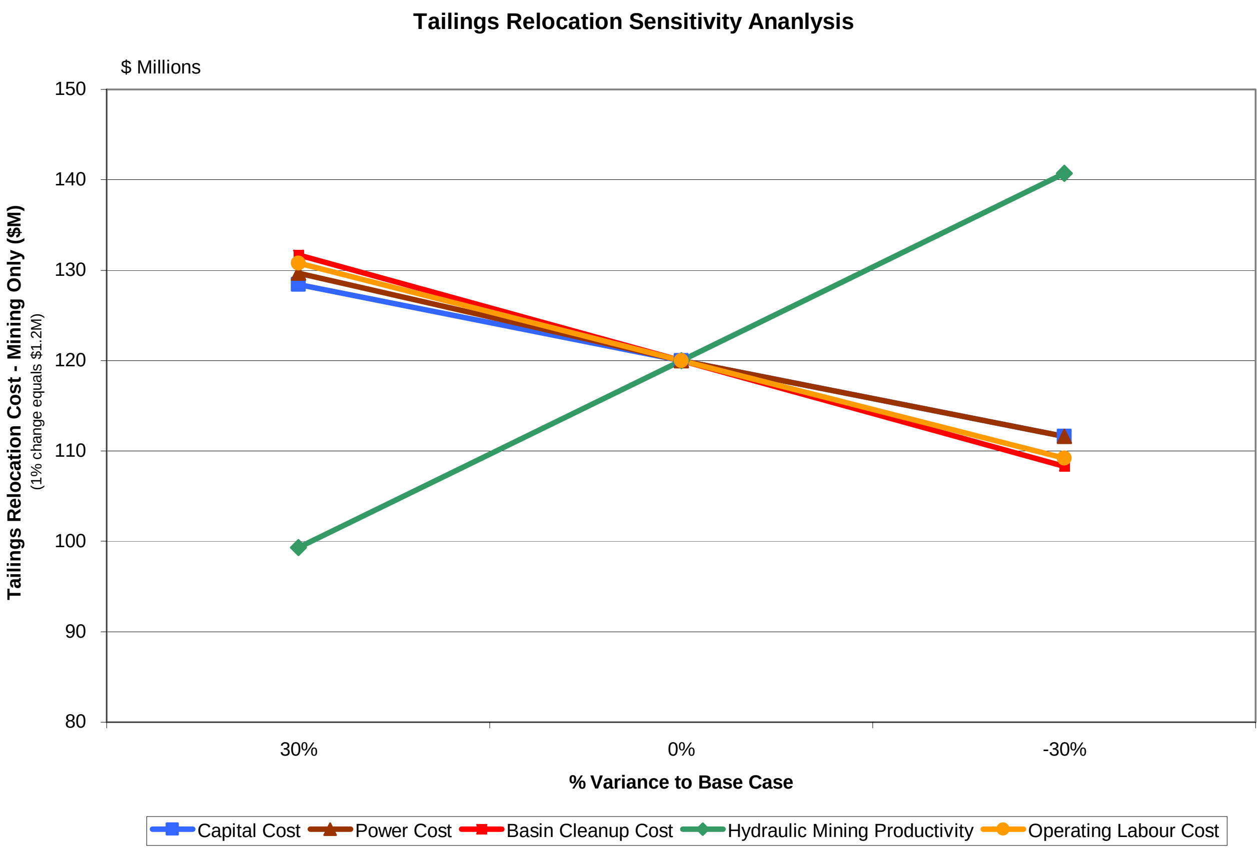

Figure 1 Sensitivity Analysis Graph showing the Effects of Variations of Input Parameters on the Project Cost

Attachment A
Case Studies

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Technical Memo

To:	Daryl Hockley, Cam Scott	**Date:**	February 27, 2006
cc:		**From:**	Gordon Doerksen
Subject:	Case Studies in Hydraulic Mining	**Project #:**	1CD003.079

1 Giant Tailings Retreatment Plant, Yellowknife, NWT

The following information was provided from:

1. The files of Gordon Doerksen, General Foreman of the Giant Tailings Retreatment Plant.
2. Wakabayashi, B.S. 1990 "Tailings Retreatment Plant Information Brochure" Giant Yellowknife Mines Limited.
3. Brodie, John 2003 A Review of Tailings Relocation Projects and Methodology, Brodie Consulting Ltd.

1.1 Background

Giant Yellowknife Mines Limited ran a 7,300 tonne per day, seasonal, hydraulic mining operation from 1988-90. Hydraulic mining was used to transport settled mill tailings from old tailings ponds to a Carbon-In-Leach (CIL) Tailings Retreatment Plant (TRP). The TRP and hydraulic mining systems were designed to economically extract residual gold from old Giant mill tailings. The project, when proposed, had a positive rate of return and the TRP was built on the strength of information gained from the results of a pilot plant operated in the summer of 1987.

Figure 1.1: Giant Yellowknife TRP Carbon-In-Leach (CIL) tanks

1.2 Hydraulic Monitoring

The hydraulic mining system at Giant utilized an undercut configuration mining the full height of the tailings in a single face (see Figure 1.2). Full face mining was very important to the TRP project economics. The Giant tailings were deposited over a period of over 40 years and the grade of the tailings varied significantly with depth. The lowest, oldest tailings had a very favourable grade of over 3g/tonne. Subsequence layers of tailings were progressively lower in grade as mill head grades declined and as advances in mineral processing extracted more gold out of the ore. The TRP project would not have been economic if the tailings had been mined from the top down.

Figure 1.2: Hydraulic monitor washing frozen tailings

The Giant tailings deposit was up to 27 m thick, however, over 90% of the material was less than 20 metres in thickness. Because the depth of the tailings was not deemed excessive, it was decided to mine the full height of the tailings at once without any intermediate benching.

The first phase of mining involved the hydraulic monitors cutting their way down into the tailings from the side of the tailings pond, advancing to the bottom on ramps made of waste rock (Figure 1.3). Initially, production was stopped when the ramps were advanced. Once mining progressed far enough, the ramp was split to allow one ramp to be advanced while the other continued to support mining.

The Giant operation utilized three English China Clay (ECC) 4 inch (10 cm) automated monitors. The monitors were fed by two Mather+Platt seven stage mine dewatering pumps arranged in parallel. (Figure 1.4) The monitors operated at up to 350 psi (2,413 kPa) pressure depending on the nozzle diameter used 1-1.5 inch; (2.5-3.8 cm). Giant machined their own monitor nozzles to provide the optimum water jet from the monitor. Maintaining a tight water jet was important as it allowed the monitors to be kept further back from the mining face for safety and because it assisted in cutting of the frozen tailings. The monitors were typically kept between 20 and 40 m from the face.

Figure 1.3: Start of the first ramp at Giant – May 1888. Note frozen tailings deflecting high pressure water. Operators booth and trommel screen are on the right.

Figure 1.4: Mather+Platt multi-stage mine dewatering pumps connected in parallel used for high-pressure monitor feed water

The monitors were controlled by one operator located in a booth well away from the mining face. The ECC monitors used at Giant were hydraulically controlled with joysticks. The monitors were run either on manual or automatic. In automatic mode, the monitor moved in a pre-set pattern allowing the operator to leave the control booth. Pelton wheels, driven by the high pressure monitor feed water, powered the hydraulic pump for the monitor controls and an alternator charged the battery used for the electronic controls.

Figure 1.5: Inside of the operator's booth showing monitor controls

Figure 1.6: Crane and Toyo submersible sump pumps on rock-fill ramp

1.3 Slurry Pumping

The hydraulic monitors slurried the tailings as they were washed from the mining face. The slurry ran back to the sump located near the monitors. One hundred horsepower Toyo submersible pumps with agitators were used as the sump pumps and were suspended by a crane (see Figure 1.7 to 1.11). The crane allowed the pumps to be moved easily and the pumps were raised and lowered depending on sump conditions. The Toyo pumps had an agitator on the bottom of them that stirred up the slurry in the sump. This pump arrangement actually allowed the pumps to dig themselves down into the sump by liquefying settled slurry prior to pumping. The Toyo pumps pumped tailings up from the sump to a trommel trash screen. The trommel allowed fine material to pass through separating the rocks and sticks from the slurry. The oversized debris was discharged into a pile to be scooped up with a front-end loader. The slurry flowed through the trommel to a 12 x10 Warman horizontal slurry pump that transported the slurry to the TRP.

The gradation of the Giant tailings was very fine with 90% of the material passing the 400 mesh sieve. The specific gravity of the tailings solids was approximately 2.7.

1.4 Operating Results

The Giant TRP experienced several operational challenges during it existence. Difficulties associated with trying to hydraulically mine frozen tailings were the biggest challenges. It was determined after the project had started that the tailings were frozen almost through their entire depth. This fact lead to the tailings being very hard to slurry and subsequently yielded low slurry densities. The target slurry composition was 40% solids by weight. While this target was reached on occasion, the percent solids in the slurry during the early part of the summer was very low and only improved as the mining faces thawed at an increasing rate (see Table 1.1 below). Slurry densities of >40% solids by weight were never maintained for long periods.

Table 1.1 Percent solids in slurry – Start of 1988 season

	Solids by Weight	*Tonnes Mined*
May	14%	49,547
June	32%	143,925
July	38%	207,970
August	36%	195,457

The hydraulic monitors proved unable to cut the tailings at a rate high enough to thaw and liberate tailings solids to maintain high slurry densities. The attempt to mine frozen tailings hydraulically can be likened to the melting of an ice cube under a stream of cold water from a tap. A significant amount of water is required to melt the ice.

Table 1.2 Giant TRP Production Statistics 1988 and 1989

	Design	*Actual*
Design Flow Rate	2,900 USGPM	Not available
Slurry Design	40% solids by weight	14-48%
Design Productivity	7,272 tonnes/day (t/d) 310 tonnes/hour (t/hr)	Not available 263 t/hr (1998) 305 t/hr (1989)
Operating		89.9% (1988) 91.5% (1989)
Total Production	1,090,800 tonnes/year	858,852 tonnes (1988) 994,634 tonnes (1989)

Figure 1.7: View of the Giant mining operation from on top of the tailings pond

Figure 1.8: View of mining face showing monitor and crane.

Figure 1.9: Toyo pump suspended from crane

Figure 1.10: Initial mining setup in the Central Pond showing crane, monitor and operator's booth

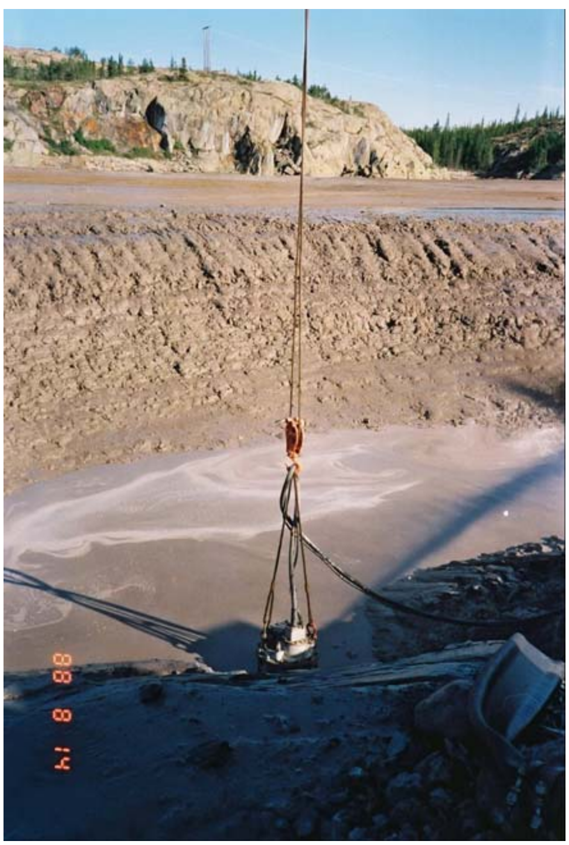

Figure 1.11; Toyo pump starting a sump in the Central Pond

It was very clear during the first mining season that production was going to be directly linked to the amount of natural thawing of the tailings during the warm summer months. As mining advanced, more and more face surface area was available for thawing and for mining by monitors.

Table 1.3 TRP Mining Cash Costs

	1988*	1989**
Mining Operations	$577,000	$970,000
Mining Maintenance	$231,000	$273,000
General***	$164,000	$329,000
Total Costs	$972,000	$1,572,000
Tonnes Mined	858,852	994,634
Cost per tonne	$ 1.13	$1.58 ($1.03 excluding dozer)

* June-December 1988

** Costs in last quarter of 1989 were forecast. 1989 costs included $552,000 for pushing tailings with a dozer

*** Prorated from total general costs

Toyo submersible pump maintenance accounted for roughly 50% of maintenance costs.

To help increase production, a contract D9 dozer was placed on the top of the tailings pond to push thawed surface tailings onto the hydraulic mining face (Figure 1.12). This arrangement lead to higher costs and the lowering of head grade to the TRP. However, it assisted with raising the density of the slurry.

Low slurry densities associated with mining the frozen tailings had the following major effects on the viability of the Giant TRP project:

1. The inability to move the desired volume of tailings meant that the actual tonnes of gold-bearing solids to the plant were reduced, lessening gold extraction.
2. The low density slurry in the plant hindered the dispersal of activated carbon in the leach tanks and therefore lessening the carbon's ability to adsorb gold.
3. To help improve the solids tonnage through the plant when low slurry densities where encountered, the total volume of water put through the plant was increased. The increased flow reduced the retention time in the tanks lessening gold dissolution time.
4. The use of a dozer on top of the pond pushing thawed tailings toward the monitors reduced the head grade going to the plant.

Figure 1.12: Dozer pushing thawed tailings to mining face

2 ERG Operation, Timmins, Ont.

The information in this section of the report was provided from personal observations of Gordon Doerksen, Mining Engineer, Giant Yellowknife Mines Limited – Timmins Division and from a University of London PhD thesis by Sadek El-Alfy entitled "Hydraulic Mining in Cold Regions" (1996). Mr. El-Alfy was the Manager of the Giant TRP and later the Timmins ERG Operations.

2.1 Background

The ERG operation in Timmins was based on the same tailings retreatment philosophy as its sister operation in Yellowknife. Tailings in the Timmins area, however, were coarser than the Yellowknife operation and had a lower average grade of 0.43 g/t. Some of the 130 million tons of ERG's tailings resources were located close to downtown Timmins so not only could ERG reprocess the tailings and generate a profit, but the town of Timmins would receive reclaimed land previously made unusable by the tailings piles. That plan, along with the consolidation of tailings deposition sites into one remote site, made the project very attractive to all levels of government.

The project was scheduled to process 1,000,000 tons per month of reclaimed tailings, 8 months per year. The tailings in Timmins were not frozen.

A new processing facility was built to extract residual gold from the tailings and was configured with two open-circuit ball mills in series followed by a cyanidation and CIP/CIL circuit. The total capital cost of the ERG project was $120M.

As with the Giant operation, ERG only ran for a 2-3 years before being shut down in the early 1990s.

2.2 Hydraulic Mining

The Timmins operation employed 6" (15 cm) and 4" (10 cm) ECC automated computer controlled monitors fed at 300 psi (2,069 kPa). Re-pulped tailings flowed at a gradient of 1% from the mining faces to either ditches or sumps. At least three separate mining areas were operated at any given time with a total flow of between 12,000 and 15,000 USGPM.

Surface vegetation (mainly grasses and alder trees) was removed from the tailings surfaces prior to mining so as not to overload the trash screens.

When mining down into tailings ponds, ramps were built in a similar fashion to the Giant operation. Arterial ramps were established every 200 metres to keep the monitors and sump pumps close to the face.

Slurry mixtures varied between 40-55% solids by weight.

2.3 Slurry Pumping

ERG used two methods of slurry handling in the operation. The first method was used when mining well drained, stacked tailings (hills or dams). The method involved advancing through the tailings using monitors positioned at the base of the stacked tailings. Slurried tailings flowed from the mining face to ditches, transporting the slurry to the main pump station. At the main pump station, the tailings were screened and pumped to the treatment plant using horizontal slurry pumps.

The second method was required when the tailings were contained in a lake or depression. A method similar to the Yellowknife mode of operation was used including ramping down into the tailings and keeping a sump pump suspended from a crane as close to the mining face as possible. Submersible Toyo pumps and vertical cantilever pumps were used as the sump pumps. The sump pumps picked up the slurry reporting to the sump and moved it to the main pump station to be screened and further pumped. The trash removal screens used at ERG were 6.8m long x 2.6m wide with 1.5mm openings. As with the Yellowknife operation, it was essential to keep rocks and wood out of the retreatment system.

The best mining costs achieved at ERG were $0.30/ton.

3 English China Clays (ECC), Cornwall, England

The following information was provided from the files of Gordon Doerksen gathered during an information gathering trip in 1985.

ECC operates several hydraulic clay mining sites in Cornwall, England. The monitors are arranged in an undercut mining configuration and are used to liberate kaolin clay from fractures in the host rock. The host rock is drilled and blasted to allow the monitor water the greatest opportunity to clean kaolin from the natural rock fractures. The slurry formed from the monitoring operation is very light with only 15% solids by weight typically achieved. Slurry from the bottom of the pit is pumped up in stages to the top of the pit and then transported to the processing plant through a series of ditches.

Figure 3.1: Dozer and loader moving waste rock while a monitor washes kaolin

In the late 1980s, ECC maintained a large engineering and fabrication division that made hydraulic monitors and monitor control systems. ECC supplied monitors and controls to both the Giant and ERG–Timmins operations. When the ERG operation shut down, Denison Environmental Services purchased the hydraulic mining equipment and used it at the Elliot Lake operation.

4 ERGO East Rand, South Africa

The following information was provided from the files of Gordon Doerksen, gathered during an information gathering trip in 1988 and from the Anglogold Ashanti website:

http://www.anglogold.com/NR/rdonlyres/3799FB7C-A2E3-4E2F-9A03-E4243AFF71E9/0/Ergo.pdf

4.1 Background

The East Rand Gold and Uranium Company (Ergo) was commissioned in 1977 to recover residual gold and uranium, as well as pyrite for the production of sulphuric acid, from slimes dams attached to old mine workings on the East Rand. Ergo was originally planned to operate for 15 years but its life was extended to 25 years in 1985 with the introduction of new technology in the form of a carbon-in-leach (CIL) process. The CIL plant at Ergo allowed for improved recovery and grade of tailings. In its heyday Ergo enjoyed a healthy financial performance, but over the last few years, the slimes dams available for treatment have had a much reduced grade and the plant's profitability has been gradually diminishing. Losses reached the stage where mine closure was scheduled for the end of March 2005.

4.2 Hydraulic Monitoring

South African tailings operations utilize both undercut and overcut methods. The overcut method is used mainly on top of well-drained tailings that have consolidated sufficiently to allow firm under-foot conditions. Undercut mining is done in locations where the tailings are higher in moisture content and therefore less stable.

Figure 4.1; Wheel mounted monitors on top of an ERGO tailings dam.

Typically one operator runs one monitor. Many monitors are operated manually without the aid of hydraulic controls or an operator's booth. Some of the monitors are mounted on wheeled carts that can be easily relocated on top of the well compacted tailings dams.

Six-inch (15cm) hydraulic monitors are the most common size and utilize pressures of 250-300 psi (1,724 - 2,069 kPa).

Figure 4.2: Undercut mining at ERGO

4.3 Slurry Pumping

Slurry from the mining faces runs down excavated ditches to central pumping stations located at lower elevations. Ditches are dug with rubber-tired backhoes. The ditches are dug 1-2 m wide, at a gradient sufficient to at provide adequate velocity to keep the solids in suspension in the slurry. Maintaining high slurry densities and keeping the ditches narrow to maintain a relatively high flow velocity are the two most important factors in keeping solids in suspension.

Figure 4.3 Slurry flowing from a mining face to a collection ditch at ERGO

The pumping stations, often located many hundreds of metres from the mining faces, are well-engineered, permanent structures that may contain several horizontal, centrifugal slurry pumps in series to provide enough pressure to pump tailings long distances to the treatment plants.

4.4 Input from Con Sabbagha

The following detailed information, provided on a question and answer basis, was provided by a former Ergo employee, Con Sabbagha, who is familiar with the hydraulic mining operation. Comments are by G. Doerksen.

Question1: What was the specific gravity of tailings solids (average and range)?
Answer: Density of dry solids 2,7t/m^3 with negligible variation.
Comment: Similar to Giant and Timmins. Less dense than Faro (4.0 t/m^3)

Q2: What was the experience with high s.g. (>3t/m^3) solids? We are planning on mining high s.g. solids (4 t/m^3). What would you anticipate to be the issues?
A: We did not pump solids with a higher density. My experience, however, is higher density increases pumping velocity depending on viscosity of fluid.
C: Agree, velocities may have to be higher with denser material.

Q3: What was the particle size distribution of the tailings (average and range)?
A: 98% of the particles smaller than 300µ, and 12% of particles smaller than 100µ.
C: Initial testing indicates the Faro tailings are much finer at 78% smaller than 100µ. This means that the Faro tailings should stay in suspension longer than Ergo tailings all other things being equal. This fact may counter the settling influence of the denser Faro solids.

Q4: What was the s.g. of slurry (average and range)?
A: Slurry density average 1,42t/m^3 varying from 1,38t/m^3 to 1,5t/m^3.
C: Faro slurry density planned at 1.53 (due to heavier solids)

Q5: What was the gradient of ditches?
A: Ditch gradient 1 : 100 with a minimum of 1 : 150.
C: Will likely need steeper ditches at Faro due to denser solids settling out quicker

Q6: What was the ditch design (width and side slope angles for a designed flow rate)?
A: Ditches had vertical sides up to 1m depth, deeper than 1, sides sloped if ground eroded easily. Width normally 1m to 1,5m.
C: Good design information

Q7: What were the lengths of ditches (ave and range)?
A: *If ditch slope of 1 : 100 can be maintained, ditch can be as long as you like. We had ditches of 2 000m. With slopes of 1 : 150 +- 500m long.*
C: Good information although the ditches will likely need to be steeper at Faro

Q8: What was the design velocity for slurry in pipelines?
A: Design velocity in pipe 1,9m/sec.
C: Similar to Faro design

Q9: What were the pipelines made of?
A: Pipes were unlined mild steel if slurry PH was above 5,5. Rubber lined if PH was lower. All bends more than 45^o rubber lined. If velocity in pipe more than 2m/sec rubber lined. If pipe life of more than 10 years is required, suggest a wear resistant lining.
C: Faro design has all lined steel or HDPE pipe.

Q10: What was the design velocity for slurry in ditches?
A: We never measured velocity in ditches, worked on gradient only, but suggest velocity not less than 1,5m/sec.
C: Should be considered for Faro

Q11: What effect did slurry density have on the ability to suspend particles (in ditches and pipelines)?
A: With densities below 1,4t/m^3 larger particles i.e. above 0,3mm started to settle out.
C: Faro slurry density is planned to be 1.53 but if it drop then settling of particles is likely to occur

Q12: How many tonnes of dry tailings per day did one monitor produce?
A: We used mainly 150mm monitor guns. As a slurry density of 1,42t/m^2 a tonnage of 8 000 to 10 000 dry tons per day could be achieved per gun.
C: Faro plan calls for only 4,500 tonnes per day per monitor

Q13: What were the common orifice sizes for the monitors?
A: Monitor orifice sizes used 48mm, 49mm, 50mm, 51mm, 52mm most commonly used 50mm.
C: Similar to Faro plan.

Q14: What was the average monitor water pressure?
A: Monitor gun pressure minimum 20 Bar, maximum 30 Bar, mostly 24 to 27 Bar.
C: Similar to Faro plan.

Q15: What was the maximum slope angle for tailings when monitors were mining from the top of the tailings?
A: Slope angles of 40^o to 45^o on dry tailings dams and operating gun manually at the barrel (front). If operating gun manually from behind or if tailings dam is wet 15^o to 30^o.
C: Good design consideration points.

Q16: What types of slurry pumps were used? How many stages? What seals were used – mechanical or packing/gland water? What is recommended?

A: Pumps rubber lined Envirotech i.e. ash pumps, single stage pumps, but up to 10 pumps in series, seals - packing with water seal. Mechanical seals not successful.

C: Faro would have only 5 pumps in series all with packing and water seals.

Q17: What operational problems were encountered and how were they overcome?

A: Main operating problems – Water hammer, overcome by correct start up or shut down procedure. Collapse of pump rubber liners overcome by sump control and pump discharge and suction pressures.

C: Water hammer should be considered and designed out of the system at Faro

Q18: What were the tonnes per man-hour for operational and maintenance crews (excluding supervisors)?

A: I do not have production stat available at present.

C: n/a

Q19: What were the safety issues of hydraulic mining?

A: Safety points – face collapse, correct gun operation, general safety procedures and standards in pump stations, water hammer, monitor water pipe laying standards and procedures in dam areas.

C: Valid points and demand consideration if Faro project goes ahead.

Q20: How was final cleanup done (removal of all tailings down to original topography)? What amount (thickness) of tailings was left on the original topography after hydraulic mining? If final clean-up was done how was it accomplished?

A: *Final clean up to original topography with original ground slope of 1 : 150 or steeper +-100mm of slime on average was left with final clean up. With flatter slopes, more material was left.*

C: Faro Project is planning to mechanically excavate the last 2000mm of tailings. This may offer a large cost upside for Faro

5 Denison Mines, Elliot Lake, Ontario

The information contained in this section came from personal correspondence with Ian Ludgate, Denison Environmental Services (Nov. 2005) and from the Denison report entitled "Decommissioning of Denison Mines Tailings Management Areas" written by Ian Ludgate and Roy Morrell.

5.1 Background

Denison Environmental Services ran a hydraulic mining operation to relocate tailings from 1993 to 1996 in Tailings Management Area No. 2 (TMA-2) at the Denison Mines' Elliot Lake operation. The purpose of the tailings relocation was to lower the elevation of the tailings to allow them to be covered with water to mitigate acid generation. A total of 1.4Mm^3 of tailings were relocated with 60% of the material going underground into old workings and 40% deposited in the lower portions of TMA-1. The project was four years in duration, starting with a pilot year in 1993 (See Table 5.1).

Table 5.1 Denison TMA-2 Tailings Relocation Volumes

Year	Volume (m3)
1993	50,000
1994	187,000
1995	1,070,000
1996	130,000
Total	**1,437,000**

The normal Denison tailings gradation was 60% passing the 200 mesh sieve. Some cycloned tailings were much coarser and could only be maintained in suspension as a slurry when they were mixed with the finer mine tailings. The specific gravity of the dry solids in the tailings was 2.7.

The operating season at Elliot Lake was typically May to late October but was longer during years of mild temperatures. No operating problems relating to freezing were encountered as long as the systems were kept running on a 24-hr basis.

Used monitors, sump pumps and some valves were purchased from the Timmins ERG operation. Other equipment was salvaged from the Denison mine site with a minimal amount of new equipment required for purchase. Denison still owns the equipment.

5.2 Hydraulic Monitoring

Hydraulic mining was done with a combination of 4" (10 cm) and 6" (15 cm) monitors located at the bottom of the reclaim face, similar to the Giant Yellowknife arrangement. Two monitoring areas were established and two monitors per area were typically used. The monitors were fed with 300 psi (2,069 kPa) water from two 200 hp 6x8x18A-411 BF Aurora pumps connected in series. The Aurora pumps were fed from a pump house on Quirke Lake.

Used monitors were kept as close as possible to the face and were often moved as close as 10-20' (3m-6m) from the face Figures 5.1 and 5.2). This arrangement yielded an average slurry density of 28-32% solids by weight. This result was acceptable to Denison staff as there was ample water supply and power costs were minimal. If slurry densities needed to be increased, the sump pump was lowered. Slurry densities also increased when small amounts of low pressure water were used to erode unconsolidated tailings.

Figure 5.1: A hydraulic monitor being relocated closer to the mining face

Monitor discharge nozzles were changed to either open or restrict the flow of water coming out of the monitors to get the right volume and density of slurry for the sump pumps.

Hydraulic monitors were outfitted with the automated control feature from ECC but this was rarely used as the operators had better success pointing the monitors at the bottom of a face when left unattended.

Figure 5.2: A hydraulic monitor working the bottom of a mining face

The cleaning of tailings down to original topography was found to be best done with low pressure water or trucks and excavators, where applicable (Figure 5.3). Hydraulic monitors were not used to do the final clean-up as they lifted up too much of the original ground with their high pressure.

Figure 5.3: Excavators and articulated trucks removing tailings from bedrock

5.3 Slurry Pumping

The slurried tailings flowed an average distance of 3-500 metres from the mining faces to a 10x10x26 Goulds VHS vertical cantilever sump pump. Each Goulds pump was configured with a 200 hp motor and was fed by two operating monitors. The pumps were fitted with fully recessed, hardened steel impellers. The fully recessed impeller allowed very large solids to be pumped. The metal impeller had to be replaced at least once per month as it wore out quickly in the highly abrasive Denison tailings.

The sump pumps were mounted in a box structure that had 9” (22.5 cm) holes cut in it (Figure 5.4). The box provided support for the pump and also acted as a screen to keep debris (sticks, etc) out of the pump. Wood mixed in the slurry, at times, created build-up problems for the pump screens and had to be manually cleaned.

Figure 5.4: Goulds vertical cantilever pump and support box

No ditches were dug from the mining face to the sumps and there seemed to be little settling of the solids in the slurry as long as reasonable slurry densities were maintained. The sump pump locations were not moved during the life of the project.

The two Goulds sump pumps delivered the tailings to two different locations. One pump discharged tailings over 2 km away to TMA-1. The discharge into TMA-1 consisted of a discharge barge attached to cables that allowed it to be moved around the pond by means of winches.

The second Goulds sump pump transported the slurry from the sump into a 5m x 5m tank connected to a 12x12x36 D-11-5G Envirotech-Galigher Ash horizontal slurry pump. The Ash pump was driven by a 450 hp motor and discharged into a 400mm HDPE pipeline leading into underground workings.

5.4 Manpower

The Denison tailings relocation operation normally ran 24 hours/day, seven days/week from April to the end of October. Operating crews worked 12-hour shifts.

Manpower for the entire tailings relocation operation was made up of 6 operators per shift covering 4 operating monitors. Generally, one operator walked between control booths to run two monitors. Other personnel were used to move monitors, pipes, booths, etc. A hydraulic excavator was used nearly full-time to move equipment. One millwright and one electrician were available on dayshift to cover routine maintenance and breakdowns.

Supervision was provided with a dayshift foreman and a night shift lead-hand.

The total complement, exclusive of management, technical and administration, was:

Table 5.2 Denison Tailings Relocation Operating Manpower

Job Title	Manpower
Operators	24
Electrician	2
Millwright	2
Supervisor	2
Total	**30**

5.5 Costs

The tailings relocation at Elliot Lake was done with the intention of keeping costs to a minimum. Used equipment, either from the existing mine or purchased, was utilized whenever possible. As a result, some mismatched or non-ideal equipment was put into service but overall the costs improved by using used equipment. Manpower appears to have been kept to a reasonable level.

Table 5.3 Denison TMA-2 Tailings Relocation Costs

Year	Volume (m3)	Cost ($)	Unit Cost ($/m^3)
1993	50,000	$ 536,435*	$10.73
1994	187,000	$ 1,016,898	$ 5.44
1995	1,070,000	$ 2,922,190	$ 2.73
1996	130,000	$ 474,725	$ 3.65
Total	**1,437,000**	**$ 4,950,248**	**$ 3.44**

*Includes capital costs
Costs include all final clean-up

To keep lime addition costs down, reject limestone from an adjacent operation was brought in and placed on the TMA-2 tailings prior to flooding to neutralize the acidity. The limestone was obtained at very little cost, except for transportation and placement.

6 Los Bronces – Chile

Source of information: John Harman and Pepe Moreno, October 2005 Communication

6.1 Background

A major tailings relocation project was undertaken at the Los Bronces copper mining complex in Chile to mitigate a safety/environmental concern relating to the location of the tailings in a valley.

The valley contained 72 million tonnes of tailings, divided into the following three compartments, from upstream to downstream: Copihues (C), Perez-Caldera 1 (PC1) and Perez-Caldera 2 (PC2).

An expansion of the mining project was approved and, by the early 90's, the mining company decided to construct a new tailings facility named "Las Tórtolas" that would contain all of the new tailings stream and the old tailings relocated from the C, PC1 and PC2 compartments.

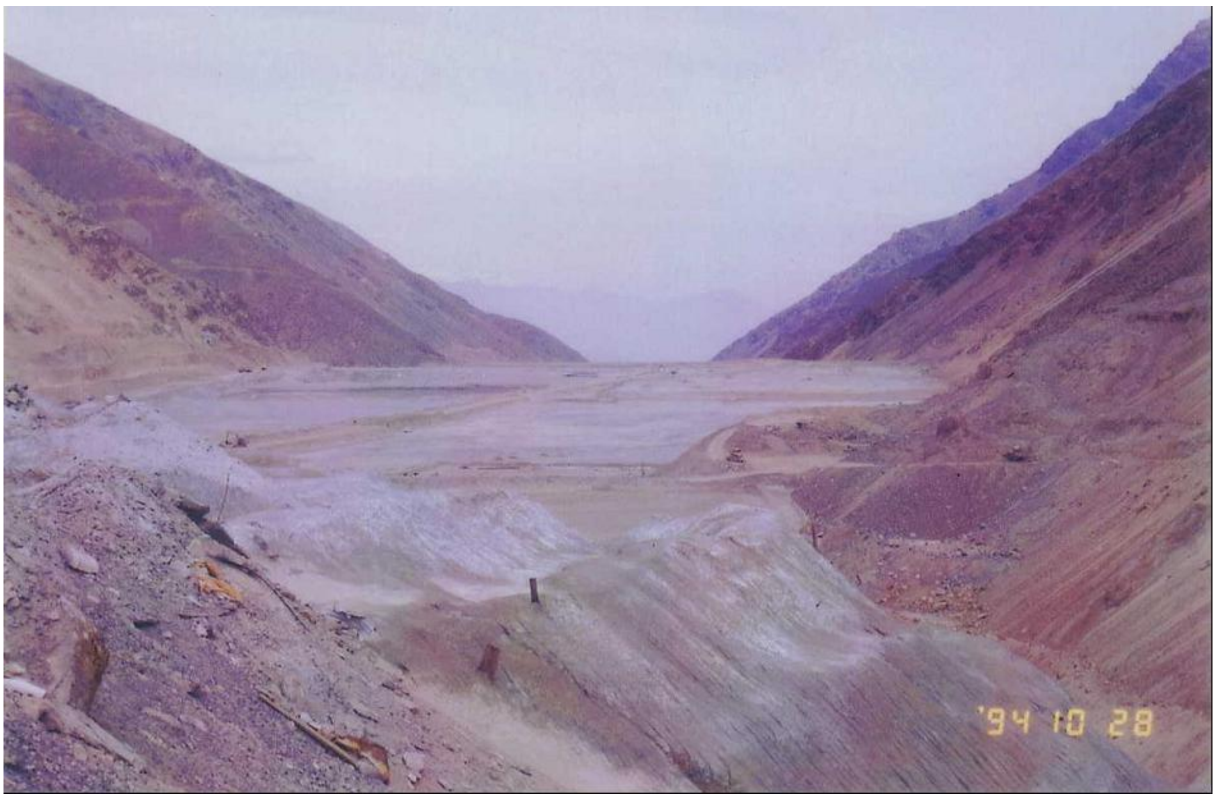

Figure 6.1: Copihues tailings dam prior to commencement of hydraulic mining

The initial project contemplated the use of hydraulic monitors to remove the old, "dried" tailings from Copihues - the smallest compartment situated at the upstream end of the complex - and PC 1, and thereafter continue with a dredge once the monitors would reach the pool level at PC2, if the pool were still there by that time.

The operation with monitors was successfully implemented and kept working until 2003 when the hydraulic head of the tailings stream was not enough to reach the pump station, which was relocated several times until it reached the toe of the PC2 wall.

The mining company decided to keep the pool at the PC2 compartment as a return water structure in order to pump it easily from there to the plant. Two dredgers were bought to carry on with the relocation works.

Fraser Alexander was the contractor retained for the works from the beginning, and as of November 2005 they have bid for a further 4-year contract period.

The contractor operates from September to May due to weather restrictions (the site is at elevation and can experience significant snow falls as shown by Figure 6.2) and its target production is 18 ktpd with a solid concentration of 40 to 45%.

The capital cost for each of the dredge barges is in the order of US$1.5 million.

6.2 Hydraulic Monitoring

The monitoring procedure was revised occasionally to adjust to the project needs. Water pressure, and proper exposure time were determined to get the right mix to flow downstream for further pumping to the Las Tórtolas dam.

The basic procedure consisted of progressively cutting a channel, using hydraulic monitors (Figure 6.3). The channel provided a confined path for the slurried tailings to run and maintain velocity to the pump station. Provided that an adequate gradient was built, the tailings and water slurry flowed downstream to the pump station where it was pumped away to the new tailings site.

Figure 6.2: Pump station during winter. The operation was stopped from June to August - the coldest time of year.

6.3 Dredge vs. Hydraulic Monitoring Comparison

It seems that both methods have been proved to have advantages and disadvantages. However, monitors are more versatile and less vulnerable to production problems and/or delays. Further comparative comments are provided below.

- The hydraulic monitors have a much lower capital cost than the dredges.
- Operational costs are difficult to analyze due to differences in the operational procedures. This difficulty was exacerbated by the preference of the owner to withhold the appropriate cost data. The dredge operates with only one person while monitors require 4 full time labourers. However, maintenance of the dredges is done on a daily basis while the monitors seldom require maintenance. Another operational difference is the time required for mobilization to a different sector. The mobilization of the dredge takes 24 hours where production is completely stopped. The monitors can be changed easily from point to point without interrupting operations.
- Energy consumption is different between the two systems. Monitors are less energy consuming devices since they only require the operation of a high-pressure pump powered by a relatively small engine. The dredge has an 800 HP engine to sustain its operation.
- Maintenance of the dredges is vital for their operation. The engine needs a daily service and the suction system wears out very quickly when handling abrasive material. The monitors need very little maintenance because the system is not directly in contact with the material.
- Dredging does not recover a constant pulp density or constant flow discharge as it depends on the amount of material cut and failed from the slope. Slope failure under water is difficult to predict. Monitors are more flexible. Pressure can be adjusted to meet certain density and flow discharges, but have limitations when the material becomes coarser, i.e. sand wall and/or starter wall.
- Both methods have limitations for a complete cleaning of the bottom of the basin. In both cases, mechanical methods will be required to finish the basin cleaning.

Figure 6.3: Monitor developing a channel for slurry flow

Figure 6.4: Operator controlling a monitor outside of the control booth.

The configuration of the system used 4 hydraulic monitors, strategically located along the channel. All four monitors are connected to a high-pressure water pump and valves were used to regulate the flow to each monitor. Normally only one monitor operated at a time.

The production rate of these monitors ranges from 17 to 22 kton per day with an average slurry composition of 42% of solids by weight.

The system required the gradient for the tailings stream to be high enough to conduct the tailings hydraulically to the point where it would be pumped.

Figure 6.5: Hydraulic monitor in operation cutting a slope

Figure 6.6: Copihues dam showing partial removal of tailings. Note the steps.

Figure 6.7: State of Copihues dam as at June 2004. Note the bottom of the basin is not completely cleaned as the monitors cannot remove the final layer of tailings due to limitations of hydraulic head to make it flow down stream.

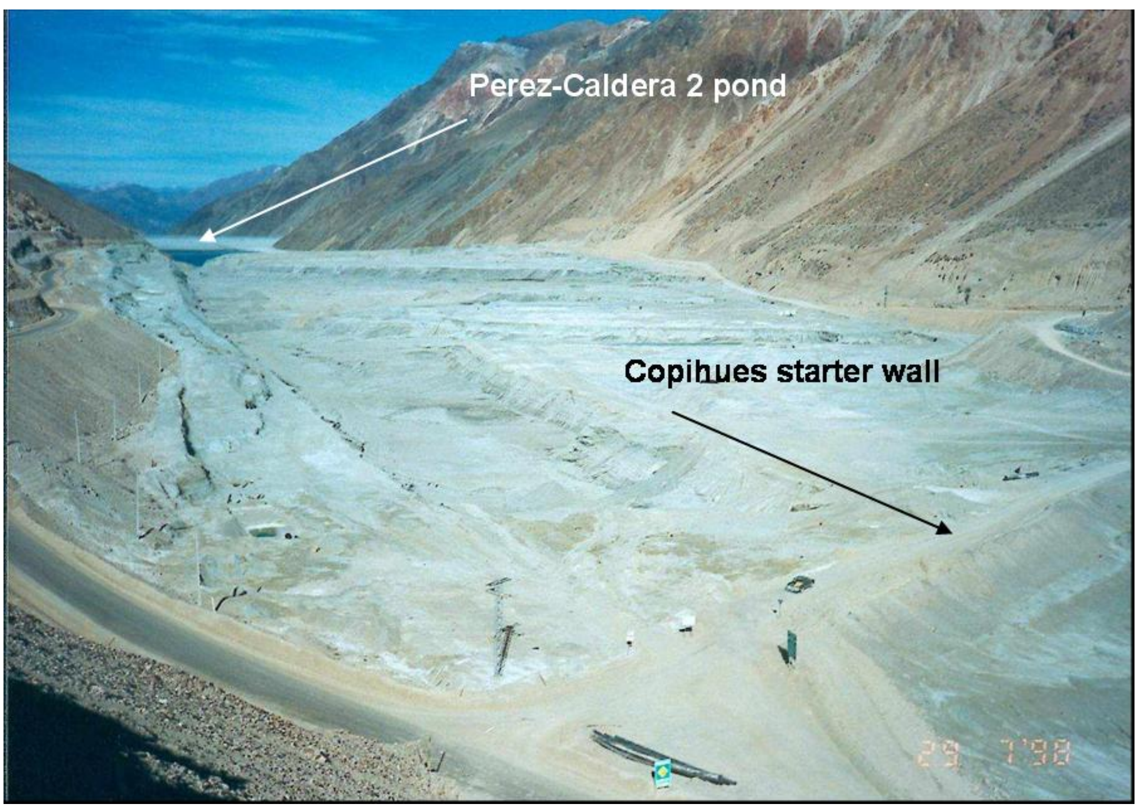

Figure 6.8: Initial mining in PC1 dam (July 1998)

Figure 6.9: June 2004 view of PC1 dam

Through the mining process, the hydraulic head of the slurry flowing to the pump site was progressively being reduced as the tailings level was lowered, forcing the pump site to be moved upstream to reduce the flow distance. Once the pump site reached the toe of the PC2 starter dam, and the tailings were no longer able to flow to the pump site, a new strategy had to be implemented.

In order to keep the main road (which runs around the perimeter of the tailings dams) in working condition, the monitors could not thoroughly clean the slope adjacent to the road and left a series of steps along the slope on which the monitors had to be installed to progress with the cut. Refer to Figure 6.6 and 6.9.

Dredging

The PC2 compartment is currently used as a water storage facility, and no other facility was planned to receive the water from that pond. Since the monitors cannot operate under water, they could not be used to mine the tailings from PC2.

Figure 6.10: Suction dredge with cutter in action

Figure 6.11: Operation of the dredge with monitor assisting. In order to speed up the process, monitors were put in place to enhance the density of the slurry reporting to the dredge. A cyclone station will be installed to thicken the slurry

The dredging plan calls for the dredge to advance parallel to a slope, removing the toe of the slope and promoting its failure. Once the slope has failed/sloughed, the dredge suction pump sucks up the sloughed material and sends it through a pipe to a booster pump station that transports the tailings to the new Las Tortolas dam.

7 Conclusions and Recommendations

The operations summarized in this report, although very different in many ways from the Faro situation, offer some guidance to the design of the Faro Tailings Relocation Project. The most important ideas to come out of the reviewed work are as follows:

1. Submersible pumps are effective but expensive to maintain. The experience at Giant Yellowknife clearly indicates that the cost of maintenance of the Toyo submersible slurry pumps was high. The pumps were very effective in digging their own sumps and they were able to pump slurry well, but they had excessive seal wear and high ongoing maintenance costs. The use of Toyo pumps at ERG in Timmins worked better than Giant as they learned lessons from Giant and did a better job of preventative maintenance. In spite of this, it is advisable to use vertical cantilever pumps vs. submersible pumps.

2. Pumps should be kept as close as possible to the mining face. The ability to move sump pumps close to the mining faces will improve the density of the slurry reporting to the sumps. The economics of the project are very dependent upon being able to maintain a high-solids slurry and therefore every effort should be made to improve the likelihood of keeping the solids in suspension to the sump pumps.

3. Gradients from the mining face to the sump or pump station must provide sufficient slurry velocity to keep the solids in suspension.

4. Final clean-up of tailings on top of original topography is not well suited to high-pressure hydraulic monitoring and is probably best done with low pressure water combined with excavators and trucks. This is the experience at Los Bronces and Elliot lake

5. It is recommended that, if the hydraulic mining of tailings at Faro is further pursued, a trip to Chile and South Africa be arranged to gather more detailed information from major tailings relocation efforts, assuming they are still in operation. South Africa, in particular, contains the vast majority of the hydraulic mining examples in the world although the shutting down of Ergo operations will limit the sites available to visit.

6. If relocation is selected, a large trial field program may be needed to demonstrate the feasibility of hydraulic mining and finalize design details.

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Attachment B
Hydraulic Monitoring

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Technical Memo

To:	Daryl Hockley, Cam Scott	**Date:**	January 17, 2006
cc:		**From:**	Gordon Doerksen
Subject:	Hydraulic Mining Plan	**Project #:**	1CD003.079

1 Hydraulic Monitoring Description

The hydraulic mining system proposed for the Faro operation is designed to move the about 86% (49Mt) of the entire tailings volume located in the tailings basin. The remaining 14% (8Mt) of tailings will be difficult to remove hydraulically and will, therefore, be moved with conventional excavators and articulated haul trucks, as will 4Mt of potentially contaminated original basin floor material.

The hydraulic mining system will consist of two high-pressure water pumps located on a barge in the Faro pit. The pumps will supply water to six operating monitors that slurry the tailings which flows to pumps and is pumped back into the Faro pit where the solids settle out and the water is reused.

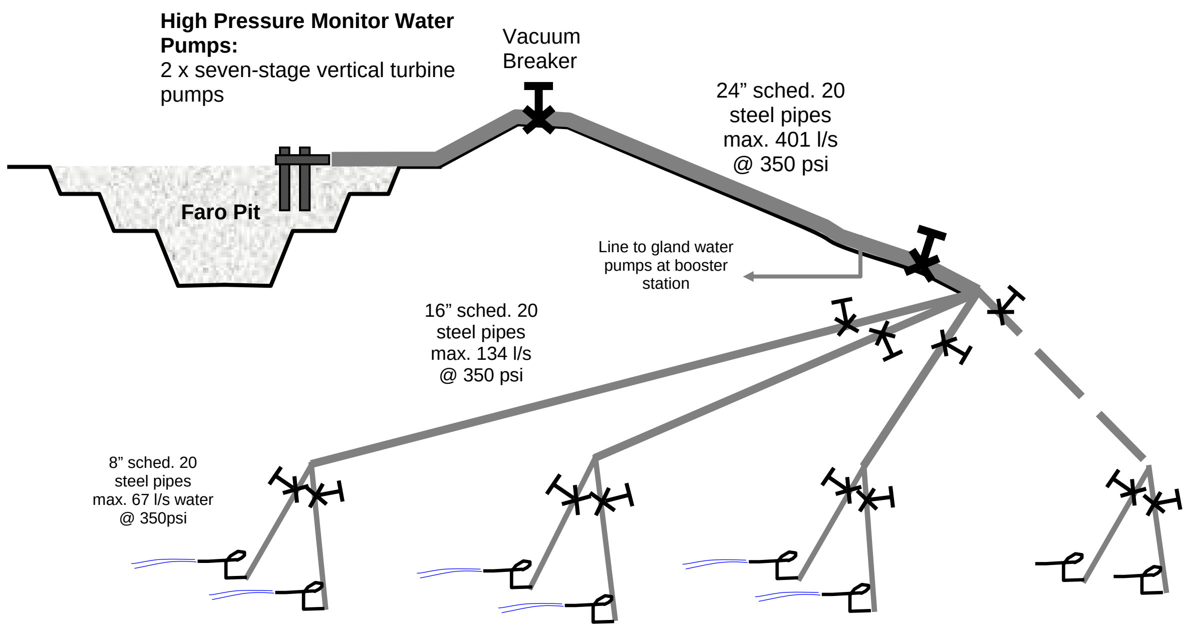

Figure 1.1 Hydraulic mining system

2 Design parameters

The parameters used in the design of the hydraulic mining system are listed in Table 2.1.

Table 2.1 Design parameters

Item	Metric	Imperial
Tonnage (dry solids):	49,000,000 tonnes total	54,014,000 tons
Volume (Bank):	24,500,000 BCM	32,046,000 BCY
Operating period:	12.5 years	
	5 months/yr	
	150 days/yr	
	90% avail. and util.	
	21.6 hours/day	
	3,240 actual op. hrs/yr	
Production: (by tonnage)	4,000,000 tonnes/yr	3,629,000 tons/yr
	26,667 tonnes/day	29,395 tons/day
	1,235 tonnes/hr	1,361 tons/hr
	20.6 tonnes/min	22.7 tons/min
	0.343 tonnes/sec	0.378 tons/sec
Production: (by volume)	2,000,000 BCM/yr	2,616,000 BCY/yr
	13,333 BCM/day	17,440 BCY/day
Dry tailings solids density:	4.0 tonnes/m^3	3.37 tons/yd^3
In situ dry tailings density:	2.0 tonnes/m^3	1.69 tons/yd^3
Slurry % solids by weight:	46%	
Slurry % solids by volume:	17.6%	
Slurry specific gravity:	1.53	
Slurry pumping requirement:	487 l/s	7,720 usgpm
Inherent moisture in tailings:	10% (by weight)	
Monitor water pumping requirement:	401 l/s*	6,356 usgpm*
Water pressure at monitor:	24 bar (250m)	350 psi (800 ft)
Elevation head	8 bar (80m)	116 psi (270 ft)
Monitor water specific gravity:	1.0	1.0

* this is the maximum volume of water required and will vary depending on the amount of moisture in the tailings.

3 High pressure water pumps and pipes

Feed water for the hydraulic monitors will be supplied by two 1,000 horsepower 7-stage vertical turbine pumps operating from a floating barge located in the Faro pit. The pumps will produce 200 l/s (3,200 usgpm) of water each at 237 m (775 feet) of head. It is assumed that the existing barge on site is adequate to support the new pumps. If this is not the case then a new float will be purchased. Three pumps will be purchased to provide one spare to accommodate preventative maintenance and unplanned breakdowns.

The high pressure pumps will be located as far as possible from the slurry discharge point into the pit to allow the greatest opportunity for the solids to settle to keep solids out of the pumps. This separation is important to keep the monitor water as solids-free as possible to prevent wear on pumps, pipes and monitors.

The high pressure pumps will be configured in parallel and feed into one 600mm (24”) schedule 20 steel pipeline leading to the mining area. The profile of the pipeline is such that a vacuum breaker will be installed at the high point in the line to prevent the suctioning of water from the pit when the pumps are shut off. See Figure 3.1 below.

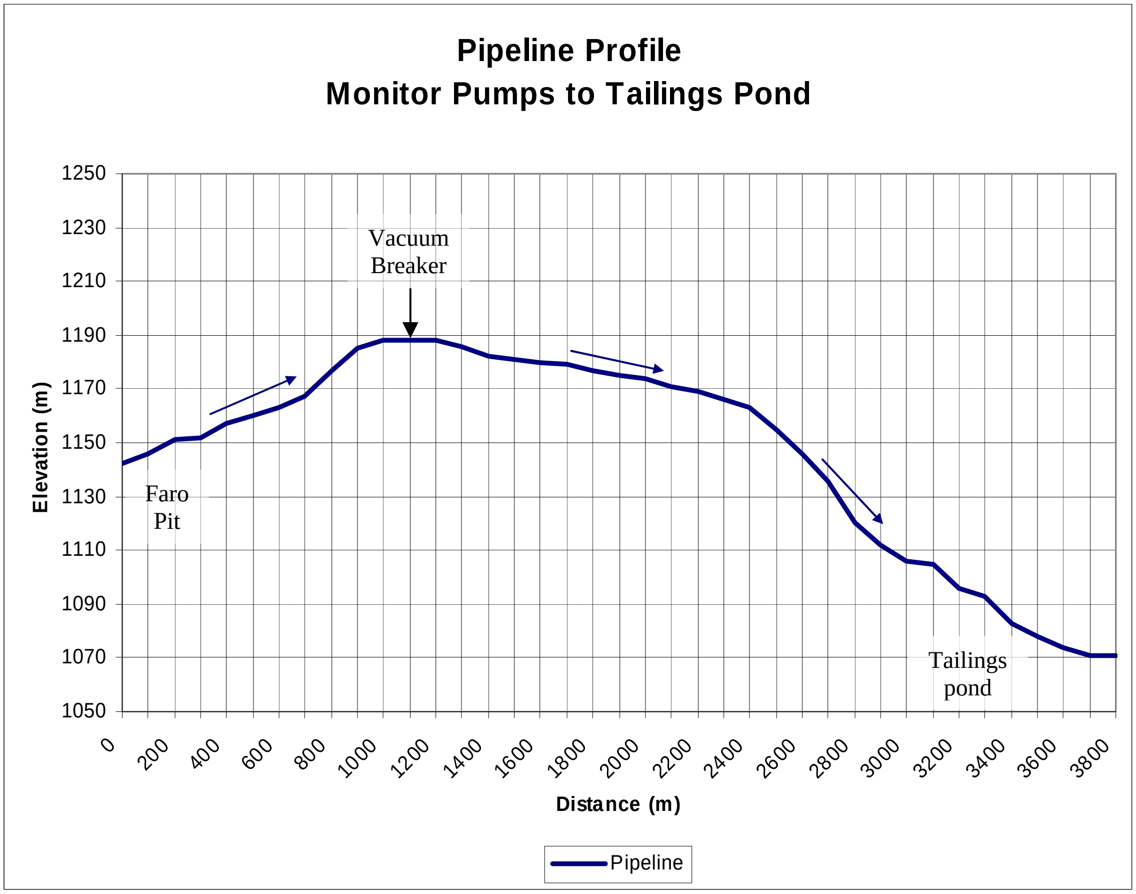

Figure 3.1 600mm diameter monitor water pipeline

The power calculation for the monitor water pump power is as follows:

$$P = \frac{q(H)(s.g.)}{6118(Eff)} \qquad P = \frac{12{,}030(237)(1.00)}{6118(.79)}$$

Where: P = Power for pumping (kilowatts)
q = Flow (litres/minute)
H = Head in metres of liquid (metres)
s.g. = specific gravity of slurry (kilograms/litre)
Eff = pump efficiency expressed as a decimal

P = 590 kW (790h.p.) per pump A 750kW (1000hp) motor will be required for each pump.

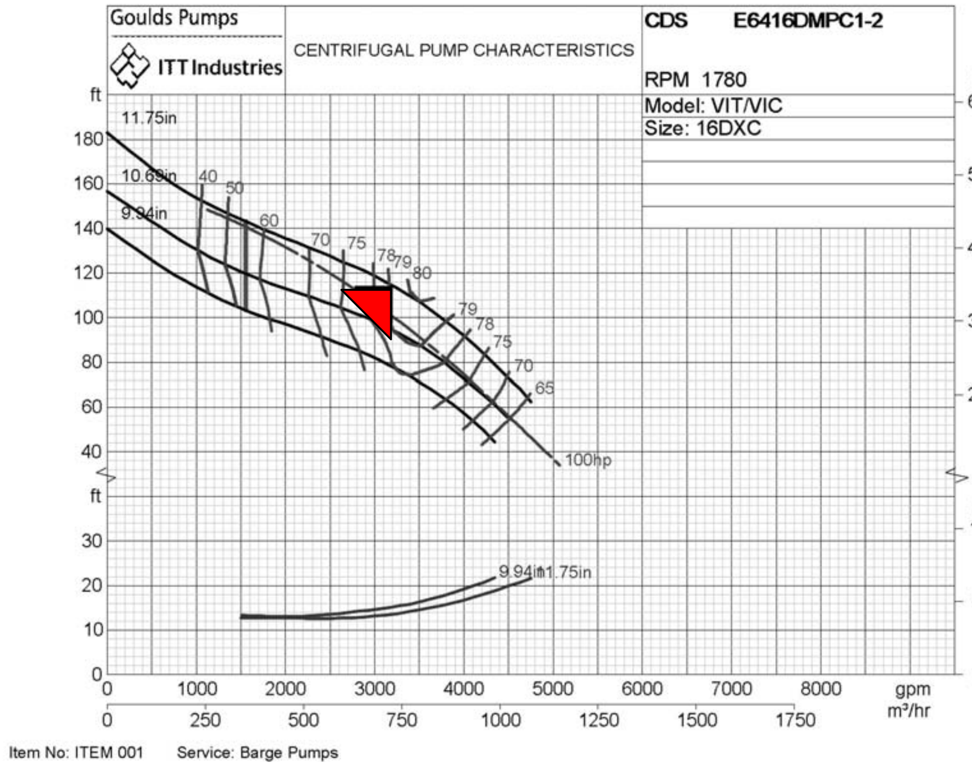

Figure 3.2 High pressure monitor water pump curve for one out of seven stages.

The 600mm high pressure pipeline will be welded together as it will not be moved for the life of the project. Water velocity in the pipeline will be a maximum of 1.4 m/s (4.5 ft/s) at 401 l/s of flow. The flow of the water to the monitors will vary depending on the ease of mining and the amount of inherent water trapped in the tailings.

At the tailing pond, the 600mm pipeline will split into between two and four 400mm (16") schedule 20 steel pipelines, up to 1,500m in length, that distribute the water to various sections of the tailings pond. The 400mm steel pipelines will be coupled together using "Victaulic"-type fittings to allow for ease of relocation of the pipeline. The 400mm pipes will then be split into 200mm (8") schedule 20 steel pipes that feed the hydraulic monitors directly. The final section of pipe prior to the hydraulic monitors may be made out of a high pressure hose to expedite the relocation of the monitors. The 400mm and 200mm pipelines will have a series of isolation valves that will enable dismantling and re-connection of any section of pipe without the affecting the rest of the operation.

A small pipe line will tee off of the main monitor feed water pipeline and for use by the gland water booster pumps. Pressure regulators rather than pumps may be needed in the gland water system to get the correct pressure to the booster pump glands. The monitor feed water will only be used for gland water if it is clean enough otherwise potable water will be used.

4 Monitoring System

There are two main options for the configuration of the hydraulic monitoring system both of which will be used at the Faro operation depending on a number of circumstances. The top slicing method will be employed initially at Faro. Top slicing allows for monitors to be moved around freely on firm, dewatered tailings surfaces as found in the Second Impoundment Area. Top slicing has been used successfully in South Africa, Chile and Timmins, Ontario.

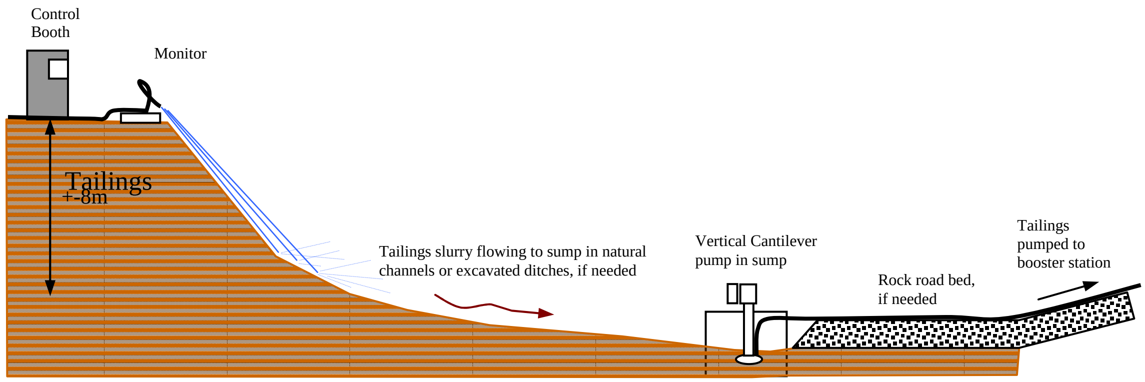

Figure 4.1 Proposed Faro hydraulic mining configuration showing the top slicing method for areas where the top of the tailings pond is firm enough for machine operation like the Second Impoundment Area.

The top slicing method requires an average of 4 monitor movements per day as each monitor only will have about 7,000 tonnes of tailings available from each set-up. See Figure 4.2 below.

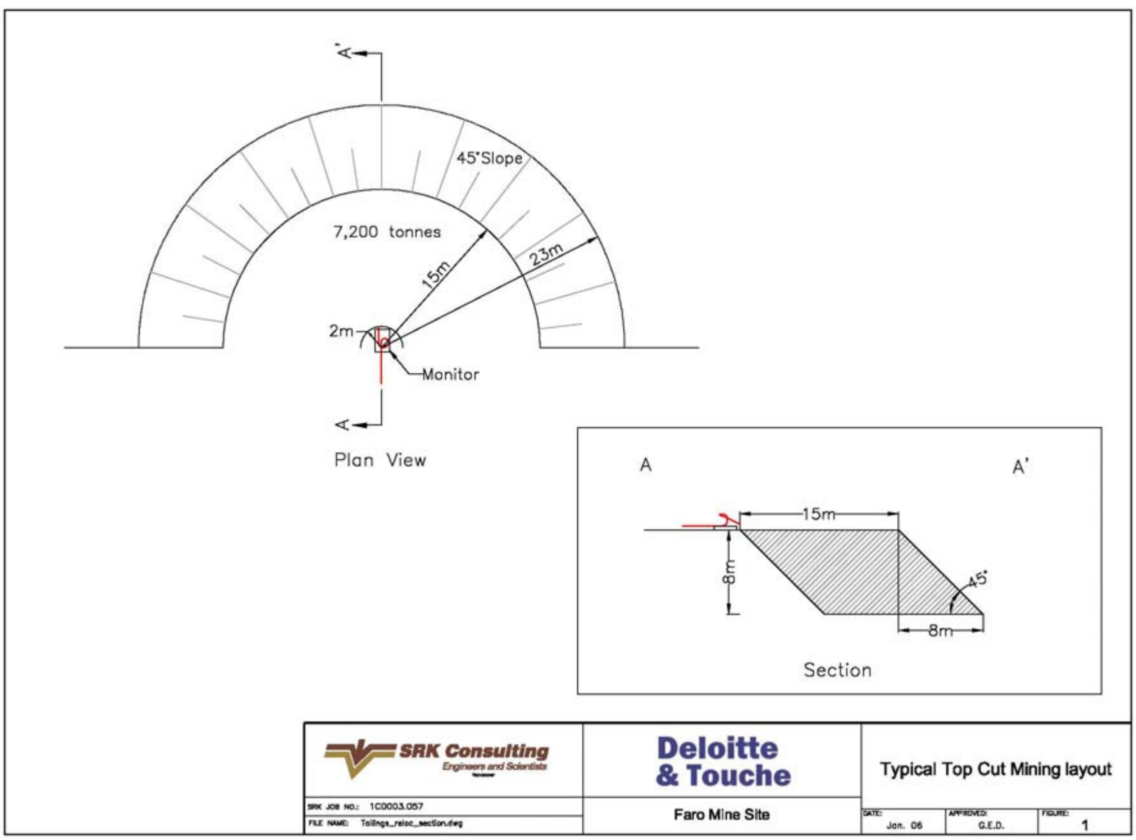

Figure 4.2 Plan and section view of a typical cut using the top cut method. Note that the volume from each set-up is dependent upon the proximity of the monitor to the edge of the bank to be mined.

Undercut mining has been employed in South Africa, Timmins, ON, Elliot Lake, ON and Yellowknife, NT and will be used when the top of the tailings pond is saturated with water to the extent that it is not possible for tracked equipment to work on it. This is very likely going to be the case in the entire Intermediate Impoundment area and the second cut of the Original and Second Impoundment areas. The undercut method has the advantage of being able to mine more material per mining setup (30-40,000 tonnes versus 7,000 tonnes) and the slurry densities may be higher than the top cut method. The disadvantage of the undercut method is that the monitor, control booth and sump pump have to be moved ahead on firm ground. This normally means a waste rock road has to be built, creating a much longer moving times and adding road building and decommissioning costs.

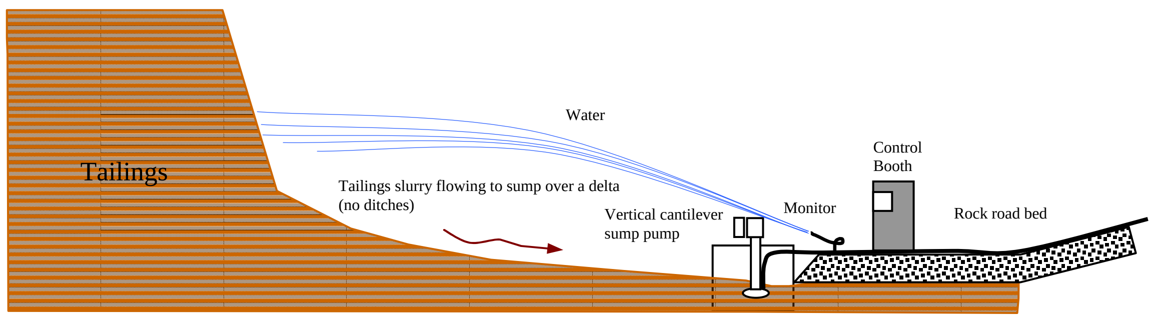

Figure 4.3 Undercutting mining method for areas where top access is not possible due to saturated tailings

Regardless of the hydraulic mining configuration, six monitors will typically be in operation at any given time yielding 4,400 tonnes per monitor per day. The three idle monitors will be moved and set-up for the next cut.

The volume of water coming from any individual monitor will be typically 60-70 l/s (1,100 usgpm) and can be adjusted for mining conditions by changing the size of nozzle on the end of the monitor. Typically 1.25” to 2” diameter nozzles will be used.

5 Labour requirements

Labour will be scheduled to work a 12-hour shift rotation, 4 days on followed by 4 days off. Maintenance personnel will work straight dayshifts and will be on call for night shift breakdowns. Operating personnel will work alternating cycles of dayshift and night shift.

Project management will be done by a Superintendent who is responsible for the entire operation including health and safety, budgets, production, maintenance, training, etc. The Superintendent will work dayshift Monday to Friday.

Supporting the Superintendent will be 4 Working Lead Hands who will work the 12-hour shift 4x4 schedule. The Lead Hands will supervise the operation and carry out the safety, mining and maintenance plans.

Table 5.1 Hydraulic monitoring manpower

Position	Dayshift Complement	Nightshift Complement
Superintendent	1	0
Working Lead Hand	1	1
Electrician	1	0
Millwright	1	0
Monitor operators	3	3
Labourers	2	2
TOTAL	**9**	**6**

6 Mining Plan

Mining is assumed to be conducted using 8-10m high benches in the case of top cutting and 8-15m high faces in the case of undercutting. The height of the benches will be determined largely by the slope stability of the mining faces and the actual thickness of tailings above the original topography.

An important element of the mining plan will be to determine the gradient or slope required to keep the tailings in suspension while they are flowing from the mining face to the sump pumps. Typical gradients seen in other operations are 1% to 2%, however, the Faro solids have a very high specific gravity (4.0 kg/l) and solids may settle out more quickly. This fast settling rate, even of very fine material, has been mentioned by the Saskatchewan Research Council in their initial assessment of the Faro tailings. With these findings, gradients greater than 2% will likely be required to maintain slurry flow velocities that keep tailings suspended. Steeper gradients will mean that sumps will have to be moved more frequently to keep them close to the mining face. The section view in Figure 6.1 shows various slope gradients.

The mining plan is comprised of 5 Phases and is divided into two main impoundment areas. Area 1 comprises the original and second impoundment basins. Area 2 is made up of the tailings in the intermediate and collection basins. See Figure 6.1.

Phase 1: Top cut mining in Area 1 – 1st Cut

Mining will start with the monitors situated on the top if the tailings on the north side of the original impoundment area. The original dam will have a ramp excavated through it at a grade of 12% to a depth 12m to start the initial sump and mining face. The ramp will be excavated with conventional mining equipment (excavator and trucks) and the initial sump will be dug with an excavator. Rock from the dam will be stockpiled adjacent to the tailings pond for future road building use.

Mining will continue steadily outward from the first sump. Once the monitors advance far enough away for slurry flow to be hindered, the sump will be advanced and split so a second and third sump are established. Advancement of the sump pumps for the first cut will be done on top of a layer of waste rock to provide support for mobile equipment. Phase 1 contains about 17 MT of material and will take the first 4+ years to complete.

Phase 2: Undercut mining in Area 1 – 2nd Cut

Phase two will continue with the mining of the final bench in Area 1. It is assumed that the tailings will not be firm enough to drive mobile equipment on so the undercut mining method will be used. Phase 2 contains approximately 9MT of tailings and will be done mainly in Years 5 -7.

Phase 3: Undercut mining in Area 2 – 1st Cut

The top half of Area 2 will be mined in Years 7-10 and comprises 15MT. Because of the saturated nature of the tailings, the undercut method will be used.

Phase 4: Undercut Mining in Area 2 – 2nd Cut

Phase 4 will be to finish the final cut in Area 2 and will take place in years 11 and 12.

Phase 5: Basin Cleanup

Clean-up of the bottom layer of tailings (8MT) immediately above the original topography will done with a fleet of conventional equipment consisting of 40-tonne articulated trucks working with a $4m^3$ hydraulic excavator. 4MT of contaminated original ground will also be removed.

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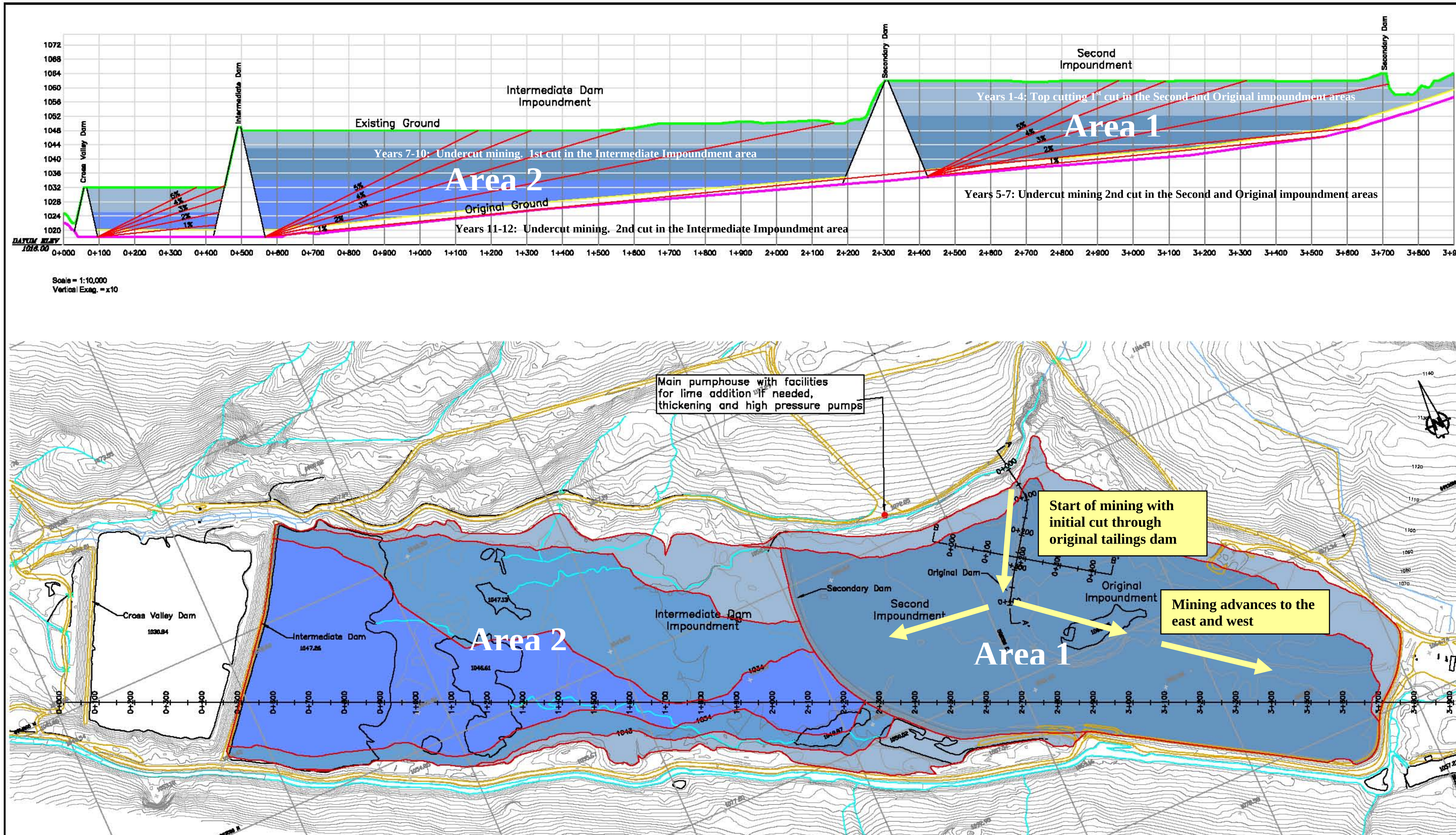

Authors Initials/typist initials

Faro Mine, Yukon

SRK Consulting Engineers and Scientists | Deloitte & Touche | Figure 6.1 Mining Plan

Attachment C
Slurry Handling

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Technical Memo

To:	Daryl Hockley, Cam Scott	**Date:**	January 11, 2006
cc:		**From:**	Gordon Doerksen
Subject:	Faro Hydraulic Mining Option - Slurry Pumping Arrangement	**Project #:**	1CD003.079

1 Introduction

The slurry pumping component of the Faro hydraulic mining operation consists of all pumps, pipes and equipment required to move tailings slurry from the Rose Creek valley to the Faro Pit. The slurry pumping will be handled in two stages. The first stage will be the use of sump pumps to transfer slurry from the tailings pond near the mining faces to the permanent booster pump station. The second stage will be the screening and pumping of tailings slurry from the permanent booster pump station to the Faro Pit. See Figure 1.1 below.

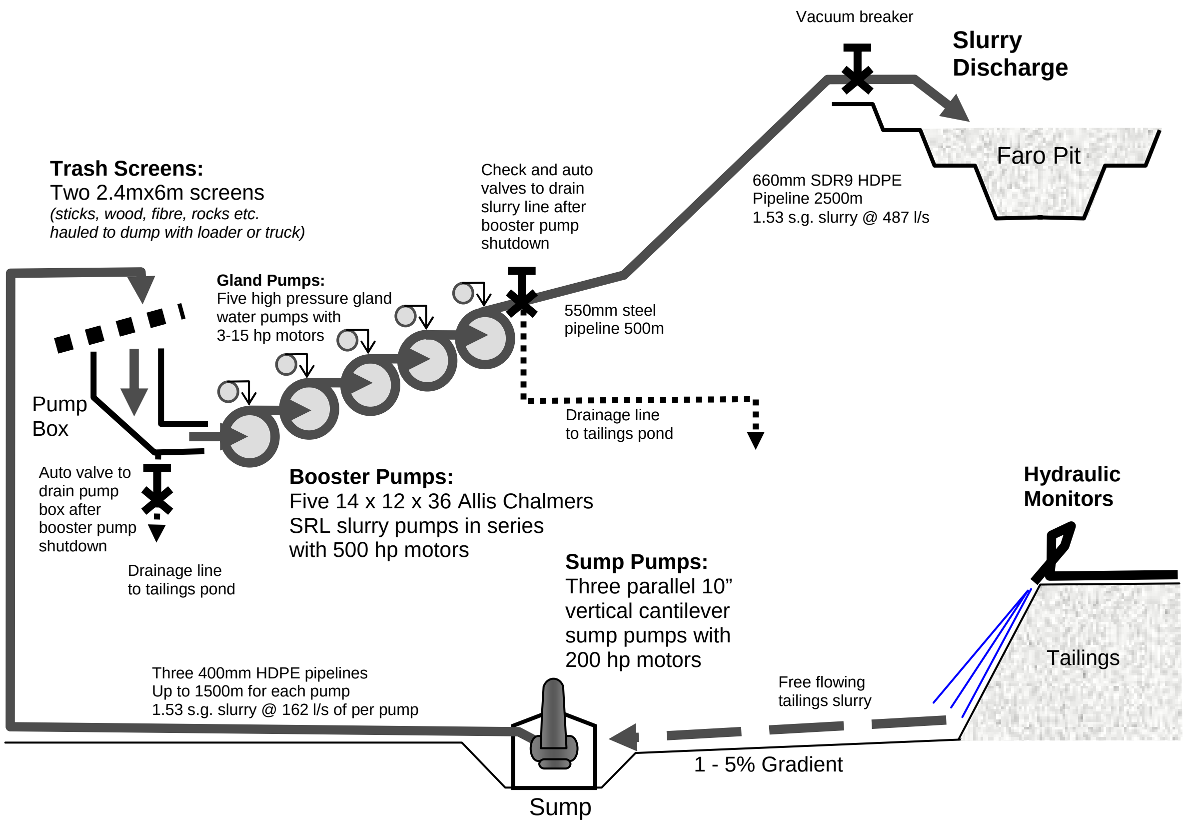

Figure 1.1 Slurry pumping system diagram

2 System parameters

The following parameters were used in the design of the slurry pumping system.

Table 2.1 Slurry pumping design parameters

	Metric	Imperial
Volume (dry solids):	49,000,000 tonnes total	54,014,000 tons
	4,000,000 tonnes/yr	4,409,000 tons
Operating period:	12.5 years	
	5 months/yr	
	150 days/yr	
	90% avail. and util.	
	21.6 hours/day	
	3,240 actual op. hrs/yr	
Production:	4,000,000 tonnes/yr	3,629,000 tons/yr
	26,667 tonnes/day	29,395 tons/day
	1,235 tonnes/hr	1,361 tons/hr
	20.6 tonnes/min	22.7 tons/min
	0.343 tonnes/sec	0.378 tons/sec
Dry tailings solids density:	4.0 tonnes/m^3	3.37 tons/yd^3
In situ dry tailings density:	2.0 tonnes/m^3	1.69 tons/yd^3
Slurry % solids by weight:	46%	
Slurry % solids by volume:	17.6%	
Slurry specific gravity:	1.53	
Slurry pumping requirement:	487 l/s	7,720 usgpm
Monitor water pumping requirement:	401 l/s*	6,356 usgpm*

* This is the maximum volume as it does not take into account water retained in the tailings

3 Sump pumps and pipes

The purpose of the sump pumps is to collect the slurried tailings near the mining face and transfer it to the booster pump station. The sump pumps will be 200 h.p. vertical cantilever pumps with rubber lined impellers. The sump pumps will operate independent of each other and will typically be working with two hydraulic monitors each. There will be three sump pumps operating at once at 162 l/s each at a slurry specific gravity of 1.53. The system will have a total dynamic head of about 30m. There will be one spare sump pump for coverage during maintenance and relocation downtime.

The sump pumps will be supported in a box-like base that will allow them to stand alone in a sump without being constantly suspended by a crane or excavator. See Figure 3.2. The support box will have perforated sides that will allow slurry to flow to the pumps but will keep larger sticks and debris away from the pumps.

The discharge of the sump pumps will connect to a short length (10-20m) of flexible, rubber-lined discharge hose that, in turn, will be connected to 400mm (16”) diameter DR11 high density polyethylene (HDPE) pipe. The HDPE pipe has and internal diameter of 328mm (12.916”) giving a slurry velocity of 1.9m/s (6.3ft/s). The HDPE pipe will carry the slurry to the booster station and deposit the slurry onto a trash screen. Through the life of the project the sump pump pipeline will vary from 200 to 1,500 metres in length.

Power for the sump pumps will be delivered through trailing cables from the booster pump substation. The estimated power draw for the pumps is 113Kw per pump. See Figure 3.1

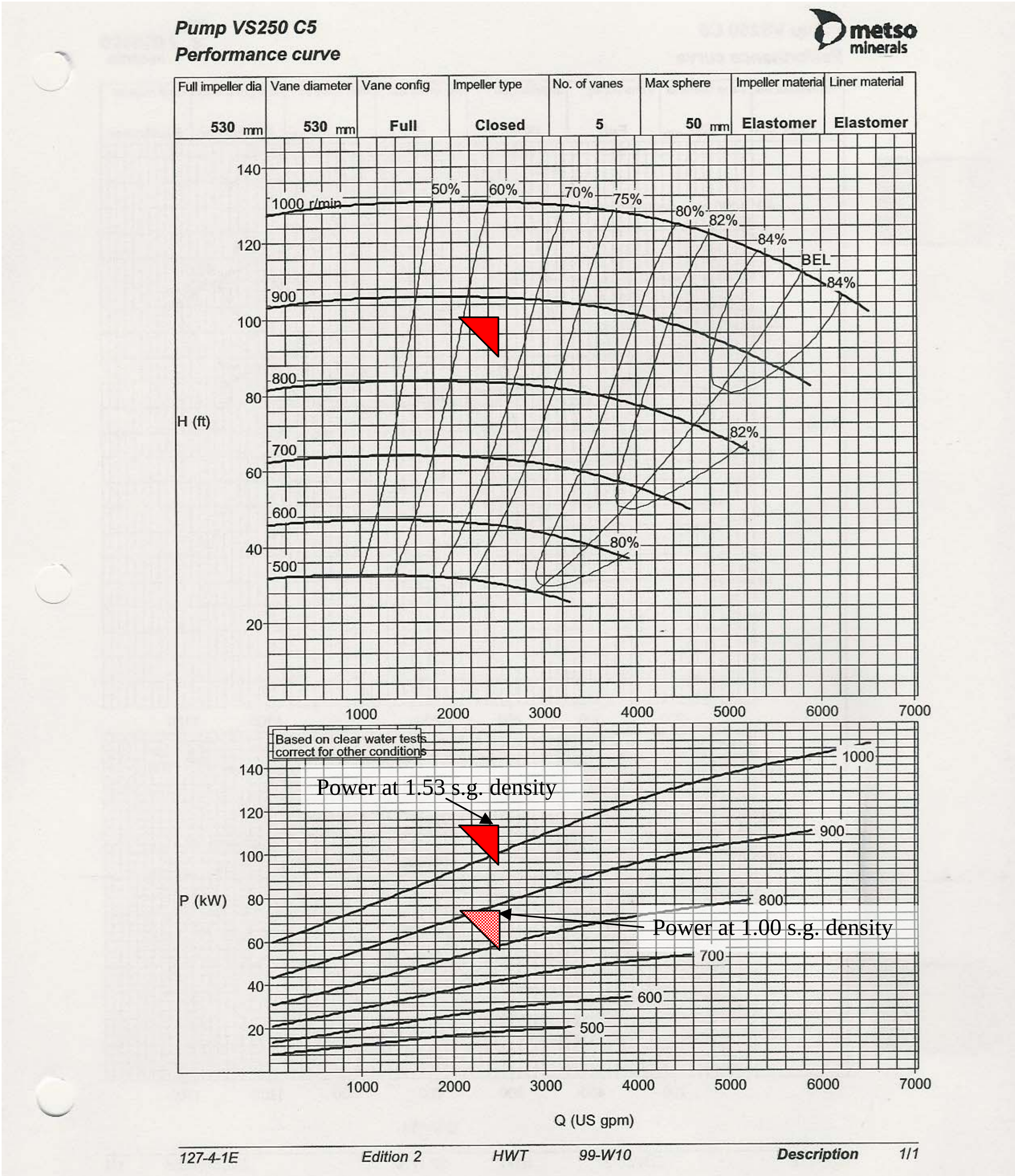

Figure 3.1 Pump curve showing operating point for sump pump

The power calculation for the sump pump power is as follows:

$$P = \frac{q(H)(s.g.)}{6118(Eff)} \qquad P = \frac{9{,}720(30)(1.53)}{6118(.65)}$$

Where:
P = Power for pumping (kilowatts)
q = Flow (litres/minute)
H = Head in metres of liquid (metres)
s.g. = specific gravity of slurry (kilograms/litre)
Eff = pump efficiency expressed as a decimal

P = 113 kW/pump Each sump pump to be fitted with a 150kW (200hp) motor

Figure 3.2 An example of a vertical cantilever sump pump and supporting perforated box being moved with an excavator. (Photo: Denison Environmental Services)

4 Booster pumps and pipes

The purpose of the booster pumps is to transport screened tailings from the booster pump station to the Faro Pit. The booster pump system will be made up of five 14x12x36 horizontal rubber-lined slurry pumps connected in series. Each pump will provide approximately 33m (110 feet) of head pressure to the system giving a total dynamic head of 165m (540 feet). The pumps will be powered by 375 kW (500 h.p.) electric motors and will draw 300 kW each when running. The first booster pump will have a standard casing. Stage two and three pumps will have heavy duty casings and the forth and fifth stage pumps will have extra heavy duty casings to handle the higher pressures. An additional extra heavy duty pump and motor will be purchased and kept on site as a spare unit.

The booster pumps will be operate at a set speed (approximately 520 rpm) except the last pump which will be connected to a variable frequency drive (VFD) system to alter it's speed as needed to match the volume and density of the incoming slurry. The VFD will be electronically linked to a sonic level indicator in the booster pump box and will vary the slurry volume pumped to the pit to maintain a constant level in the booster pump box.

The booster pumps will be programmed to start in series starting with the low pressure pump and will be energized at intervals of a couple of seconds to reduce the peak electrical demand. The five booster pumps have a combined rating of 2,500 connected horsepower, therefore starting them independently is important to reduce demand spikes.

Conventional gland water and packing will be used with individual gland water booster pumps supplied for each slurry pump. The five gland water pumps will be rated at 20 usgpm each and be stepped up in pressure to handle the higher head at each stage.

The pipeline running from the booster pumps to the Faro Pit with be comprised of a 500m length of

500mm diameter lined steel pipe followed by 2500m of 660mm (26”) diameter DR 9 HDPE pipe. The pipeline is made up of two sections to accommodate the higher pressures at the pump discharge end. The internal diameter of the HDPE pipe is 505mm (19.876”) producing a slurry velocity of 2.43m/s (8.0ft/s).

The slurry pipeline to the pit requires a vacuum breaker at the highest elevation of the pipeline. This is necessary to reduce negative pressures in the pipe when the booster pumps are turned off and there is still slurry in the lines. See Figure 5.1. The pipeline will also have a check valve and an automatic drainage valve located at the discharge of the last booster pump. The purpose of these valves is to drain the slurry line automatically upon booster pump shut down. Draining the pipeline is important to ensure the line does not silt up with settled solids. The check valve in the pipeline prevents slurry running back through the pumps, turning them backwards, which can potentially cause damage. A drain valve will also be installed in the bottom of the booster pump box to drain the box and the pumps when the system is off. The slurry from the drainage valves will flow back into the tailings pond.

Pipeline Profile
Main Pump Station to Faro Pit

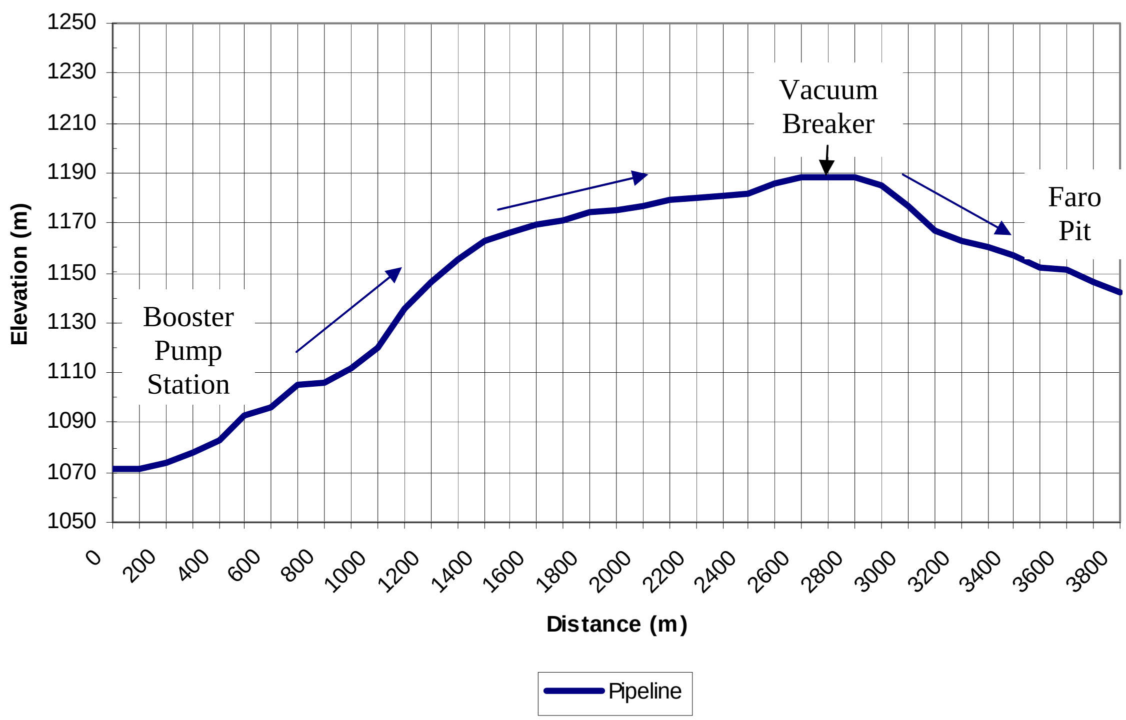

Figure 4.1 Slurry pipeline profile

The power calculation for the booster pump power is as follows:

$$P = \frac{q(H)(s.g.)}{6118(Eff)} \qquad P = \frac{29{,}220(165)(1.53)}{6118(.79)}$$

Where:
P = Power for pumping (kilowatts)
q = Flow (litres/minute)
H = Head in metres of liquid (metres)
s.g. = specific gravity of slurry (kilograms/litre)
Eff = pump efficiency expressed as a decimal

P = 1,500 kW total

P = 300 kW/pump (402 h.p./pump) Therefore each pump to be fitted with a 375kW (500hp) motor

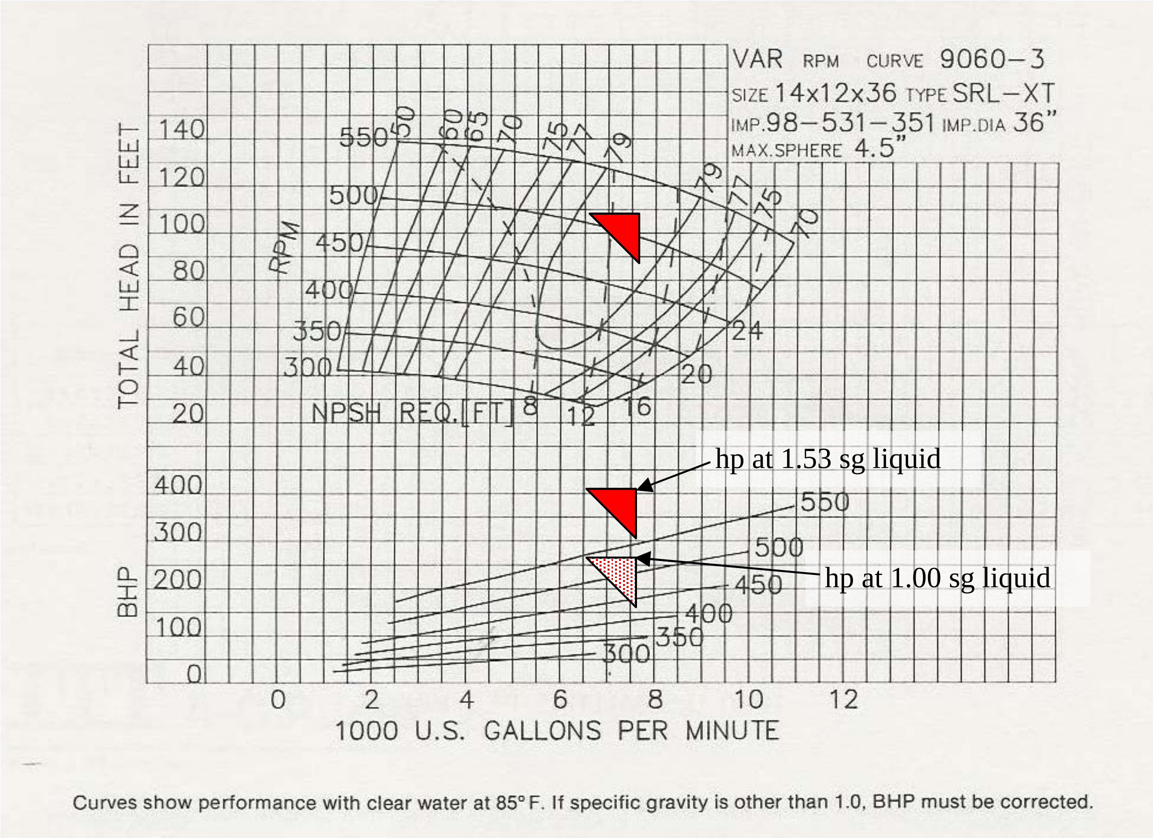

Figure 4.2 Allis-Chalmers 14x12x36 SRL booster pump curve for one of five stages showing operating point

5 Trash screens

The booster pump station will have two 2.4m wide x 6.0m long single-deck trash screens located above the booster pump box. The screen deck will remove any debris that the sump pumps may have picked up in the sumps. The volume of debris will be quite considerable when mining is taking place near the original valley topography, as the sump pumps will pick up sticks, branches and organic matter disturbed by the monitors. The screen deck media will be made of rubber or urethane to provide long life and minimize pegging and blinding of the openings with debris.

The overflow of the screen (trash) will drop into a bin located under the discharge end of the screen deck and will be emptied with a loader. The screen underflow (slurry) will drop directly into the booster pump box.

6 Labour

Labour is scheduled to work a 12-hour shift rotation, 4 days on followed by 4 days off. Maintenance personnel work straight dayshifts and are on call for night shift breakdowns. Operating personnel work alternating cycles of dayshift and night shift.

Management and supervision have been accounted for in the hydraulic monitoring manpower complement and do not show here. It should be noted that labour in the hydraulic monitoring and slurry pumping areas will be interchangeable. The hydraulic monitoring complement consists of 8 operating and maintenance personnel on day shift and 6 operating personnel on night shift in addition to the slurry pumping manpower in Table 6.1 below.

Table 6.1 Slurry pumping manpower

Position	Dayshift Complement	Nightshift Complement
Electrician	1	0
Millwright	1	0
Labourers	2	2
TOTAL	**4**	**2**

7 Electrical power

The electrical power for the slurry pumping system transmitted from a new transformer substation located at the booster pump station. The substation will have the capacity to handle the 2.5 MW of installed power required for the system. See Table 7.1.

The booster substation will be fed with an overhead line from the Faro mill substation.

Table 7.1 Power requirements of slurry pumping system

Component	Installed power motor (h.p.)	Installed power motor (kW)	Estimated power draw (h.p.)	Estimated power draw (kW)
Sump pump #1	200	149	151	113
Sump pump #2	200	149	151	113
Sump pump #3	200	149	151	113
Trash Screen #1	50	37	40	30
Trash Screen #2	50	37	40	30
Gland water pump #1	3	2	2	2
Gland water pump #2	5	4	4	3
Gland water pump #3	8	6	6	4
Gland water pump #4	10	7	8	6
Gland water pump #5	15	11	11	8
Booster pump #1	500	373	402	300
Booster pump #2	500	373	402	300
Booster pump #3	500	373	402	300
Booster pump #4	500	373	402	300
Booster pump #5	500	373	402	300
Misc. lights, controls	25	20	25	20
Total	**3,266**	**2,438**	**2,599**	**1,940**

Attachment D
Lime Addition

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Technical Memorandum

To:	Daryl Hockley; Cam Scott	**Date:**	December 12, 2005
cc:		**From:**	John Chapman
Subject:	Rose Creek Tailings Lime Amendment Costs	**Project #:**	2005/06 Task 22(a)

Objectives

The objective of this memorandum is to summarise the results from the verification lime neutralization tests completed in 2005 and to update the estimate lime neutralization costs for the Rose Creek Tailings.

Sample Selection

Samples selected for the verification testing mirrored the initial sample selection and included additional samples at depth that previously would have been considered to have no lime demand. The composite samples were prepared as shown in Table 1. The locations of the drillholes identified in Table 1 are provided in the attached figure.

Table 1. Composite Sample Selection

Drillhole ID	Impoundment	Interval (m)	Initial Composite	Verification Composite
LA-05-1	Original	0 - 5 5-12	LA Comp 1 -	B1 B2
LA-05-2	Secondary	0 - 5 5 - 10	LA Comp 2 LA Comp3	B3 B4
LA-05-3	Intermediate	0 - 5 7-8	LA Comp 4 LA Comp 5	B5 B6
NA-05-2	Intermediate	0 - 5 5 - 12	NA Comp 1 NA Comp 2	B7 B8
NA-05-5	Secondary	0 - 5 5 - 12	NA Comp 3 -	B9 B10
NA-05-10	Original	0 - 5 5 - 12	NA Comp 4 NA Comp 5	B11 B12

Test Results

Test results are summarised in Table 2. Good reproducibility was observed for the coarse tailings from the original impoundment, which has the highest lime demand. There is some variability in the test results, most noticeably for the samples from the Original Impoundment fines, Secondary Impoundment fines, and the Secondary Impoundment Coarse samples.

Table 2. Lime Demand Test Results

Hole-ID	Location	Zone	Depth (m)	Initial (kg/tonne)	Repeat (kg/tonne)	Average (kg/tonne)
LA-05-1	Original	Fine	0 – 5	29.6	11.8	20.7
			5-12	-	10.0	10.0
NA-05-10	Original	Coarse	0 - 5	45.5	47.8	46.6
			5 - 12	14.4	9.8	12.1
LA-05-2	Secondary	Fine	0 - 5	32.0	21.0	26.5
			5 - 10	9.2	2.0	5.6
NA-05-05	Secondary	Coarse	0 - 5	11.1	20.5	15.8
			5 - 12	-	12.8	12.8
LA-05-3	Intermediate	Fine	0 - 5	7.0	7.5	7.3
			7-8	2.7	1.3	2.0
NA-05-02	Intermediate	Coarse	0 - 5	18.3	15.8	17.0
			5 - 12	3.7	2.6	3.1

These results were used to calculate the overall lime demand as described in the 2004 assessment, by using the isopach estimates of tailings volume in each depth interval. It was further assumed that 30 %, 20 % and 25 % of the tailings in the Original, Secondary and Intermediate Impoundments respectively are coarse.

The results are summarised in Table 3. The lime costs are based on a cost of $373 per tonne of quicklime. As shown in the table, the verification tests indicate a marginally lower overall net lime neutralization cost than the initial tests.

Table 3. Estimated Overall Lime Demand

Description	Original	Secondary	Intermediate	Total
Initial Estimate.	$46,071,000	$50,793,000	$38,276,000	$135,140,000
Verification Tests	$41,579,000	$42,972,000	$24,447,000	$108,998,000
Average	**$43,828,000**	**$46,884,000**	**$31,364,000**	**$122,076,000**

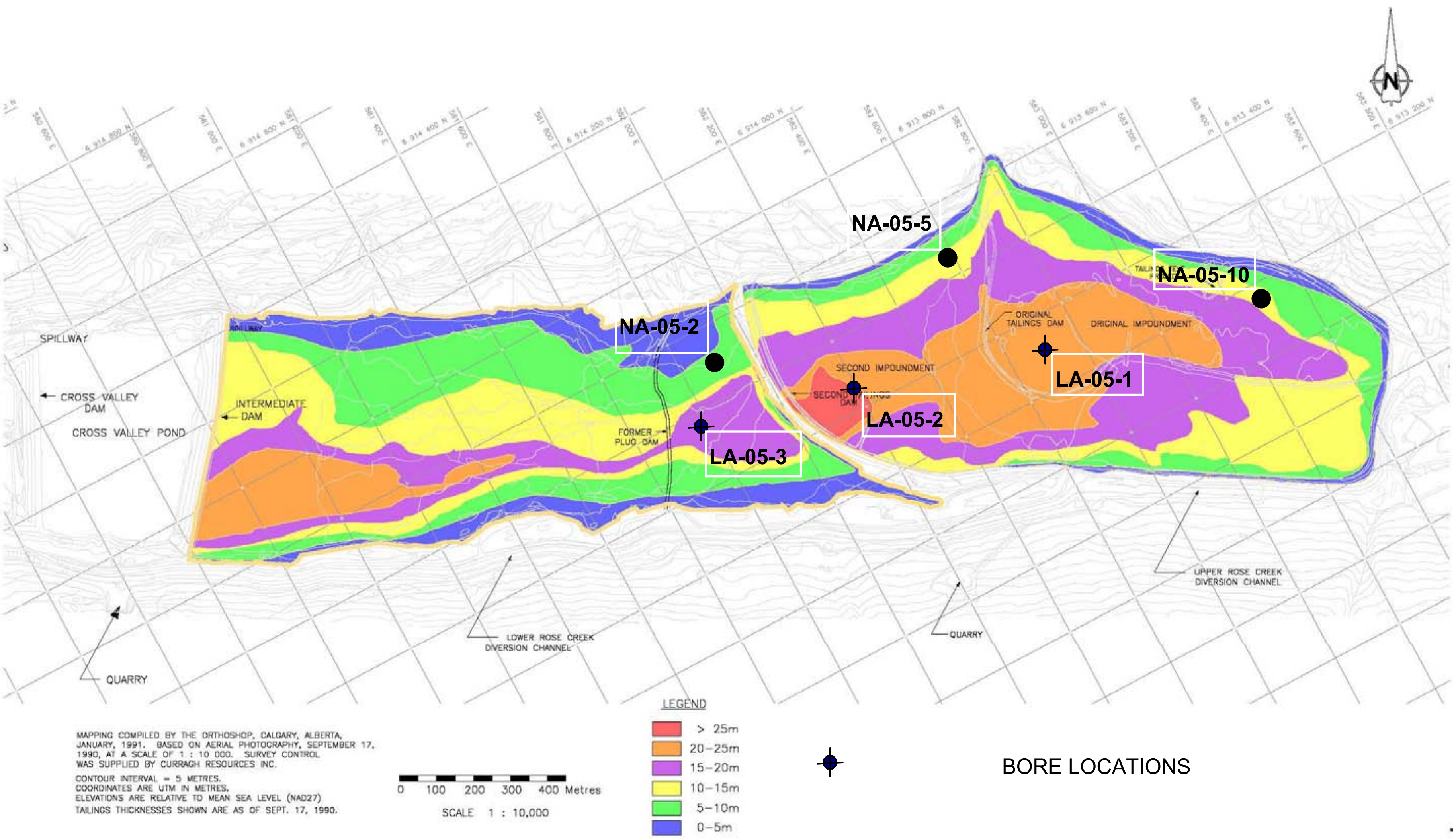

Figure 1. **Bore Locations**

Attachment E
Rheology Testing

Rheology Testing of Faro Tailings Using SRC's 53 mm Flow Loop

Prepared for

SRK Consulting Inc. and
Deloitte & Touche Inc.
on behalf of
Faro Mine Closure Planning Office

by

M. McKibben

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SRC Publication No. 12069-1C06

March 2006

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SUMMARY

Faro tailings have been tested at the SRC's Pipe Flow Technology Centre in Saskatoon for SRK Consulting Inc. (SRK) and Deloitte & Touche Inc. on behalf of the Faro Mine Closure Planning Office. The main test objective was to characterize the rheology of the tailings slurries over a range of solids concentrations. Tests were performed in a 53 mm horizontal pipeline for three slurry concentrations: 60%, 44% and 34% w/w (solids by mass). Slurries were prepared by adding wet solids to a mixed feed tank filled with the process water supplied from the Faro Pit and, at the later stage of testing, with local tap water. Lime slurry was then added to ensure the pH was between 9 and 10. Tests were conducted from high concentration to low by diluting with the supplied water. All experiments were carried out at a constant temperature of 10°C.

For each slurry density examined, pressure gradients were measured over a range of pipeline velocities. Particle deposition was monitored using a transparent observation section. Vertical concentration profiles were collected at a velocity of 1.5 m/s for each mixture. These profiles give an indication of the degree of particle segregation in the pipeline.

For slurries that behave as settling or heterogeneous mixtures, the two layer model is often appropriate for modelling purposes. Although the Faro tailings would be classified in this manner, some of the tailings properties are outside the database range for which the SRC two layer model (Pipeflow 2005) was developed. It was therefore necessary to verify the reliability of this approach by comparing model predictions (pressure gradients and deposit velocities) with those actually measured in the 53 mm pipeline. This analysis indicated that the SRC model had to be used with two different sets of inputs to get reliable predictions of both pipeline pressure gradient and deposit velocity. Predictions were within a few percent with the exception of the deposit velocity prediction for the 60% w/w slurry which was roughly 10% low. Based on these results, SRC's version of the two layer model was deemed suitable for preliminary estimates of the behaviour of Faro tailings slurries in larger pipe diameters. Scale-up predictions for a 46% by mass slurry in 328 mm and 505 mm ID pipelines have been included in this report.

EQUIPMENT

53 mm Pipeline Flow Loop

The layout of SRC's 53 mm diameter pipeline is shown in Figure 1. The loop contains the following items of equipment:

1. A 3x2 centrifugal pump with variable frequency motor control to adjust the flow rate.
2. A 1.5 m^3 feed/mixing tank fitted with a ⅓ hp mixer.
3. A straight, horizontal test section (L=4.88 m, D=53.1 mm) where the frictional pressure gradient could be measured using an electronic pressure transducer.
4. A glass observation section for observing solids deposition.
5. An electromagnetic flowmeter for determining mixture flow rates.
6. A thermistor for monitoring the slurry temperature.
7. A double pipe heat exchanger section which utilizes glycol to maintain the desired mixture temperature.
8. A traversing gamma ray densitometer which sends a narrow, horizontal beam of gamma radiation through the pipe to determine the solids concentration (average across each chord) as a function of vertical position.
9. A full-stream recirculation/discharge valve used to collect samples for particle size and density analyses.

This pipeloop has an internal volume of 105 L. From pressure gradient versus velocity measurements for clear water, the effective roughness of the pipe wall was determined to be fairly smooth at 7 μm in the test section. Wall roughness affects the frictional losses in turbulent pipe flow. Commercial steel is typically about 45 μm but slurry flow lines can be smoother due to the polishing effect of the particles on the pipe wall.

The gamma ray densitometer was used to obtain absorption measurements at ten increments in the vertical direction. These provided information on the degree of solids segregation in the line. The densitometer can also be used to monitor the solids concentration as a function of velocity at y/D=0.05, where y is distance measured up from the bottom of the pipe. This is done to aid in determining when a stationary bed of particles forms under conditions where visual observations in the transparent section of pipe become difficult (ie. fine particles coating the pipe wall).

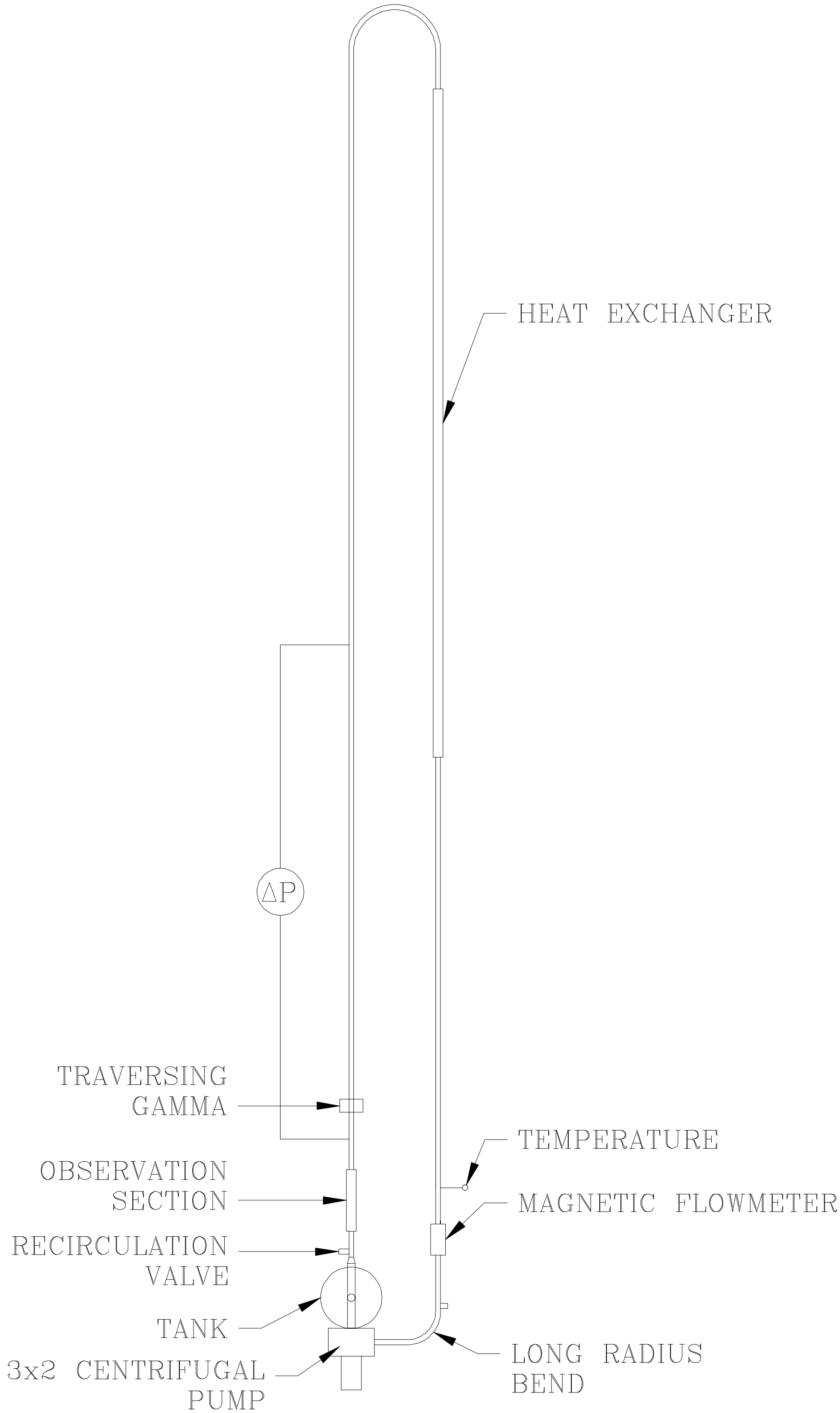

Figure 1. Plan view of SRC's 53 mm pipeline flow loop.

TEST PROCEDURES

The Faro tailings arrived as 78 individually packaged samples ranging in size from 0.7 kg to 4.0 kg. Each sample was labelled as to location and depth of removal from the tailings impoundment. The locations and sample weights, as supplied by SRK, have been included in this report in the Data Appendix, Figure A1 and Table A1. Some of the samples were fairly wet while others were quite dry. The total combined wet weight (excluding sample bags) was measured to be 138 kg. To achieve a concentration of 60% w/w solids, approximately 115 kg of dry solids was required. Therefore, all of the supplied (wet) material was blended to create the slurry for loop testing.

The wet solids were added to a known amount of process water in the feed tank with the mixer running. Lime slurry was then prepared according to the supplied instructions at a ratio of 100 g CaO (vertical pulverized quicklime from Mississippi Lime) to 1 L water. This mixture was then slowly added to the stirred slurry in the mixing tank until the pH was between 9 and 10 (9.5 was considered ideal). The pH was then allowed to stabilize for a few hours before testing. In total, 7.0 L of lime slurry was used to neutralize the 138 kg of wet solids in 70 L of process water. The stabilized pH was measured to be 9.8. The slurry was then charged to the pipe loop. The target concentration for the first test was 55-60% w/w; the achieved concentration was determined to be 60% w/w.

To attain the lower slurry concentrations, additional process water was added to the pipeline through the feed tank while discharging a predetermined volume of concentrated slurry through the discharge valve. The pH was readjusted as necessary to ensure it fell between 9 and 10 for all runs. As there was insufficient process water available for the final dilution, City of Saskatoon water was substituted. This was not unreasonable given that the fines component of these mixtures was very dense and settled rapidly. The use of process water is more critical when clay particles are present (ie. bentonite, illite, montmorillonite, all which have a much lower specific gravity of roughly 2.7) as these particles do not settle but are active in the fluid phase. Their presence can dramatically affect the properties of the liquid phase (ie. viscosity) with notable dependency on the water pH and certain ion concentrations.

Throughout the test program, the mixture temperature was maintained at 10°C. Instrument readings were collected over a range of flow rates using InstraNet model 100 data acquisition boards and a computer. Pipeline pressure gradients were collected as a function of velocity which was incrementally decreased until a stationary bed was observed in the transparent section of pipe. Collecting data below this velocity is not appropriate in a recirculating pipe loop as the flowing concentration becomes a variable once particles become stationary on the pipe bottom.

Density profiles in the vertical direction were measured at a selected velocity (1.5 m/s) for each concentration examined. This provided the in-situ solids concentration as well as information on the degree of particle segregation at this velocity. Pipeline samples were also collected at the beginning of the test program for size and density analyses.

RESULTS

The three concentrations examined were 60%, 44% and 34% solids by mass. Although in-situ concentrations were measured volumetrically (v/v), these were converted to mass concentrations (w/w) using the solids density. The density of the blended solids was determined by adding a known volume of water to a known mass of dry solids to reach a set volume in an accurately calibrated volumetric flask (the mixture was de-aired by vacuum). The volume occupied by the solids could then be calculated to give an average solids density or specific gravity. Four such measurements were made with resulting values ranging from 3.80 to 3.89 g/cc to give an average solids density of 3.84 g/cc (tonnes/ m^3). Table 1 provides a summary of specific gravity data from previous investigations.

Table 1. Specific gravity (Gs) data from previous investigations (provided by SRK).

Source	Gs range	Gs mean	Comments
Original Impoundment, SRK, 1991	4.19 - 4.53	4.5	
Second Impoundment, SRK, 1991	3.66 - 4.08	3.8	
Intermediate Impoundment, SRK, 1991	3.66 - 4.04	3.9	
Total Tailings Impoundment, SRK, 1991		4.0	weighted mean, taking into account tonnage per impoundment
Fine tailings, Golder, 2004	4.008 - 4.300	4.154	
Coarse tailings, Golder, 2004	3.720 - 4.168	3.944	
Total Tailings Impoundment, Golder, 2004		4.0	average based on the mean of each of the fine & coarse fractions
Oxidized surface tailings, SRK, 2003	2.68 - 3.46	2.8	representative of only the shallow oxidized tailings

Particle size analysis of the combined samples was conducted using a representative pipeline sample (full stream discharge) taken at the beginning of the test program. The particle size distribution is located in the Data Appendix, Table A2, and shown in a figure directly below the data. The mass median particle size (d_{50}) of the tailings sample was found to be approximately 40 µm. For modelling purposes, the d_{50} of the coarse tailings (those greater than 44 µm) is often required. This was estimated to be 100 µm.

Pipeline Pressure Gradients

The pressure gradient data is presented in Table A3 along with measured pH values for each concentration. Measurements were collected with decreasing velocity until a stationary deposit was observed in the transparent observation section. Figure 2 shows the pipeline pressure gradients as a function of velocity for the three mixtures examined. The curve for water flow (20ºC) is also shown for comparative purposes.

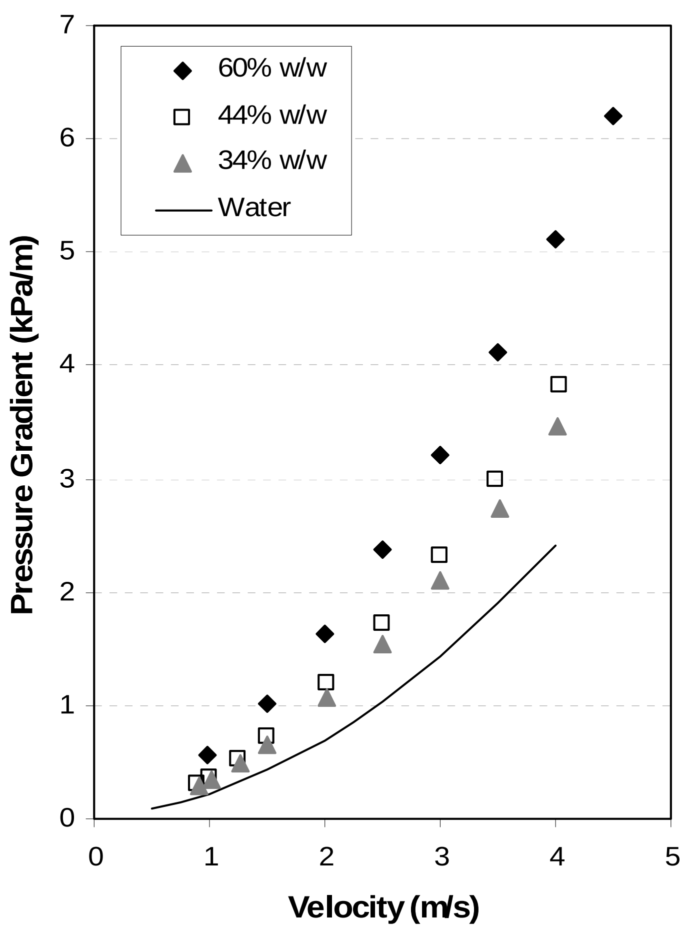

Figure 2. Pressure gradient data for Faro tailings in the 53 mm pipeline at 10°C.

Particle Deposition

Solids deposition was monitored visually in the transparent observation section. The velocity at which a stationary deposit begins to form is the final data point collected and is indicated with an asterisk in Table A3. At 60% w/w solids, the deposit velocity was measured to be 1.0 m/s while at 44% and 34% solids, it was slightly lower at 0.9 m/s. Although some slurry pipelines operate with a stationary deposit, the deposit velocity represents the minimum recommended velocity for pipeline design. Operating below this point can cause unstable flow situations and difficult pipeline control.

In the turbulent flow regime, the presence of turbulent eddies contributes to particle suspension and the velocity at which solids begin to deposit can often be predicted by the correlations described by Gillies et al. (R.G.Gillies, J.Schaan, R.J. Sumner, M.J. McKibben,and C.A.Shook, Deposition Velocities for Newtonian Slurries in Turbulent Flow, Can. J. Chem. Eng., 704-708, 2000). These correlations have been incorporated into SRC's settling slurry model and their applicability to the Faro tailings will be discussed in the section titled Slurry Modelling.

Concentration Profiles

Concentration profiles were determined at a bulk velocity of 1.5 m/s for each slurry examined. This velocity was selected as it was somewhat above deposition, a typical operating point for settling slurries. Chord averaged volumetric concentrations were collected as a function of y/D where y/D = 0 is the pipe bottom and y/D = 1 is the top. Values at the two extreme positions (0.05 and 0.95) should be interpreted with caution due to the increased measurement sensitivity to position at these locations.

Concentration data is located in Table A4. In-situ slurry concentrations were calculated from these measurements and the known chord lengths and were determined to be 60%, 44% and 34% w/w for the three mixtures examined. Figure 3 shows the volumetric concentration profiles. Although these tailings are quite dense, they are also very fine and the profiles taken at 1.5 m/s show little particle segregation with a slight increase in particle density near the pipe bottom.

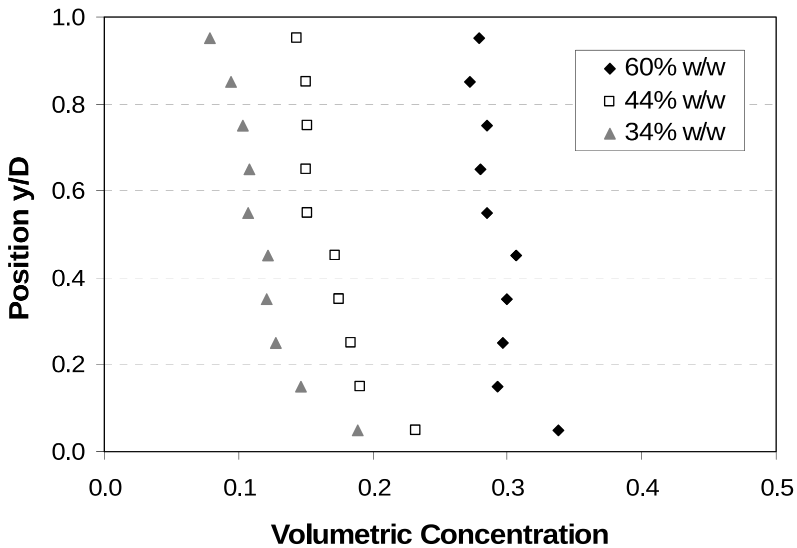

Figure 3. Concentration profiles for Faro tailings at 1.5 m/s in the 53 mm pipeline at 10ºC.

Slurry Modelling

For "settling" or heterogeneous slurries, the two layer model is often appropriate. This is a mechanistic model meaning it is based as much as possible on known physical laws. Since somewhat different simplifying assumptions have been used during its evolution, the model must be regarded as a dynamic entity, evolving as knowledge progresses. The approach seems to have originated with Newitt et al. as a coarse particle pressure gradient equation (Newitt, D.M., Richardson, J.F., Abbott, M. and Turtle, R.B., Hydraulic Conveying of Solids in Horizontal Pipes, Trans. Inst. Chem. Engrs., Vol. 33, 93-113, 1955). The derivation was extended and made rigorous by K.C. Wilson in a series of papers between 1970 and 1976 (Wilson, K.C., Slip Point of Beds In Solid Liquid Pipeline Flow, Proc. ASCE, J.Hyd. Div., Vol. 96, 1-12, 1970 and Wilson, K.C., A Unified Physically Based Analysis of Solid Liquid Pipeline Flow, Proc. Hydrotransport 4, BHRA Cranfield, U.K., Paper A1, 1-16, 1976). The SRC contribution has included its extension to fine particles and high concentrations and has incorporated a broad range of experimental data obtained with pipes ranging up to 495 mm in diameter.

As Faro tailings have properties which are outside the range of the database from which the model has been created, pipeline experiments were necessary to either ensure the model was suitable or to indicate that model improvements were necessary to allow the inclusion of these mixtures. For the majority of slurries tested at SRC, the fine particles (< 44 µm) are clays and are considered part of the fluid phase and contribute to the "carrier fluid" density and viscosity, the latter of which is determined by testing the fines plus water in a concentric cylinder viscometer. Although their inclusion in the "carrier fluid" is accounted for in the two layer model, the fine particles do not contribute to frictional losses in the same way that coarser particles do. Unfortunately, the fine particles in the Faro tailings are very dense and settle rapidly in the fluid phase, making viscometer testing impossible. Therefore, the slurry needed to be tested in a pipeline flow loop. The contribution made by the fine particles could then be determined from experimental results.

The gathered Faro tailings data was examined using the most recent version of the two layer model, SRC's Pipeflow 2005, to compare predicted pressure gradients and deposit velocities with those actually measured in the 53 mm pipeline. In a typical modeling endeavor, Pipeflow 2005 requires inputs which include the fines fraction (fraction of particles less than 44 μm - used to calculated carrier fluid density) along with carrier fluid viscosity and the mass mean particle size or d_{50} of the coarse (plus 44 μm) particles, measured in this case to be 100μm (Table A1). Using these inputs resulted in fairly accurate deposit velocity predictions (±0.1 m/s) but frictional pressure loss predictions that were considerably lower than measured values. This was not surprising given that the Faro fine particles were much denser than the majority of slurries the model has been based on. To check if these fine particles are contributing to the pipeline friction in the same way the coarse particles do, it was necessary to input the d_{50} of the total tailings (40 μm) and to consider the fines fraction to be zero. In other words, all particles were considered to behave as coarse particles. When this was done, the frictional losses were very similar to those measured. The comparison is shown in Figure 4 for the three slurry concentrations examined.

Pipeflow 2005 inputs for experimental comparisons:

Pipe wall roughness	7 microns (measured pre-test)
Pipe inclination	0º
Volume fraction in settled bed*	0.62 (measured at SRC)
Particle density	3840 kg/m^3 (measured at SRC)
Water density and viscosity at 10ºC	

*A measure of the packing density in a stationary deposit. This was determined using the traversing densitometer at a position of y/D=0.05 under no flow conditions (velocity=0).

Unfortunately, the deposit velocity predictions using this approach were low. Again, this was not unexpected as the correlations used to predict this velocity are very particle size dependent. It is the coarser particles that drop out of the flow first and therefore, using the d_{50} of the coarse fraction would be a much better indicator of when these particles would begin to deposit.

Based on these results, using the two layer model to estimate flow behaviour in larger diameter pipelines appears to be a reasonable approach. However, the model needs to be run twice for each condition examined, once using the total tailings d_{50} to predict pressure losses and once using the coarse particle d_{50} to estimate the deposit velocity. It must be cautioned that any estimates made for larger diameter pipelines be considered preliminary, especially if the pipe diameter of interest is significantly larger than the experimental loop diameter for which the model was verified.

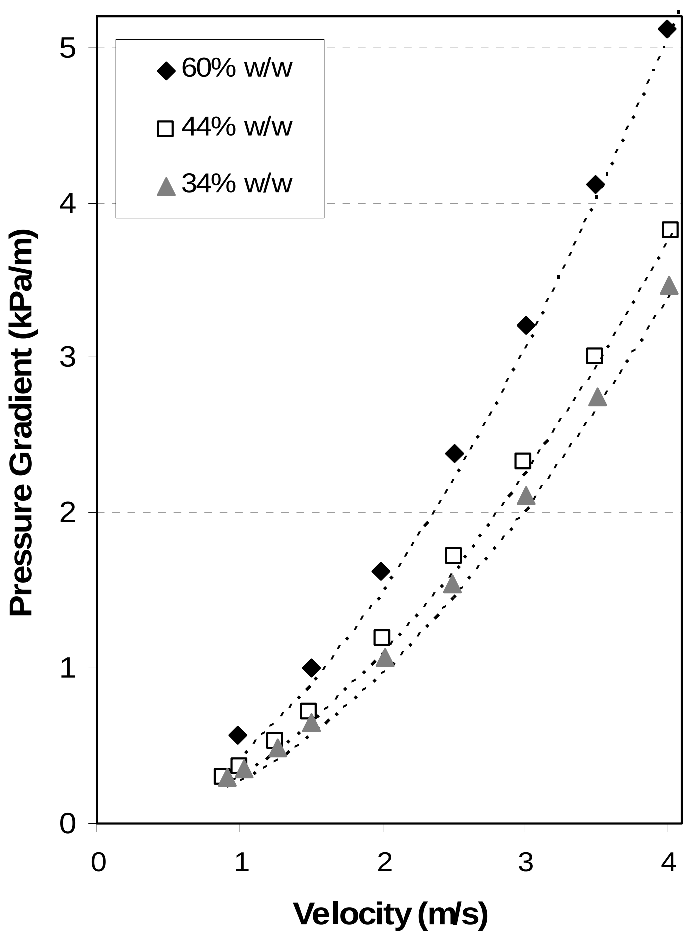

Figure 4. Pipeline data from 53 mm flow loop along with model predictions (dashed lines).

Approximations of the frictional pressure gradient were made for two larger pipe diameters, 328 and 505 mm, at the design concentration of 46% w/w Faro tailings using SRC's PipeFlow 2005 along with the following program inputs:

Pipe wall roughness	45 microns (typical value for mild steel)
Pipe inclination	0º
Volume fraction in settled bed	0.62 (measured at SRC)
Particle density	3840 kg/m^3 (measured at SRC)
Mass mean particle diameter	40 microns (measured at SRC)
Fines fraction	0
Water density and viscosity at 10ºC	

For deposit velocity estimations only, the following inputs were modified and the model rerun:

Mass mean particle diameter	100 microns (measured at SRC)
Fines fraction	0.536 (measured at SRC, Table A1)

Frictional pressure losses for a horizontal pipe are shown in Figure 5 for both pipe diameters. The deposit velocity was the same for both diameters at 1.1 m/s for a concentration of 46% w/w. This velocity was very similar to that measured in the 53 mm pipeline (0.9 m/s). These results demonstrate that deposit velocities for fine particle slurries like the Faro tailings are fairly insensitive to pipe diameter.

As mentioned above, for fine particle slurries, the deposit velocity is fairly insensitive to pipe diameter and, relatively speaking, quite low. However, because the Faro tailings deposit is large (28.6 million m^3) and was deposited hydraulically, variations in particle diameter undoubtedly exist from one location to the next (recall that these tests were conducted on a blend of solids taken from numerous locations in the tailings impoundment). To demonstrate the significant effect of particle diameter, deposit velocity predictions have also been made for a range of coarse particle d_{50} values in the two pipe diameters of interest. The results are located in the table below. As the particle size increases, there is a dramatic increase in the velocity at which deposition occurs. Frictional pressure losses have not been included in this "what if" analysis as they are not nearly as sensitive to variations in d_{50} for the slurry concentration examined here.

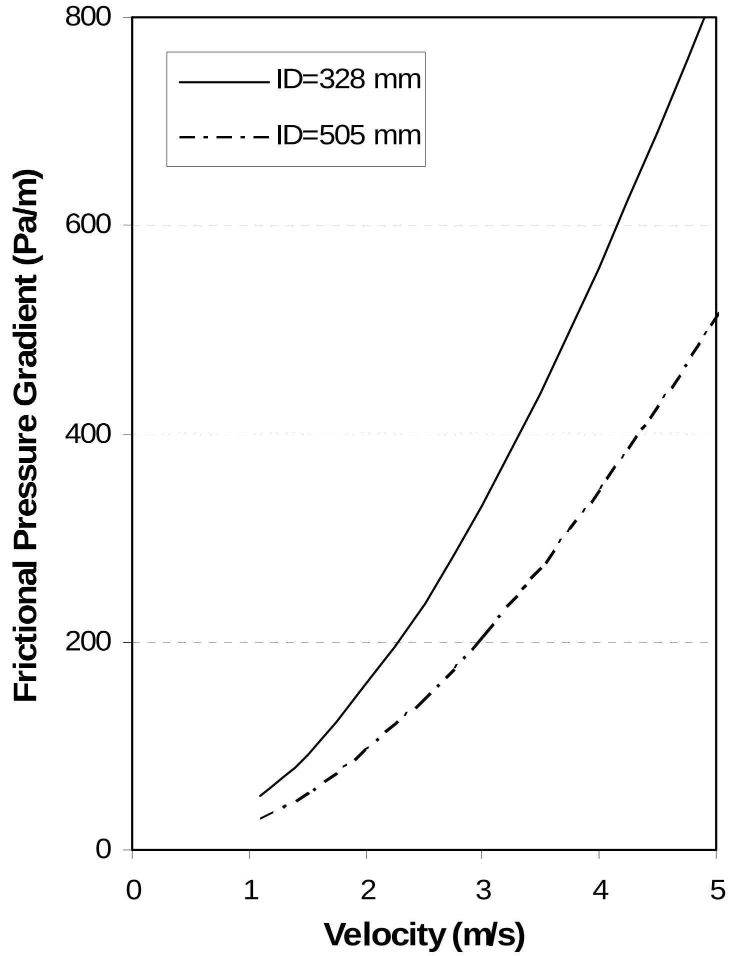

Figure 5. Frictional pressure loss predictions for larger pipe diameters (46% w/w, 10ºC).

Table 2. Deposit velocity predictions for coarser particles (46% w/w solids).

Pipe Diameter (mm)	$d_{50\ coarse}$ (mm)	Deposit Velocity (m/s)
0.328	0.10	1.1
	0.12	1.5
	0.15	2.7
	0.18	3.8
0.505	0.10	1.1
	0.12	1.7
	0.15	3.3
	0.18	4.7

CONCLUSIONS

Faro tailings slurries have been tested in a 53 mm diameter pipeline flow loop for solids concentrations of 34%, 44% and 60% solids by mass. Pipeline pressure gradient curves and deposit velocities were determined for each concentration examined.

In general, these slurries behaved like settling mixtures which could be adequately represented using SRC's modified two layer model. However, it was necessary to run the model twice with different particle size inputs in order to accurately predict both the frictional losses and the velocity at which a deposit began to form.

Predictions were made for two larger pipe diameters of 328 mm and 505 mm for the design slurry concentration of 46% w/w. These results have been included in this report along with a demonstration of the significant effect of particle diameter on the deposit velocity for these two pipe sizes. The latter was included to show that the variation in tailings properties throughout the tailings impoundment could lead to dramatic changes in the minimum pipeline operating velocity. It is important to note that design predictions for pipeline diameters significantly larger than 53 mm must be considered preliminary.

DATA APPENDIX

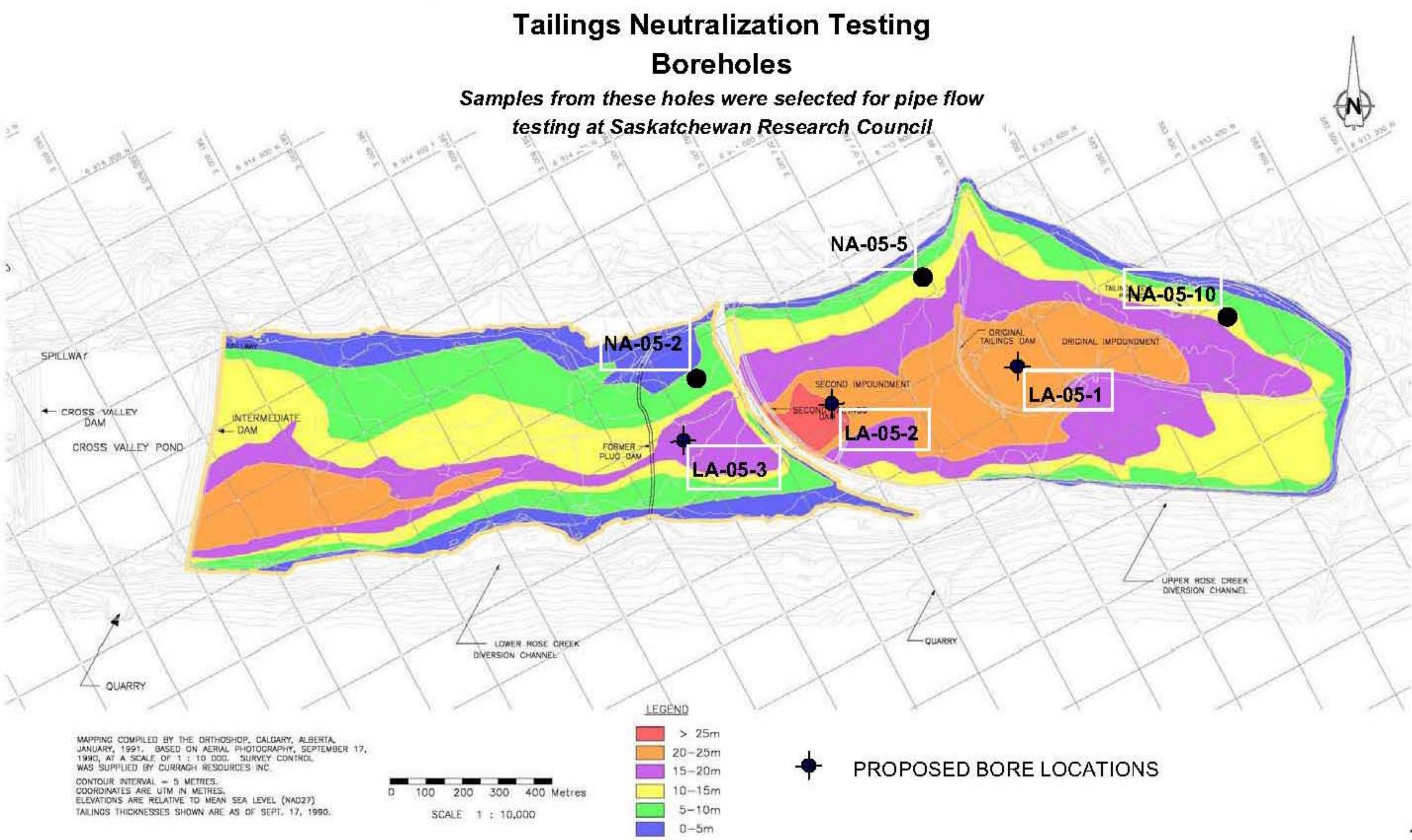

Figure A1. Drill hole location plan (provided by SRK).

Table A1. Supplied tailings samples.

Hole-ID	Sample interval	Mass Available (kg)
LA-05-1	0-1m	2.5
	1-2m	2.8
	2-3m	3.3
	3-4m	3.4
	4-5m	2.4
	5-6m	4.0
	6-7m	4.0
	7-8m	1.8
	8-9m	2.2
	9-10m	2.1
	10-11m	2.8
	11-12m	1.8
	12-13m	2.4
	13-14m	2.8
	14-15m	2.3
	15-16m	1.8
	16-17m	2.3
	17-18m	2.2
LA-05-2	0-1m	1.0
	1-1.5m	0.8
	3-4m	1.5
	4-5m	0.8
	5-6m	1.3
	6-7m	2.9
	7-8m	1.7
	8-9m	1.9
	9-10m	3.0
	10-10.5m	2.2
	12-13m	3.0
	13-14m	2.0
	14-15m	2.1
	15-16m	2.0
	16-17m	2.7
	17-18m	2.4

Hole-ID	Sample interval	Mass Available (kg)
LA-05-3	0-1m	1.7
	1-2m	1.7
	2-3m	1.9
	3-4m	1.4
	4-5m	1.7
	5-6m	2.1
	6-7m	1.5
	7-8m	0.7
	8-9m	1.7
	9-10m	1.8
	10-11m	2.1
	11-12m	2.1
	12-13m	2.3
NA-05-2	0-1m	1.4
	1-1.5m	1.4
	1.5-3m	1.1
	3-4m	2.5
	4-5m	1.3
	5-6m	2.2
	6-7m	1.7
	7-8m	1.7
	8-8.8m	2.7
NA-05-5	0-0.8m	1.8
	0.8-2m	1.6
	2-2.5m	1.3
	2.5-3m	1.6
	3-4m	1.4
	4-5m	1.5
	5-6m	1.9
	6-7m	2.1
	7-8m	1.5
	8-8.9m	1.9
NA-05-10	0-1.1m	1.9
	1.1-2m	1.6
	2-3m	1.8
	3-4m	1.3
	4-5m	0.8
	5-6m	2.3
	6-7m	1.4
	7-8m	1.9
	8-9m	2.3
	9-10m	2.4
	10-11m	2.3
	11-12m	2.4

Table A2. Particle size distributions.

Size (microns)	Total tailings % Finer by mass	Coarse tailings (>44µm) only* % Finer by mass
210	91.2	81.0
149	86.0	69.7
106	78.0	52.5
74	65.5	25.7
53	57.9	9.1
44	53.6	0.0
d_{50}	~40 µm	~100 µm

*The coarse tailings distribution has been calculated using the total tailings distribution.

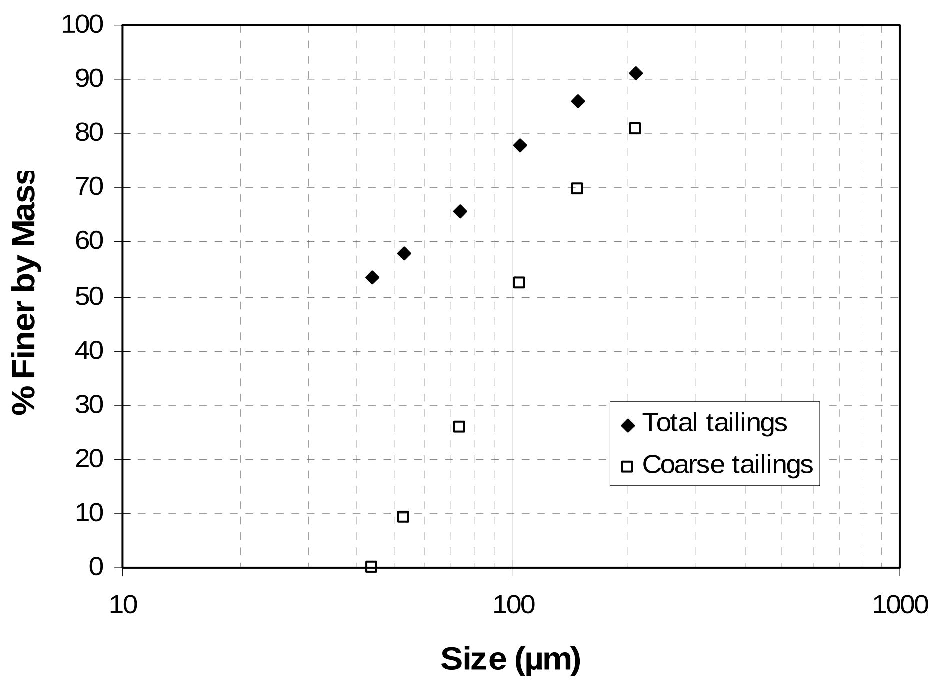

Table A3. Pipeline pressure gradient data for Faro tailings in the 53 mm pipeline at 10°C.

Water data (20ºC)		60% w/w pH=9.8		44% w/w pH=9.7		34% w/w pH=9.8	
Velocity (m/s)	dP/dz (kPa/m)	Velocity (m/s)	dP/dz (kPa/m)	Velocity (m/s)	dP/dz (kPa/m)	Velocity (m/s)	dP/dz (kPa/m)
4.00	2.41	5.01	7.38	4.03	3.82	4.01	3.46
3.51	1.90	4.50	6.20	3.49	3.00	3.51	2.74
3.01	1.44	4.00	5.11	3.00	2.33	3.01	2.11
2.49	1.03	3.50	4.12	2.51	1.72	2.50	1.54
2.00	0.70	3.00	3.21	2.01	1.19	2.02	1.07
1.50	0.43	2.50	2.38	1.49	0.72	1.51	0.65
0.99	0.22	1.99	1.62	1.26	0.53	1.26	0.49
0.49	0.08	1.51	1.01	1.01	0.37	1.03	0.35
		0.99*	0.57	0.89*	0.30	0.92*	0.30

* Indicates velocity at which a stationary deposit was observed.

Table A4. Volumetric concentration profiles (v/v) for Faro tailings in the 53 mm pipeline at a velocity of 1.5 m/s (10°C).

Position y/D	60% w/w	44% w/w	34% w/w
0.95	0.28	0.13	0.08
0.85	0.27	0.14	0.09
0.75	0.28	0.15	0.10
0.65	0.28	0.15	0.11
0.55	0.28	0.15	0.11
0.45	0.31	0.17	0.12
0.35	0.30	0.17	0.12
0.25	0.30	0.18	0.13
0.15	0.29	0.18	0.15
0.05	0.34	0.22	0.19

Attachment F
Cost Estimation

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Technical Memo

To:	Daryl Hockley, Cam Scott	**Date:**	January 13, 2006
cc:	Roger Payne, John Brodie	**From:**	Gordon Doerksen
Subject:	Cost Estimates and Risks Definition of Hydraulic Relocation of Faro Tailings	**Project #:**	1CD003.079

1 Introduction

The following memo contains a review and explanation of cost estimates for the relocation of 61,000,000 tonnes of tailings and basin floor material from the Rose Creek Valley to the Faro Pit. The cost estimate is based on a number of sources; supplier budgetary quotes, personal experience, and previous work done by Marija Jurcevic (SRK) and David Jansson (ECMP). Costs only include direct capital and operating costs for hydraulic mining, slurry pumping and basin cleanup. Indirect costs such as contractor profit and overhead, insurance, bonding, living out allowances, engineering, project management and taxes are not included. Assumptions used in the cost estimate are listed in the design parameters. Cost risks are outlined in Section 7.

2 Summary

Cost estimates for the Faro tailings relocation are as follows:

Table 2.1 Tailings Relocation Cost Summary

Unit Operation	Tonnes (M)	Capital Cost		Operating Cost		Total Cost	
		Tot. Cost ($M)	Unit Cost ($/t)	Tot. Cost ($M)	Unit Cost ($/t)	Tot. cost ($M)	Unit Cost ($/t)
Hydraulic mining	49.0	$4.9	$0.10	$38.1	$0.78	$43.0	$0.88
Slurry pumping	49.0	$6.4	$0.13	$31.1	$0.64	$37.5	$0.77
Basin cleanup	12.0	$0	$0	$39.0	$3.25	$39.0	$3.25
Decommissioning	61.0	$0	$0	$0.5	$0.01	$0.5	$0.01
TOTAL	**61.0**	**$11.3**	**$0.19**	**$108.7**	**$1.78**	**$120.0**	**$1.97**

It must be noted that the costs in Table 2.1 are for the physical removal of tailings only and do not include lime addition or dam removal. These costs will be detailed in other memos.

49,000,000 tonnes of tailings will be moved hydraulically and the remaining 8,000,000 tonnes will be moved with conventional equipment as hydraulic mining methods are not conducive to final cleanup and grading. The final cleanup will also require some over-excavation herein assumed to be 4,000,000 tonnes giving a total conventionally mined tonnage of 12,000,000 tonnes.

Hydraulic mining and slurry pumping operating costs are distributed as follows:

Category	**Unit cost**	**%**
Labour:	$ 0.47	34%
Spares/cons./misc:	$ 0.47	33%

Electricity:	$ 0.47	33%
TOTAL	**$ 1.41**	**100%**

3 Design parameters

The following main assumptions were used in the cost estimate:

Total mass of tailings:	57,000,000 dry tonnes
Conventional mining mass:	12,000,000 dry tonnes (approximately 2m of tailings over the entire valley floor plus 1m of over-excavation)
Hydraulic mining mass:	49,000,000 dry tonnes
Hydraulic operating period:	5 months / 150 days per year 21.6 hours per day 3,240 actual operating hours per year 194,400 minutes 20 tonnes per minute
Solids density:	4.0 tonnes/m^3
In situ tailings bulk density:	2.0 tonnes/m^3
Slurry % solids by wt.:	46%
Slurry density:	1.53 tonnes/ m^3
Hydraulic mining equipment	
Mechanical availability:	90% for the entire hydraulic mining and slurry system
Equipment utilization:	100%

Quoted prices are as of the December 2005. Note that pipeline costs can be highly variable based on the going rate for steel and crude oil

No inflation factor was used

No contingency factor was used other than budgetary (undiscounted) vendor pricing

Labour is employed for six months/year although hydraulic mining only takes place for 5 months/year. One month will be used for start-up, shutdown, and repairs.

No salvage value of equipment was considered.

No use of equipment current at the mine site was considered for use except as noted.

4 Hydraulic Mining Costs

4.1 Direct Capital Costs

Hydraulic mining capital costs are made up of the purchase and installation of high pressure water pumps, pipes and monitors. Most of the project capital mobile equipment is also included in the hydraulic monitoring operation as this is the area where it will be most used.

The hydraulic mining capital is made up of the following components (including installation and spares, excluding taxes):

Category	**Cost**
Pumps	$ 300,000
Pipelines	$ 1,764,000
Monitors and accessories	$ 1,255,000
Mobile equipment	$ 1,020,000
Misc.	$ 595,000
TOTAL	**$ 4,934,000**
	$ 0.10/tonne

It was assumed that the barge currently supporting the pumps in the Faro pit will be sufficient for the monitor feed pumps. A third, spare water pump is included in the capital estimate.

It was also assumed that the power line running to the pit would be of sufficient capacity to handle the high pressure monitor water pumps. $100K is included in the capital plan for new pit transformers, if required, switchgear and various miscellaneous lighting installations around the site.

All water monitor pipe is steel and recent price increases in the steel industry have driven up steel pipe costs considerably. This trend may continue or reverse in the future. No salvaged pipelines were considered when determining the capital requirement for the project and if suitable pipe can be found at the mine site then a cost savings may be realized. Pipeline accessories (couplings, tees, elbows, valves, etc.) are costed at 20% of the total 16" and 8" pipeline costs. These pipes are connected using flanges or Victaulic-type fittings in order to maintain mobility. The 24" main line is a solid welded line.

A total of 9 monitors, booths and controls are included in the capital plan. The initial setup of the monitors and 8" pipeline will be done with the hydraulic mining crew at the start of the project.

Mobile equipment purchase is included in the capital for the project but an alternative may be to lease equipment each season. This option should be looked at in detail if the hydraulic monitoring option proceeds. The mobile equipment required for the operation is as follows:

Item	Quantity
Excavator	1
Rough terrain crane or 2nd excavator	1
Tele-handler	1
Pick-up trucks	2
Service truck (Hiab)	1

The excavator and rough terrain crane will be used to move sump pumps, monitors, pipeline, etc. Soft ground conditions may make a 2nd excavator preferable over a rough terrain crane, however, excavators are much less mobile. The telehandler will be used for general site maintenance and clean-up as it will be fitted with a forklift, jib or bucket. Two pick-up trucks will be used for supervision and operating crews. The service truck will be used by the maintenance personnel.

A changeroom/dry, misc. freight, and engineering/procurement/construction management costs round out the capital expenditures for hydraulic mining. Details of the capital costs are as follows:

Table 4.1 Hydraulic Mining Cost Details - Capital

Item	Description	Quantity	Unit Cost	Units	Total cost	Comments
Capital Costs						
	Pumps					
1	Vertical turbine pumps - 1000 hp 7 stage rated at 227l/s at 26 bar mounted on existing barge (includes electrical switchgear and instrumentation and spare pump)	3	$100,000	ea	$300,000	Laurie Caviggion, CAC Industrial
	Subtotal				*$300,000*	
	Pipelines					
2	Primary pipeline 24" schedule 20 (standard) steel pipe	3,500	$202	m	$706,201	Melanie Cadieux, Napsteel, Van.
3	Install primary pipeline	3,500	$85.71	m	$300,000	Klondike Welding
4	Secondary pipelines 16" schedule 20 steel pipe	2,000	$111	m	$222,966	Melanie Cadieux, Napsteel, Van.
5	Install secondary pipeline	2,000	$75.00	m	$150,000	Klondike Welding
6	Tertiary pipeline 8" schedule 20 steel pipe	2,000	$48	m	$95,341	Melanie Cadieux, Napsteel, Van.

7	Install tertiary pipeline				$50,000	estimate
8	Pipeline accessories 20% of 8” and 16" pipe cost				$63,661	estimate
9	Pipe transport (8"-74lb/m, 16"-172lb/m, 24"-310lb/m)	27	$6,500	trip	$175,500	Continental Cartage Vancouver to Faro using 60,000lb flat deck
	Subtotal				*$1,763,669*	
	Hydraulic Monitors and accessories					
10	Hydraulic monitors	9	$102,222	ea	$920,000	ECMP
11	Monitoring accessories	1	$200,000	ea	$200,000	ECMP
12	Monitor control huts	9	$15,000	ea	$135,000	estimate
	Subtotal				*$1,255,000*	
	Mobile Equipment					
13	Rough terrain crane (28 ton) or excavator	1	$300,000	ea	$300,000	estimate
14	Tele-handler	1	$150,000	ea	$150,000	estimate
15	Excavator	1	$300,000	ea	$300,000	estimate
16	Pick-up trucks	2	$60,000	ea	$120,000	estimate
17	Service truck (Hiab)	1	$150,000	ea	$150,000	estimate
	Subtotal				*$1,020,000*	
	Misc.					
18	Change room	1	$150,000	ea	$150,000	estimate
19	Other transport	7	$6,500	trip	$45,500	estimate
20	Electrical service		$100,000	ea	$100,000	estimate
21	Engineering/construction/project management				$300,000	estimate
	Subtotal				*$595,500*	
	TOTAL CAPITAL				**$4,934,169**	**Project**

4.2 Direct Operating Costs

Hydraulic mining operating costs are estimated to be $ 0.78/tonne and are comprised of electrical power, spares/consumables, road building, and labour as outlined in Table 4.2.

Electrical power is based on a total charge of $0.147/kWh as quoted by Yukon Energy.

Maintenance supplies and consumable costs are not done in detail and are assigned a cost of 2% of the hydraulic mining capital cost per month.

Road building is included at a fixed rate of $500,000 per year to take care of costs associated with advancing a waste rock ramp and road should they be necessary for sump pump advancement.

Labour makes up 44% of the hydraulic mining operating costs. The operation runs 24 hours per day, 7 days per week with crews rotating on a 12 hr., 4 on 4 off schedule. Each shift is comprised of:

- 1 x Working Lead Hand
- 3 x Monitor Operators
- 2 x Labourers

Every day shift also has one electrician and one millwright. A Superintendent oversees the operation on a Monday to Friday basis.

Table 4.2 Hydraulic Mining Cost Details – Operating Costs

Operating Costs							
1	Electrical power (2 x 1,000 hp x .746 x .85=1268 kW * 648hr/mo = 821664 x 5mo)		4,108,320	$0.147	Kwh	$603,923	
2	Maintenance / supplies / tools / misc		6		mo	$592,100	
3	Road building / ditch maintenance					$500,000	
Subtotal						**$1,696,023**	**per season**
						$282,671	**per month**
Operating Labour (assumes a 4x4, 12 hour shift schedule)							
1	Superintendent	$50/hr, 40% OH, 160 hr/mo.	1	$11,200	6 mo	$67,200	
2	Working Lead Hand	$37/hr, 40% OH, 180 hr/mo.	4	$9,324	6 mo	$223,776	
3	Electrician	$35/hr, 40% OH, 180 hr/mo.	2	$8,820	6 mo	$105,840	
4	Millwright	$35/hr, 40% OH, 180 hr/mo.	2	$8,820	6 mo	$105,840	
5	Monitor operators	$30/hr, 40% OH, 180 hr/mo.	12	$7,560	6 mo	$544,320	
6	Labourers	$25/hr, 40% OH, 180 hr/mo.	8	$6,300	6 mo	$302,400	
Subtotal			**29**			**$1,349,376**	**per season**
						$224,896	**per month for 6 months**
Annual Operating Cost						**$3,045,399**	**per season**
						12.5	**seasons**
Total Cost						**$38,067,492**	**per project**
Hydraulic Mining Unit Operating Cost						**$0.78**	**/tonne**
Total Hydraulic Mining Unit Cost (including Capital)						**$0.88**	**/tonne**

5 Slurry Pumping Costs

5.1 Direct Capital Costs

The preferred mode of operation for the slurry pumping system requires the use of 3 vertical cantilevered sump pumps positioned as close to the mining faces as possible. The pumps are housed in box structures that support the pumps without the use of a crane. The structures are perforated to allow slurry into the pumps but keep large sticks and debris out. From the sump pumps the slurry flows over a trash screen to the booster pumps that convey the slurry to the Faro pit via steel and HDPE pipe. The booster pump system comprises five Allis Chalmers 14 x 12 x 36 SRL pumps in series to provide the necessary hydraulic head to pump into the pit.

In summary, the slurry pumping system capital costs are as follows:

Category	**Cost**
Pumps (sump and booster):	$ 1,793,000
Pipelines:	$ 3,434,000
Misc.:	$ 1,125,000
TOTAL	**$ 6,352,000**
	$ 0.13/tonne

The capital cost includes one spare pump for each type of pump. It also includes costs for the construction of the booster pump station including concrete structure, switchgear, and the cost of running power from the plant area to the booster pump station. Installation and freight for all pipelines is included in the capital cost. Pipeline costs are very high, due in part by the fact that HDPE pipe costs have risen substantially in the past year due to the increased price of oil.

Pipeline accessories costs including valves, tees, elbows, couplings, etc. are included at 20% of the pipeline cost.

The details of the capital cost of the slurry pumping system can be found in Table 5.1 below.

Table 5.1 Slurry Pumping Capital Cost Details

Item	Description	Quantity	Unit Cost	Units	Total cost	Comments
Capital Costs						
	Pumps and support					
1	Vertical cantilever sump pumps rated at 165 l/s @ 30m head + 1 spare	4	$75,000	ea	$300,000	Laurie Caviggion, CAC Industrial
2	Support structure for sump pumps	1	$100,000	ea	$100,000	estimate
3	Slurry booster pumps (AC 14x12x36 SRL-C,CHD and 2 x CXHD + 1 spare)	6	$113,667	ea	$682,000	Laurie Caviggion, CAC Industrial
4	Pump station (including electrical switchgear and instrumentation)	1	$100,000	ea	$100,000	estimate
5	Trash screen and support	1	$500,000	ea	$500,000	ECMP
6	Booster pump box and support	1	$75,000	ea	$75,000	estimate
7	Gland water pumps (Grundfos CR 30 6, 12 and 15 stage pumps)	6	$6,000	ea	$36,000	Laurie Caviggion, CAC Industrial
	Subtotal				*$1,793,000*	
	Pipelines					
8	Sump pump pipeline (HDPE 16" DR11 60' (18.3m) lengths)	4500	$171	m	$769,500	KWH Pipe - Wosley Eng. - Don
9	Install sump pipeline(Tech $400/day+R&B/6 fusions day)= 41 days	4500	$22	m	$99,600	KWH Pipe - Wosley Eng. - Don
10	Steel booster pump pipeline (550mm standard wall, PE lined)	500	$375	m	$187,500	Marija Jurcevic - SRK Reno
11	HDPE booster pump pipeline (26" SDR9 - 60'lengths)	2500	$538	m	$1,344,160	KWH Pipe - Wosley Eng. - Don
12	Install steel booster pump pipeline	500	$300	m	$150,000	Marija Jurcevic - SRK Reno
13	Install HDPE booster pump pipeline	2500	$99	m	$247,400	Marija Jurcevic - SRK Reno
14	Pipeline accessories 20% of cost				$460,232	estimate
15	Pipe transport (550mm-300lb/m, 650mm SDR9-298lb/m, 16" DR11-95lb/m)	27	$6,500	trip	$175,500	Continental Cartage Van-Faro 604 540-7999 - 60,0000 lb flat deck
	Subtotal				*$3,433,892*	
	Misc.					
16	Other transport	10	$6,500	trip	$65,000	estimate
17	Pick-up trucks	1	$60,000	ea	$60,000	estimate
18	Electrical service for booster pumps and sump pumps	1	$500,000	ea	$500,000	estimate
19	Engineering/construction/project management				$500,000	estimate
	Subtotal				*$1,125,000*	
	TOTAL CAPITAL				**$6,351,892**	**Project**

5.2 Direct Operating Costs

The direct operating cost of slurry pumping is $ 0.64/tonne and is divided into electricity (49%), spares and consumables (30%) and labour (21%).

The focus of the slurry pumping operation is to keep the equipment running at peak efficiency, with the maximum amount of operating time.

One millwright and one electrician work dayshift for 12 hours every day. This number is complemented by one millwright and electrician scheduled for the hydraulic monitoring operation, providing adequate manpower for routine maintenance and breakdown repairs on a daily basis. Night time breakdowns are covered on overtime.

Two labourers on every shift monitor the pumps and move the sumps as necessary. Labourers also assist with moving monitors and pipelines. Labour for the project is employed for 6 months of the year, although the operation will only run for 5 months. The extra month is used for training, start-up and shutdown tasks.

Maintenance supplies and consumables are costed at 2% of the total capital cost per month.

Electrical costs are calculated using a total electricity charge of $0.147/kWh.

Slurry pumping operating costs details are shown in table 5.2.

Table 5.2 Slurry Pumping Operating Cost Details

						Cost/year		Cost/project
1	Electrical power (2200kW / .90 = 2429 kW * 648hr/mo = 1,574,000 x 5.25mo)		8,262,000	$0.147	Kwh	$1,214,514		$15,181,425
2	Maintenance / supplies / tools / misc		6		mo	$762,227		$9,527,838
						$1,976,741		**$24,709,263**
						$329,457	**per month**	
						$0.49	**/tonne**	
	Labour (assumes a 4x4, 12 hour shift schedule)							
1	Electrician hr/mo.	$35/hr, 40% OH, 180	2	$8,820	6	$105,840		$1,323,000
2	Millwright hr/mo.	$35/hr, 40% OH, 180	2	$8,820	6	$105,840		$1,323,000
3	Labourers hr/mo.	$25/hr, 40% OH, 180	8	$6,300	6	$302,400		$3,780,000
			12			**$514,080**		**$6,426,000**
						$85,680	**per month**	
						$0.13	**/tonne**	
			Total Operating			**$2,490,821**		**$31,135,263**
						12.5	**seasons**	
			Unit Operating Cost			**$0.64**	**/tonne**	
			Total Unit Cost (inc. Capital)			**$0.77**	**/tonne**	

6 Basin Clean-up Costs

Hydraulic mining is not conducive to completely removing all tailings to a defined, flat gradient. The high pressure from the water monitors tends to dig down below original topography very quickly and the organic matter (sticks and braches) picked up by high pressure can cause substantial sump and pump clogging problems for a hydraulic operation. From experience at other hydraulic mining operations, it is assumed that a layer of tailings 2 metres thick would remain over the entire area of the tailings basin after hydraulic mining. Conventional mining is planned to follow hydraulic mining to remove the remaining tailings and grade the valley floor to a desirable topography. It is also estimated that an additional 4,000,000 tonnes of over-excavation will need to be done with conventional mining equipment.

Articulated trucks and tracked excavators are chosen as the prime mining equipment for the basin cleanup. Articulated trucks are the only off-highway trucks that perform well in wet, muddy conditions. Excavators are the preferred loading machines as they also work very well in muddy conditions and are able to do the best job at fine cleaning ad grading tasks.

Caterpillar's Fleet Production and Cost (FPC) Version 3.05B software was used to estimate equipment productivities, fleet requirements and cycle times. The analysis of fleet optimization was not extensive, however, excavator size, truck size and the number of trucks required received preliminary optimization. The input parameters and results of the production and cost analysis can be found in Appendix A. A summary of the costs is in Table 6.1.

The fleet selected for the basin clean up consists of 1 x 4m^3 excavator matched with 6 x 40-tonne articulated haul trucks. The long, up hill haul from the tailing basin to the pit are the main drivers for the high number of trucks per excavator. The time to finish the project is based on the number of fleets of excavators and trucks put on the job. One fleet (one excavator and six haul trucks will move 1.4 MBCM/year operating 7 days/week, 24 hours/day, minus equipment availability and utilization)

The haul cycle for the trucks is based on a route from the approximate volumetric centroid of the tailings and utilizes existing roads as much as possible. Some money was included in the first year for road building.

It was assumed that the final clean-up of tailings would be done using a contractor. Equipment costs are generated from the BC Roadbuilder's blue book of costs. Blue book costs include all owning and operating costs but do not include supervision costs. Supervision costs are added separately as a line item.

Table 6.1 Basin Cleanup Costs – Contractor

Item	Description	Quantity	hr	Unit Cost	Units	Total cost	Comments
1	Construct haul road from valley to Faro Pit (contractor all-in)					$1,000,000	estimate
2	Tailings clean-up (2m depth over entire valley floor, contractor all-in)	6,000,000		$6.33	m^3	$25,282,985	Estimate based on CAT FPC software and unit costs from BC Road Builders blue book
	Labour						
3	Supervision	4	3285	$54.79	$/hr	$719,941	estimate
4	Surveying/Engineering	1	3120	$48.00	$/hr	$149,760	estimate
5	Excavator operator	12	3285	$36.60	$/hr	$1,442,772	RB Blue Book
6	Truck drivers	72	3285	$36.60	$/hr	$8,656,632	RB Blue Book
7	Cat operator	4	3285	$36.60	$/hr	$480,924	RB Blue Book
8	Grader operator/water truck	4	3285	$36.60	$/hr	$480,924	RB Blue Book
9	Lube truck operator	2	3285	$36.60	$/hr	$240,462	RB Blue Book
10	Maint.	4	3285	$40.00	$/hr	$525,600	RB Blue Book
	Total labour	**98**				**$12,697,015**	
	Total Clean-up					**$38,980,000**	
		Unit Operating Cost				**$3.25**	**/tonne**

7 Project Risk and Sensitivity Analysis

The relocation of tailings using a combination of hydraulic and conventional mining is a concept that has been used elsewhere albeit in different conditions and on different scales.

The cost and productivity assumptions in this cost estimate are in line with industry standards and the experience at other operations. As with any project, however, there are inherent risks. It relatively easy to determine capital costs for a project. Equipment specifications and vendor quotes are easily obtained and confirmed. What are not easily confirmed are the productivity related estimates that, in turn, drive operating costs. This uncertainty is the greatest concern to the hydraulic mining operation.

One uncertainty is the ability of the Faro tailings to flow in a slurry to the sump pumps. The slurry is assumed to be comprised of 46% solids by weight. If this density of slurry is not achieved, as was the case in Yellowknife, then the operating costs of the project go up substantially. If the slurry carries 30% less solids than planned then, all things being equal, production is reduced and the project will take 30% longer to complete corresponding to roughly 30% higher direct hydraulic mining costs.

Risk analysis was done by varying the factors that are deemed to be key to the success of the project, namely;

- Capital cost
- Power cost
- Basin cleanup cost
- Hydraulic mining productivity
- Labour cost

As expected, hydraulic mining productivity has the greatest impact on costs followed by basin cleanup costs. Capital costs have the least impact as they comprise only 10% of the total project expenditure. See Figure 7.1.

It may be possible to mitigate potential problems by developing solutions in advance. See Table 7.1.

Other problems may occur with the hydraulic mining operation but it is believed that the concept is viable and offers a cost competitive solution when compared to other methods of tailings relocation.

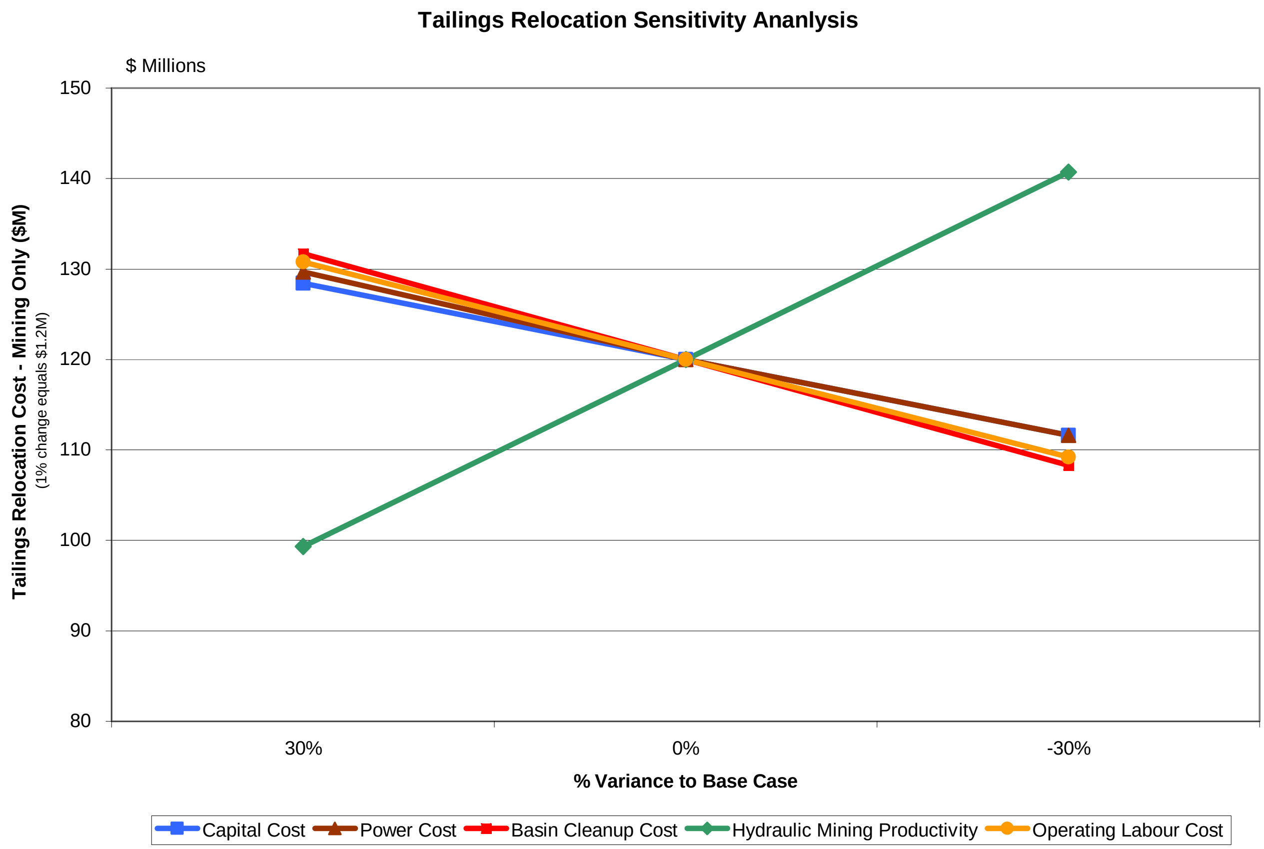


Figure 7.1 Sensitivity Analysis Graph showing the affects of variations of input parameters on the project cost

Table 7.1 Potential hydraulic tailings relocation problems and solutions

System	Potential Issue	Corrective Action	Cost Implication
Monitoring	Insufficient slurry density (low % solids in slurry)	- Change configuration of the mining plan to an undercut method - Keep sumps closer to mining face. - Excavate ditches for slurry flowing to sumps.	Major concern and cost. A drop in the % solids in the slurry from 45% to 32% (by weight) will increase project costs by almost 20% from $120M to over $140M. The cost of changes the mining configuration should be relatively minor as the cost for building roads to support an undercut mining method have already been included.
Basin Cleanup	Greater than 2m thickness of tailings left after hydraulic mining	Use low pressure water to passively wash tailings to sump while hydraulic mining is taking place	Basin clean-up is a large component (33%) of the total tailings relocation costs. If the remaining tailings thickness increases by 1m approximately $13M will be added to the project cost.

APPENDIX A

Caterpillar Fleet Production and Cost Software Analysis Sheets

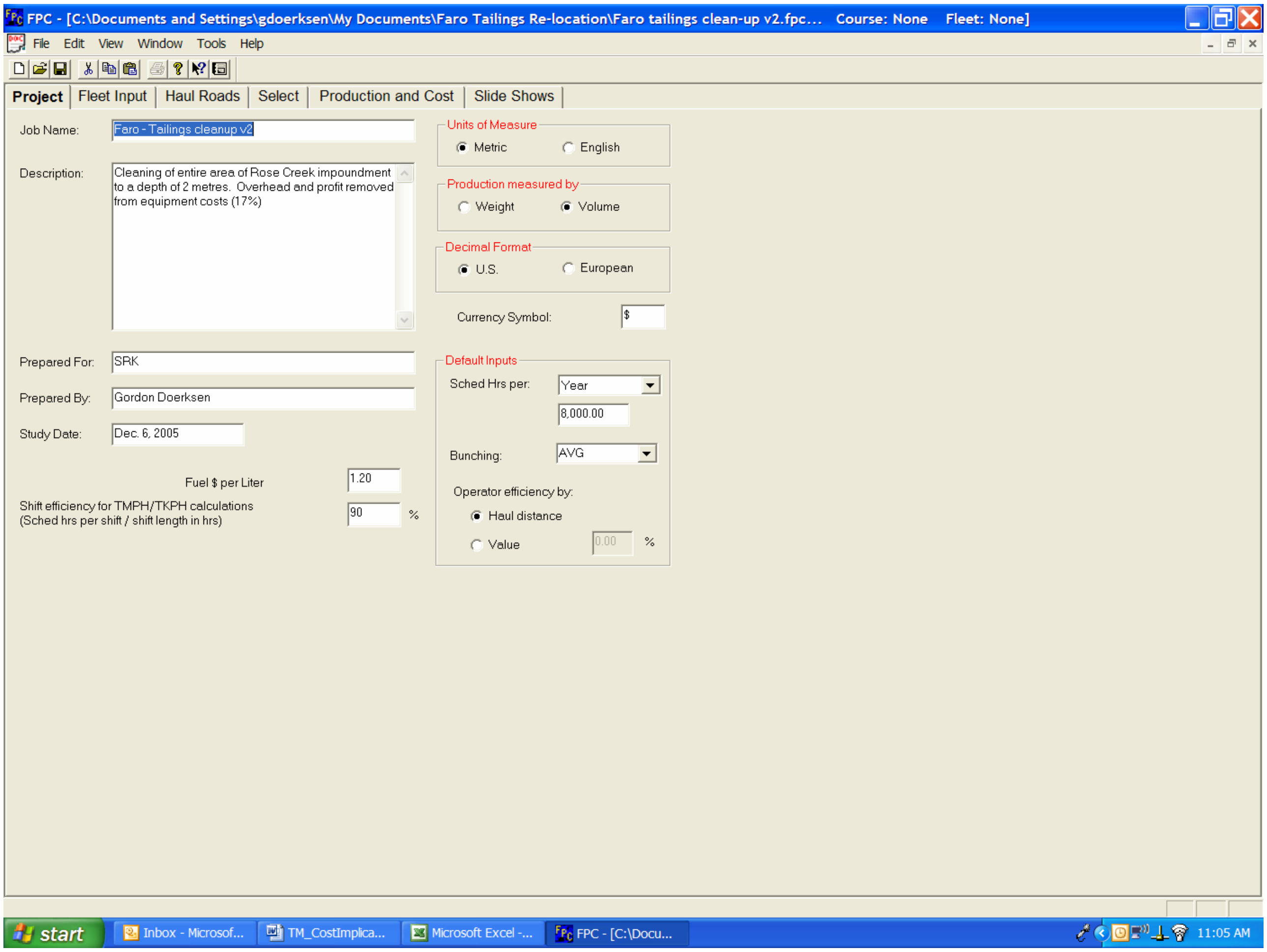


Figure A.1 FPC Project Description

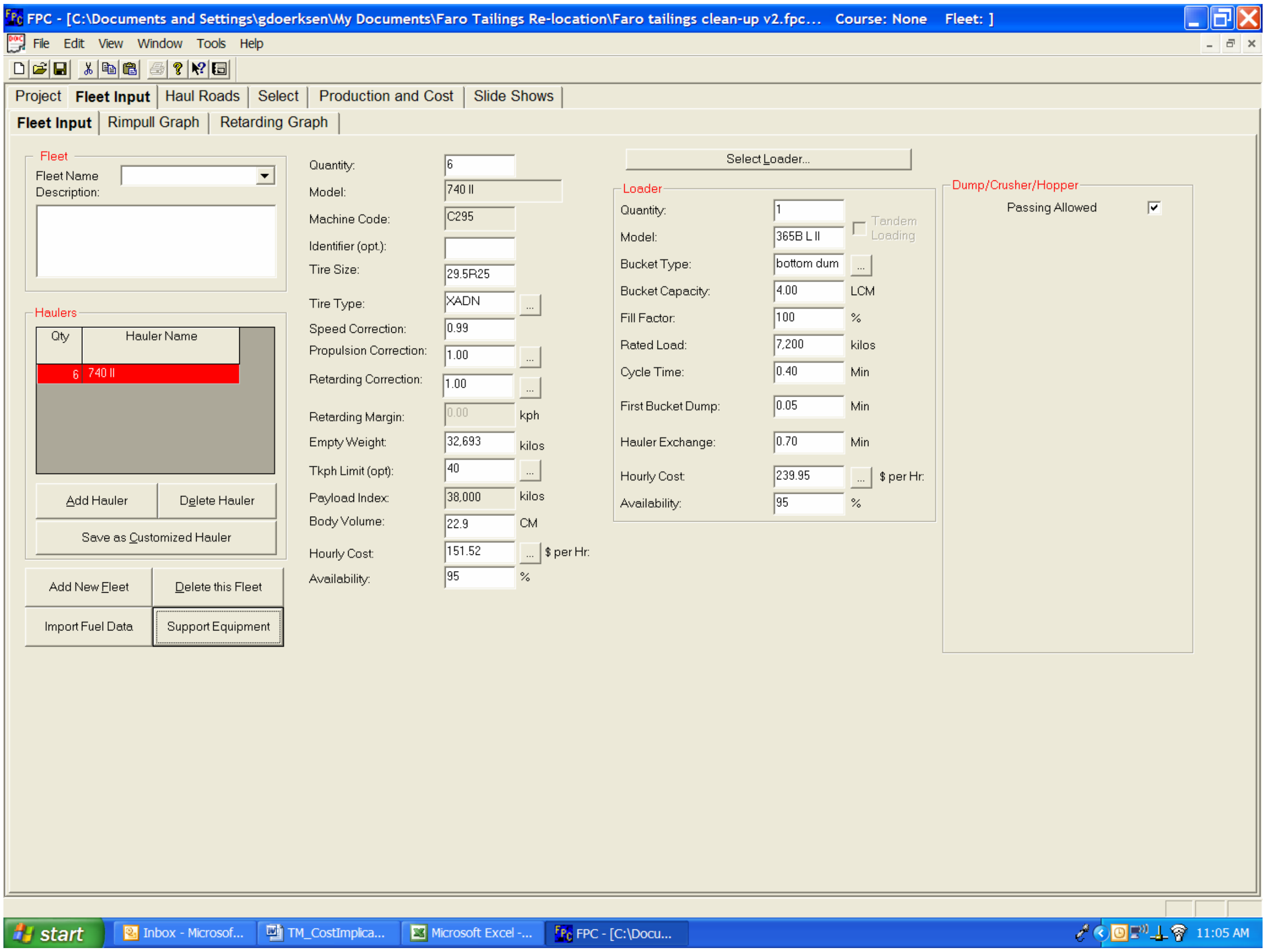


Figure A.2 FPC Equipment Description

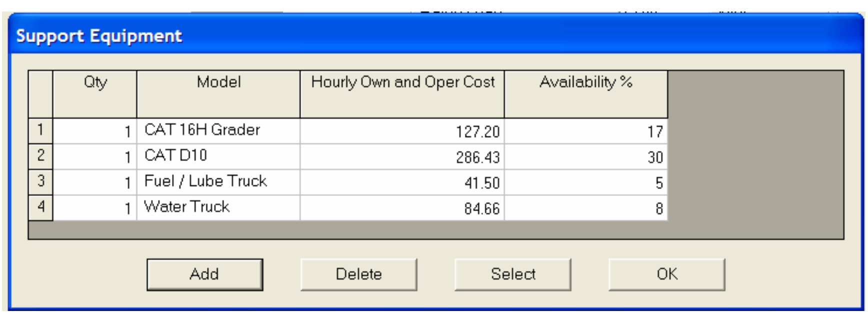


Figure A.3 Basin Clean-up Support Equipment

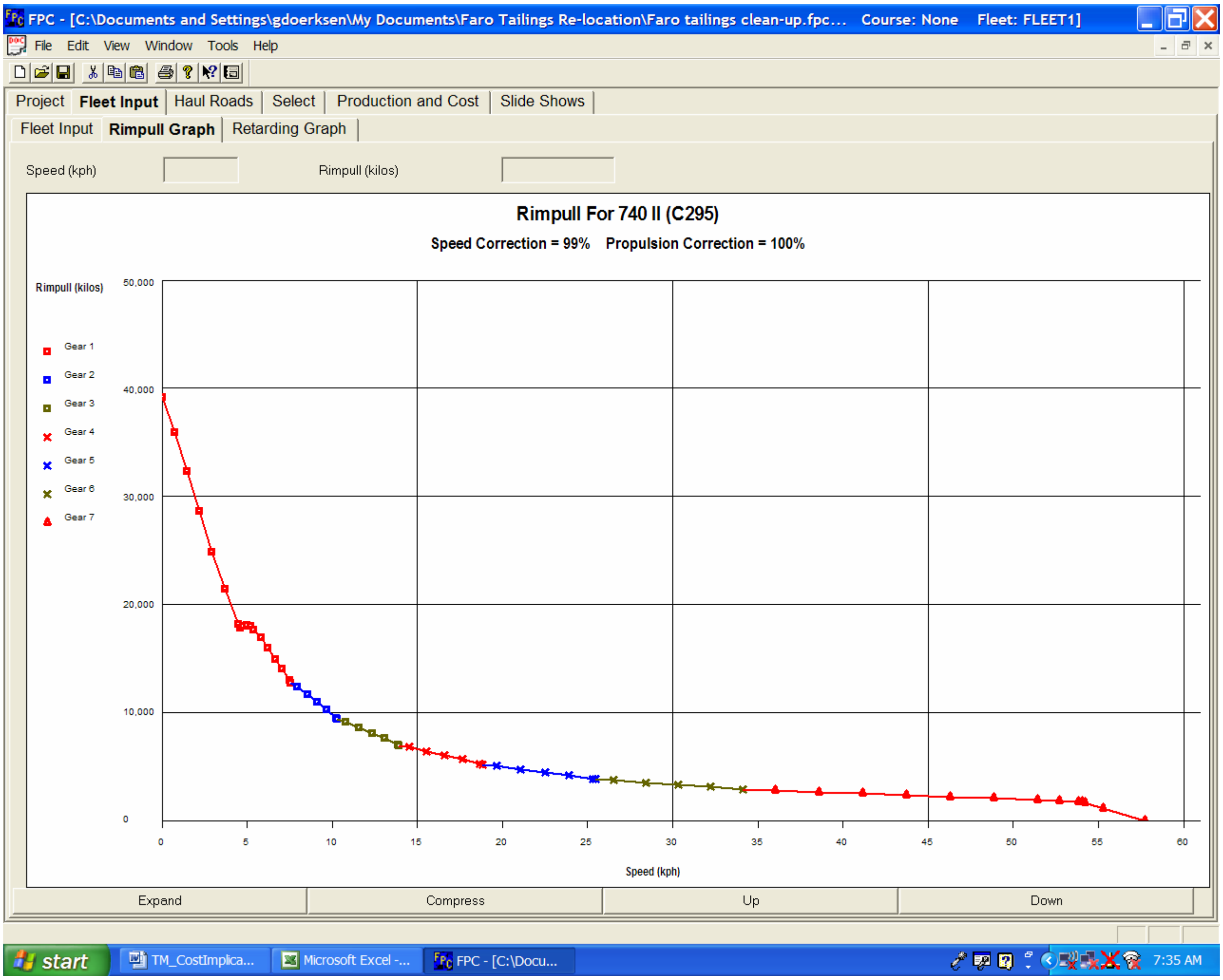


Figure A.4 FPC Rimpull graph for CAT 740II Trucks

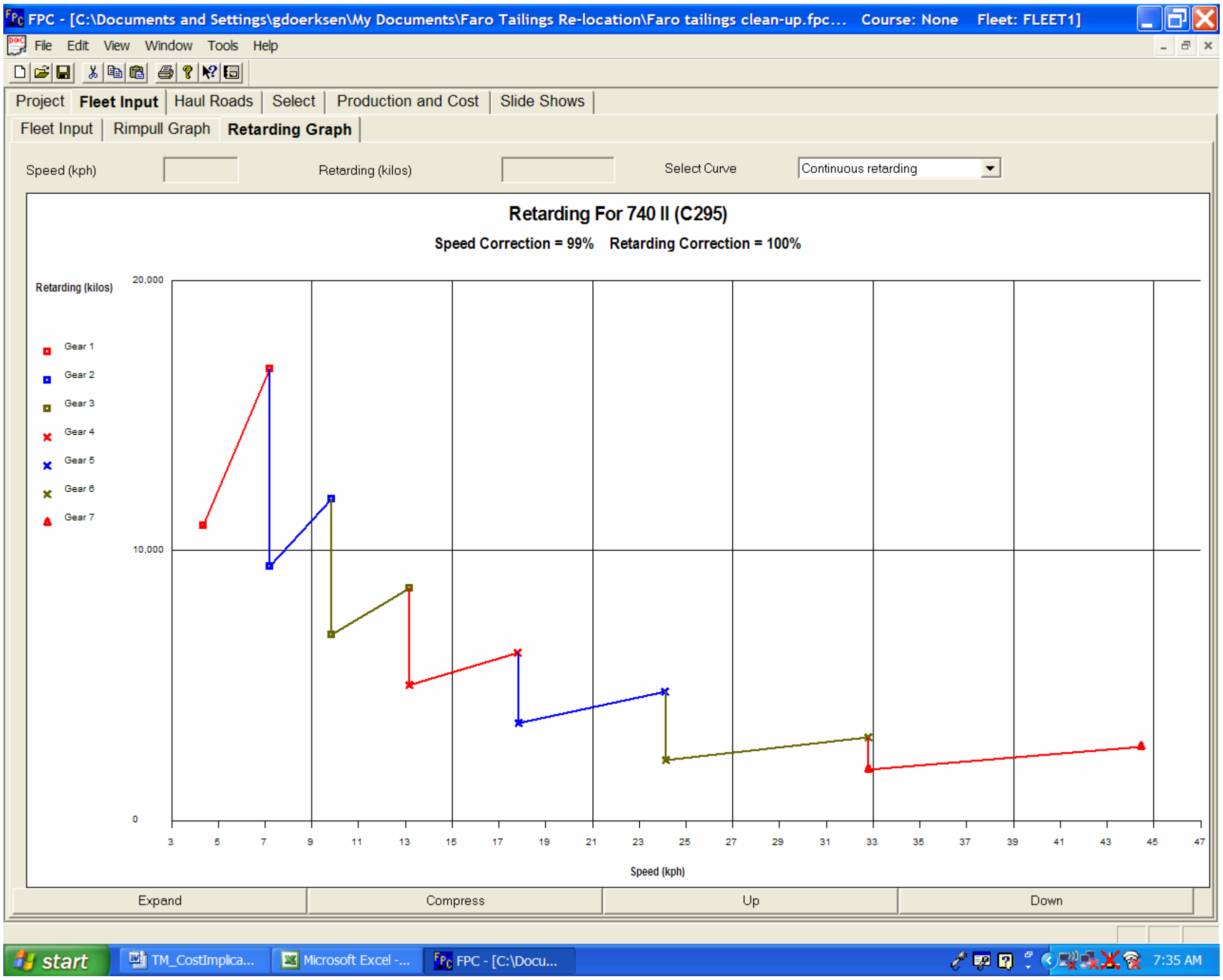
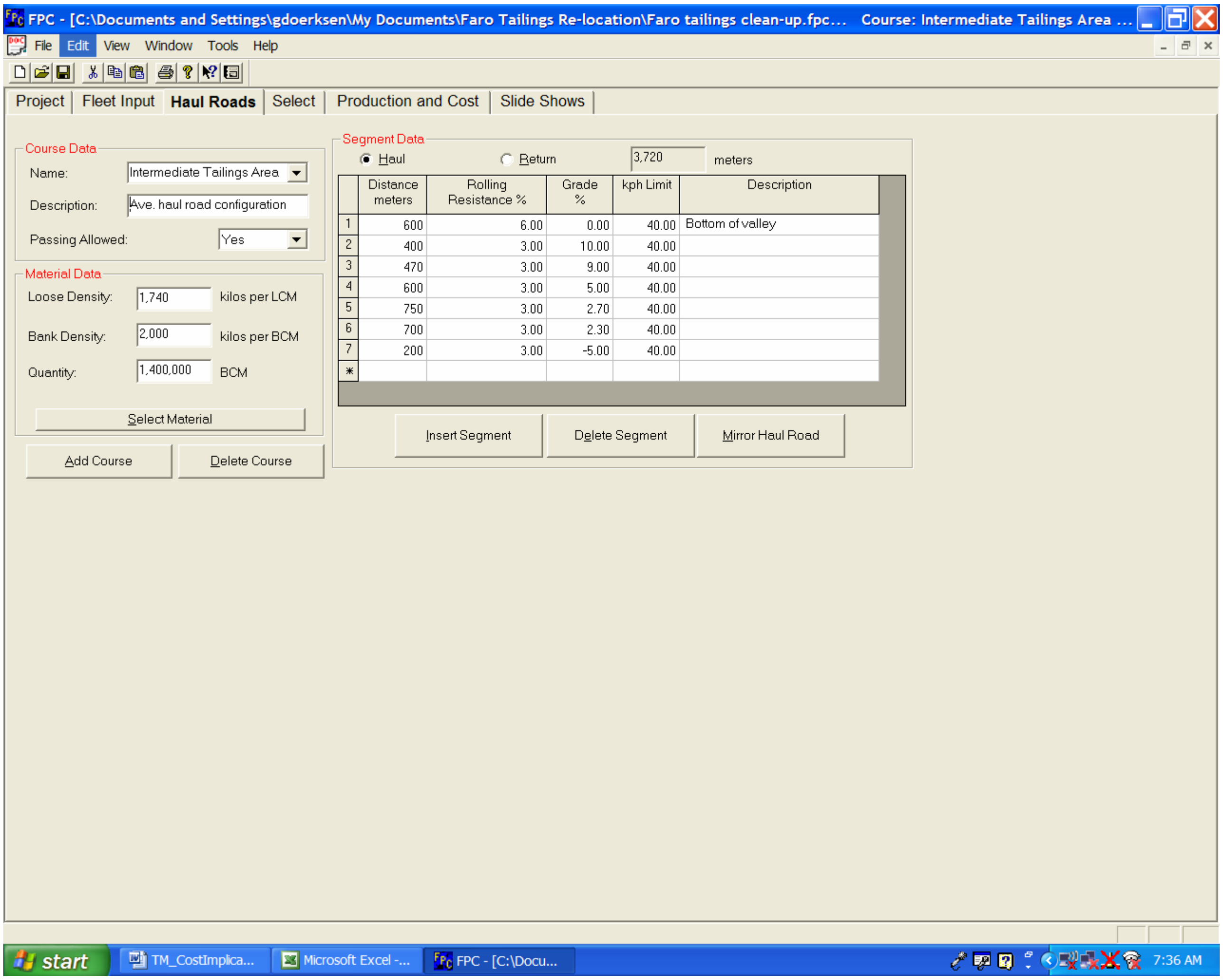


Figure A.5 FPC Retarding Graph for CAT 740II Trucks



	Distance meters	Rolling Resistance %	Grade %	kph Limit	Description
1	600	6.00	0.00	40.00	Bottom of valley
2	400	3.00	10.00	40.00	
3	470	3.00	9.00	40.00	
4	600	3.00	5.00	40.00	
5	750	3.00	2.70	40.00	
6	700	3.00	2.30	40.00	
7	200	3.00	-5.00	40.00	
*					

Figure A.6 FPC Haul Road Profile – Loaded (Return haul route is the mirror image)

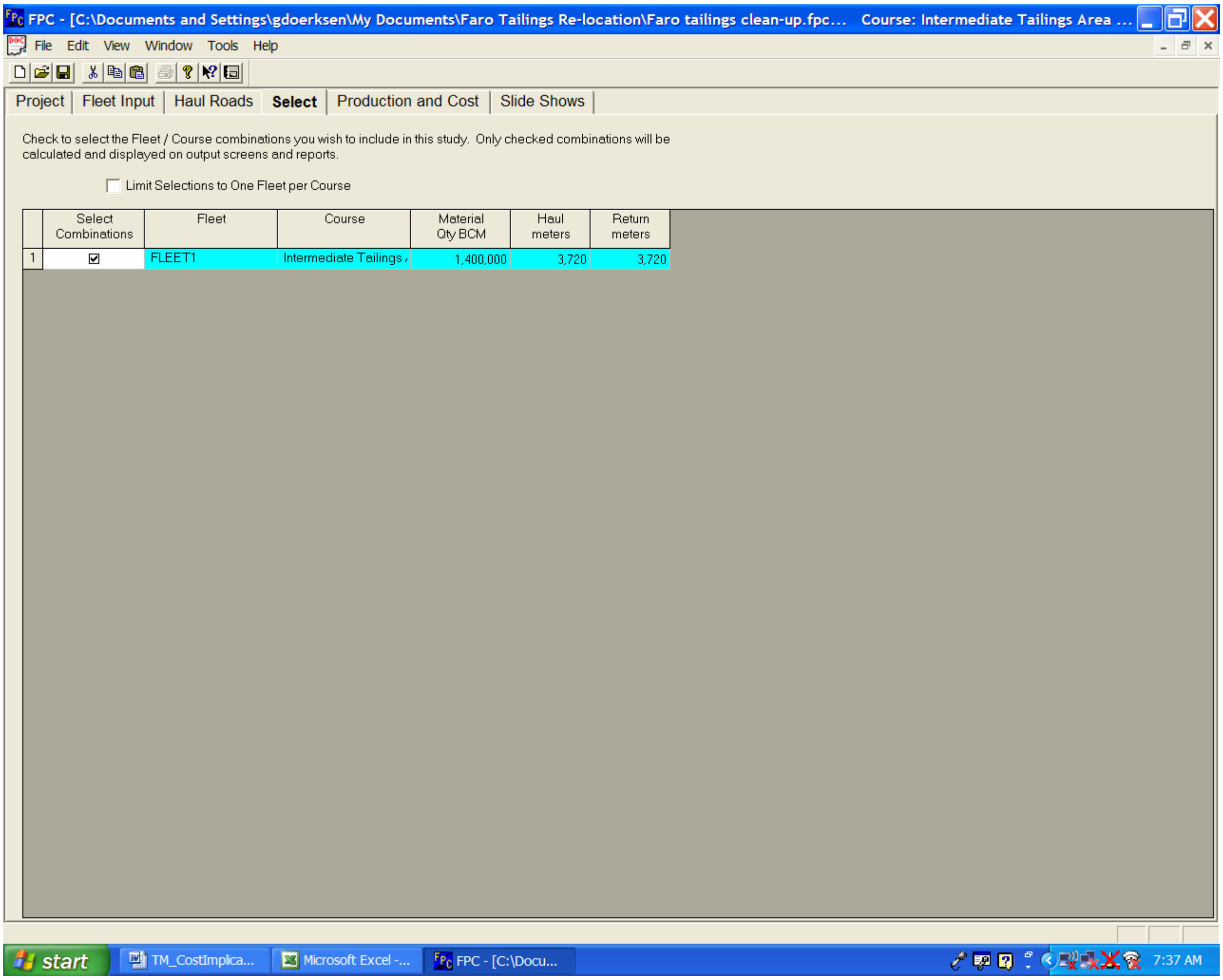


Figure A.7 FPC Fleet Selection

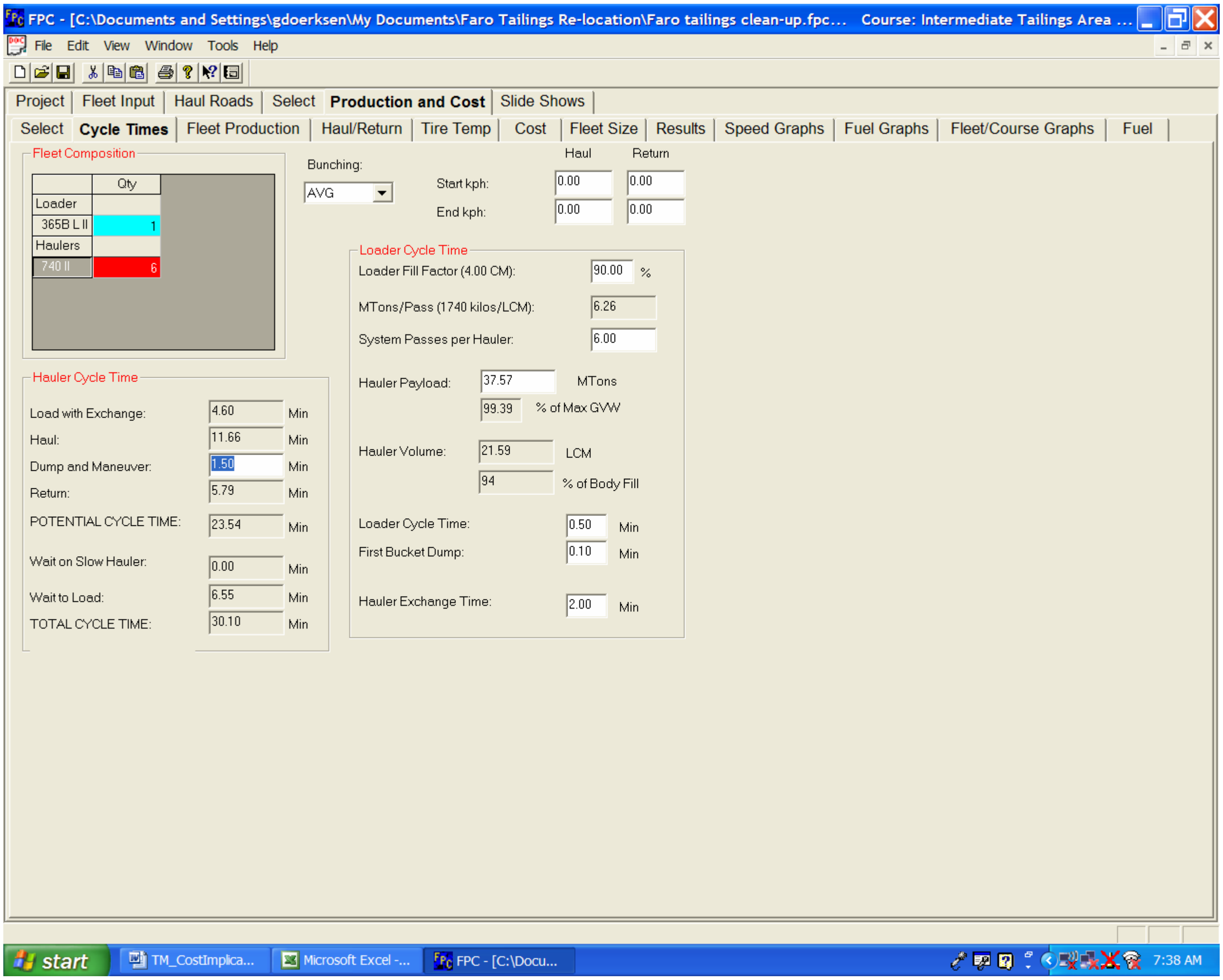


Figure A.8 FPC Cycle Time Estimate

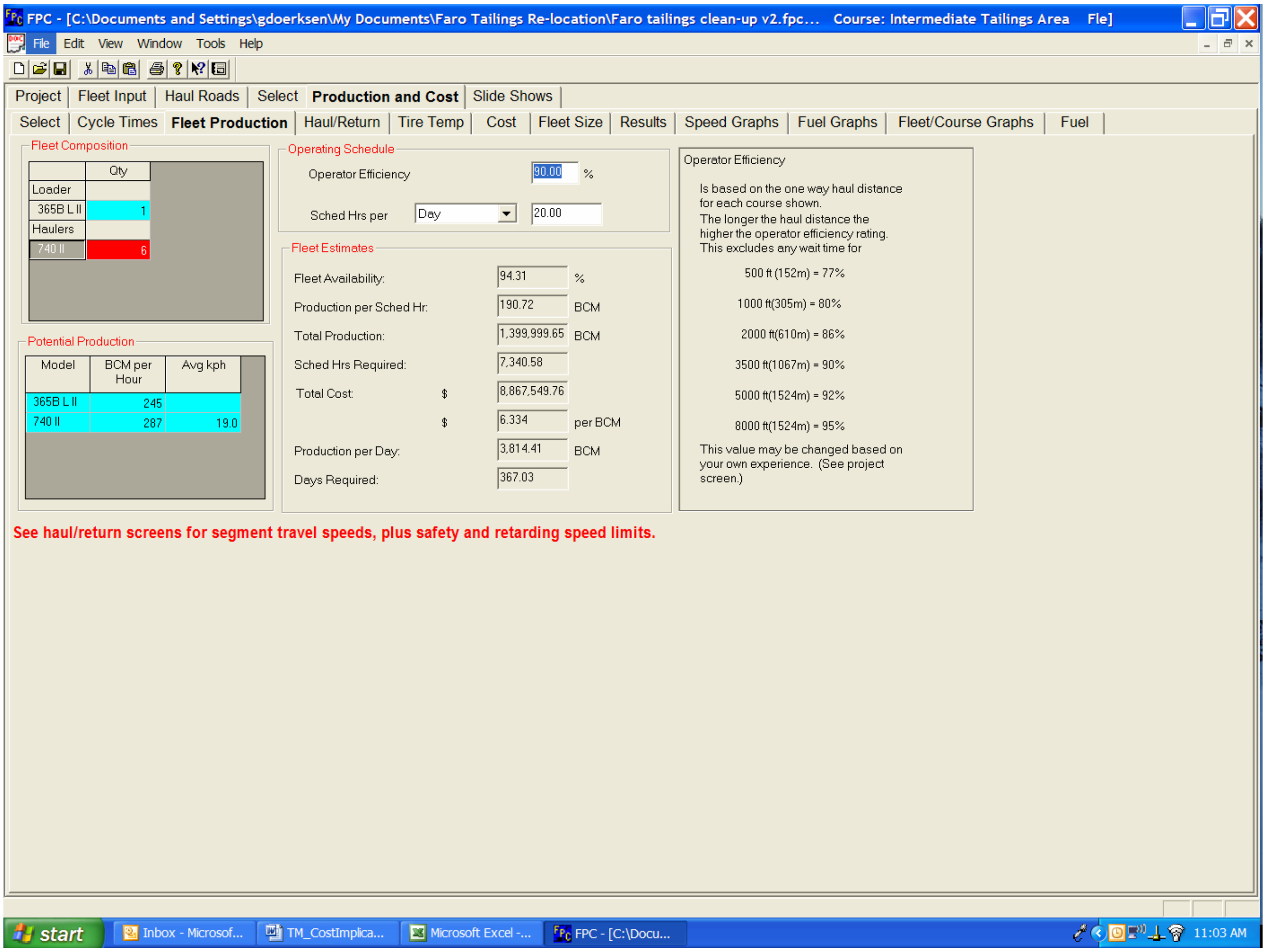
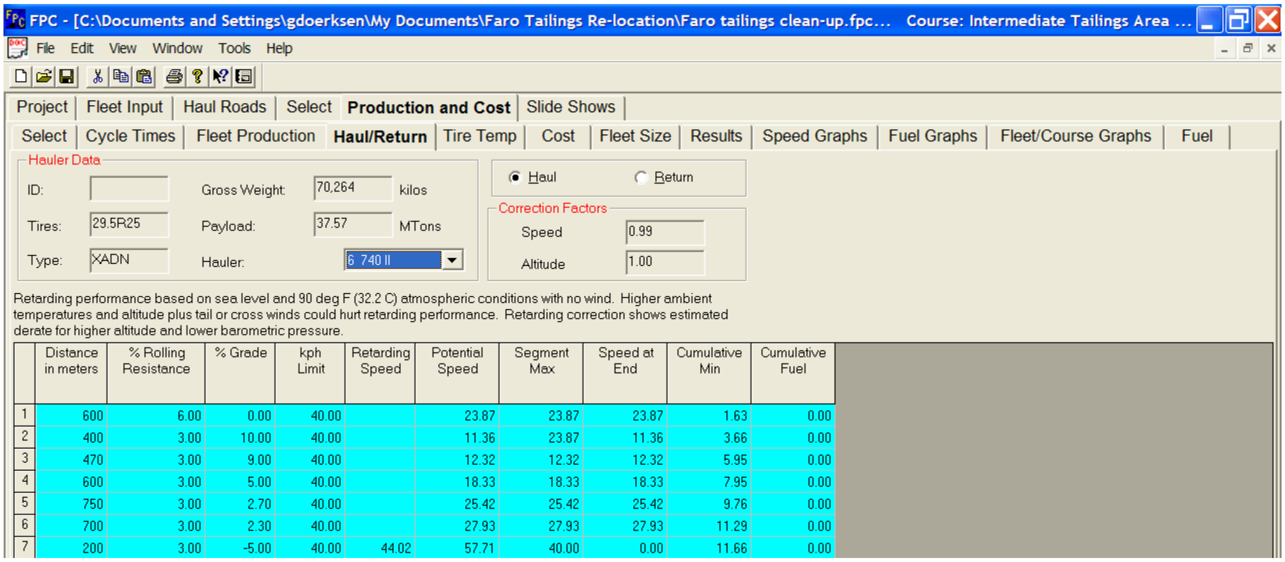
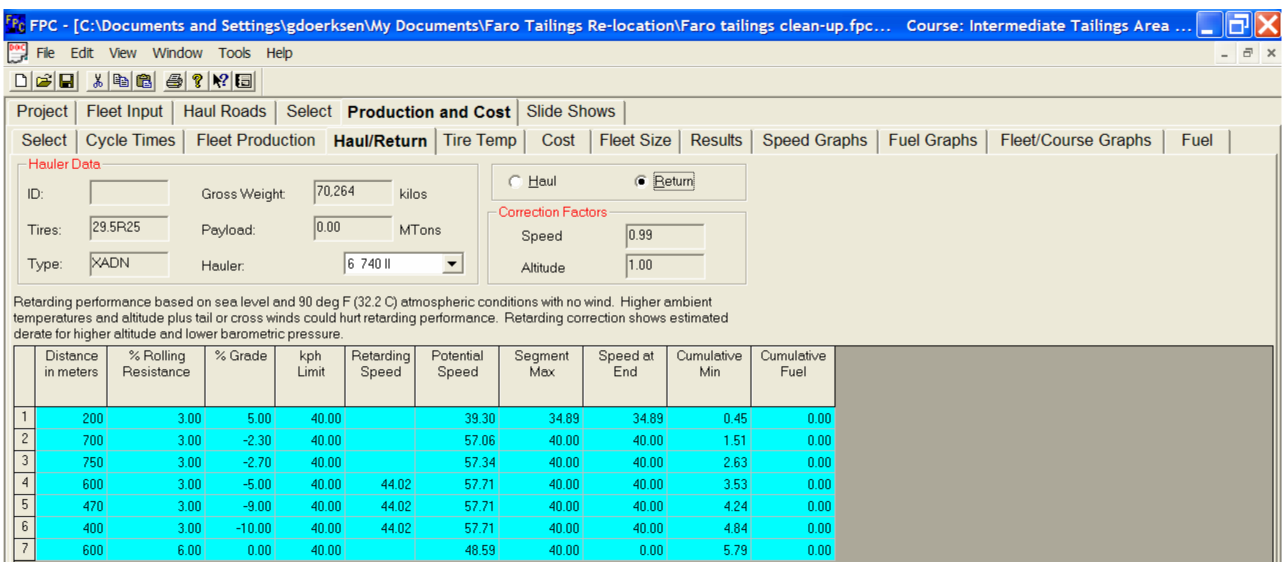


Figure A.9 FPC Fleet Production



	Distance in meters	% Rolling Resistance	% Grade	kph Limit	Retarding Speed	Potential Speed	Segment Max	Speed at End	Cumulative Min	Cumulative Fuel
1	600	6.00	0.00	40.00		23.87	23.87	23.87	1.63	0.00
2	400	3.00	10.00	40.00		11.36	23.87	11.36	3.66	0.00
3	470	3.00	9.00	40.00		12.32	12.32	12.32	5.95	0.00
4	600	3.00	5.00	40.00		18.33	18.33	18.33	7.95	0.00
5	750	3.00	2.70	40.00		25.42	25.42	25.42	9.76	0.00
6	700	3.00	2.30	40.00		27.93	27.93	27.93	11.29	0.00
7	200	3.00	-5.00	40.00	44.02	57.71	40.00	0.00	11.66	0.00

Figure A.10 FPC Haulage Cycle Details – Loaded

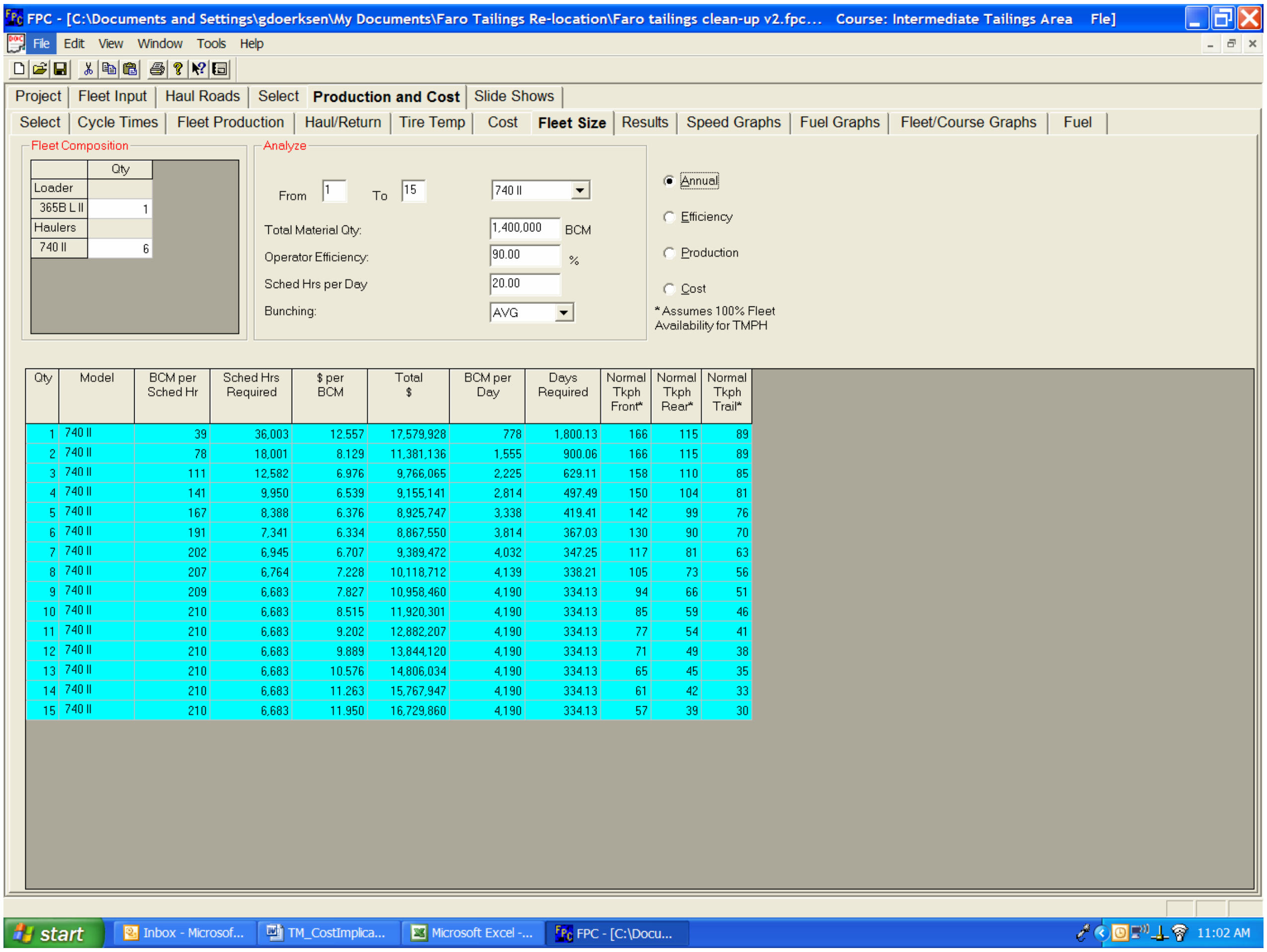


	Distance in meters	% Rolling Resistance	% Grade	kph Limit	Retarding Speed	Potential Speed	Segment Max	Speed at End	Cumulative Min	Cumulative Fuel
1	200	3.00	5.00	40.00		39.30	34.89	34.89	0.45	0.00
2	700	3.00	-2.30	40.00		57.06	40.00	40.00	1.51	0.00
3	750	3.00	-2.70	40.00		57.34	40.00	40.00	2.63	0.00
4	600	3.00	-5.00	40.00	44.02	57.71	40.00	40.00	3.53	0.00
5	470	3.00	-9.00	40.00	44.02	57.71	40.00	40.00	4.24	0.00
6	400	3.00	-10.00	40.00	44.02	57.71	40.00	40.00	4.84	0.00
7	600	6.00	0.00	40.00		48.59	40.00	0.00	5.79	0.00

Figure A.11 FPC Haulage Cycle Details – Return (Empty)

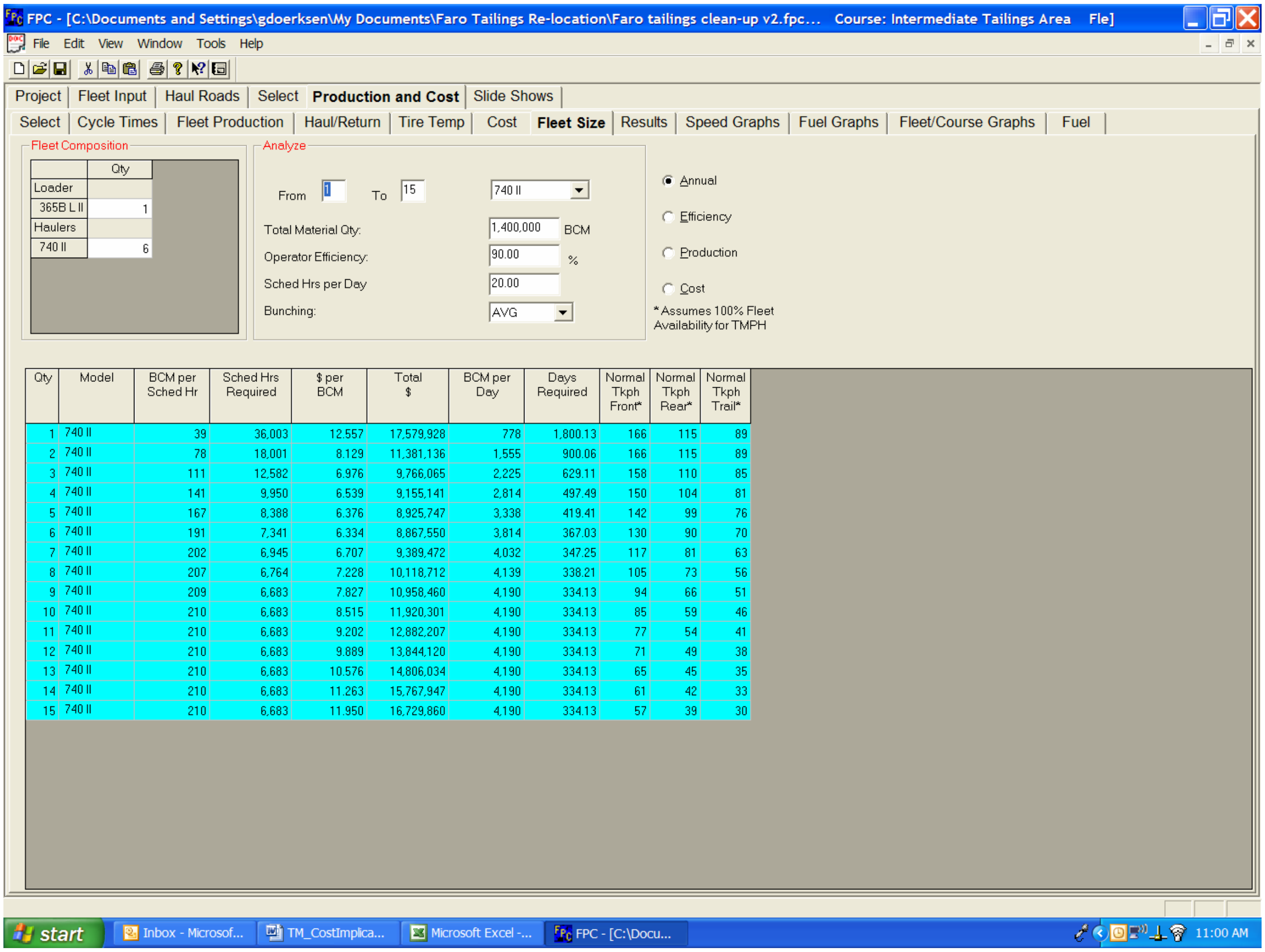
	Qty	Model	Machine Code	Hourly Cost Each Unit	Operating Hours	$ Total	$ per BCM
Loaders:	1	365B L II		239.95	6,974	1,673,304	1.195
Haulers:	6	740 II	C295	151.52	41,841	6,339,796	4.528
Totals	6				41,841	6,339,796	4.528
Support:	1	CAT 16H Grader		127.20	1,248	158,733	0.113
	1	CAT D10		286.43	2,202	630,769	0.451
	1	Fuel / Lube Truck		41.50	367	15,232	0.011
	1	Water Truck		84.66	587	49,716	0.036
Totals	4				4,404	854,450	0.610
Fleet Totals	11				53,219	8,867,550	6.334

Figure A.12 FPC Costs and Hour Summary for One Fleet (1 excavator and 6 trucks)



Qty	Model	BCM per Sched Hr	Sched Hrs Required	$ per BCM	Total $	BCM per Day	Days Required	Normal Tkph Front*	Normal Tkph Rear*	Normal Tkph Trail*
1	740 II	39	36,003	12.557	17,579,928	778	1,800.13	166	115	89
2	740 II	78	18,001	8.129	11,381,136	1,555	900.06	166	115	89
3	740 II	111	12,582	6.976	9,766,065	2,225	629.11	158	110	85
4	740 II	141	9,950	6.539	9,155,141	2,814	497.49	150	104	81
5	740 II	167	8,388	6.376	8,925,747	3,338	419.41	142	99	76
6	740 II	191	7,341	6.334	8,867,550	3,814	367.03	130	90	70
7	740 II	202	6,945	6.707	9,389,472	4,032	347.25	117	81	63
8	740 II	207	6,764	7.228	10,118,712	4,139	338.21	105	73	56
9	740 II	209	6,683	7.827	10,958,460	4,190	334.13	94	66	51
10	740 II	210	6,683	8.515	11,920,301	4,190	334.13	85	59	46
11	740 II	210	6,683	9.202	12,882,207	4,190	334.13	77	54	41
12	740 II	210	6,683	9.889	13,844,120	4,190	334.13	71	49	38
13	740 II	210	6,683	10.576	14,806,034	4,190	334.13	65	45	35
14	740 II	210	6,683	11.263	15,767,947	4,190	334.13	61	42	33
15	740 II	210	6,683	11.950	16,729,860	4,190	334.13	57	39	30

Figure A.13 FPC Fleet Size Annual Data Analysis (1 excavator and between 1 and 15 trucks)



Qty	Model	BCM per Sched Hr	Sched Hrs Required	$ per BCM	Total $	BCM per Day	Days Required	Normal Tkph Front*	Normal Tkph Rear*	Normal Tkph Trail*
1	740 II	39	36,003	12.557	17,579,928	778	1,800.13	166	115	89
2	740 II	78	18,001	8.129	11,381,136	1,555	900.06	166	115	89
3	740 II	111	12,582	6.976	9,766,065	2,225	629.11	158	110	85
4	740 II	141	9,950	6.539	9,155,141	2,814	497.49	150	104	81
5	740 II	167	8,388	6.376	8,925,747	3,338	419.41	142	99	76
6	740 II	191	7,341	6.334	8,867,550	3,814	367.03	130	90	70
7	740 II	202	6,945	6.707	9,389,472	4,032	347.25	117	81	63
8	740 II	207	6,764	7.228	10,118,712	4,139	338.21	105	73	56
9	740 II	209	6,683	7.827	10,958,460	4,190	334.13	94	66	51
10	740 II	210	6,683	8.515	11,920,301	4,190	334.13	85	59	46
11	740 II	210	6,683	9.202	12,882,207	4,190	334.13	77	54	41
12	740 II	210	6,683	9.889	13,844,120	4,190	334.13	71	49	38
13	740 II	210	6,683	10.576	14,806,034	4,190	334.13	65	45	35
14	740 II	210	6,683	11.263	15,767,947	4,190	334.13	61	42	33
15	740 II	210	6,683	11.950	16,729,860	4,190	334.13	57	39	30

Figure A.14 FPC Fleet Efficiency Analysis (1 excavator and between 1 and 15 trucks)

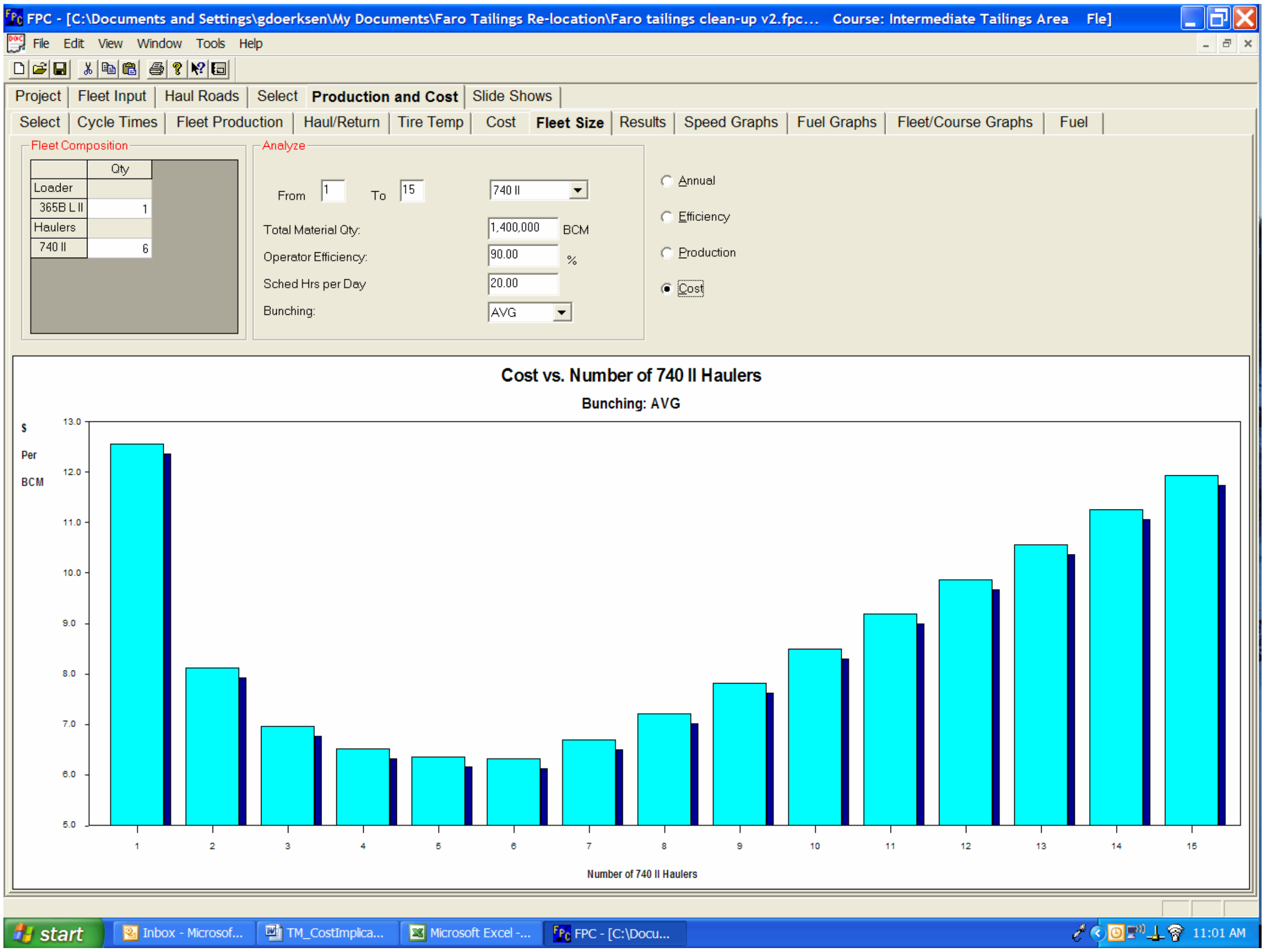


Figure A.15 FPC Fleet Cost Graph showing the lowest fleet cost is achieved with 6 trucks per excavator

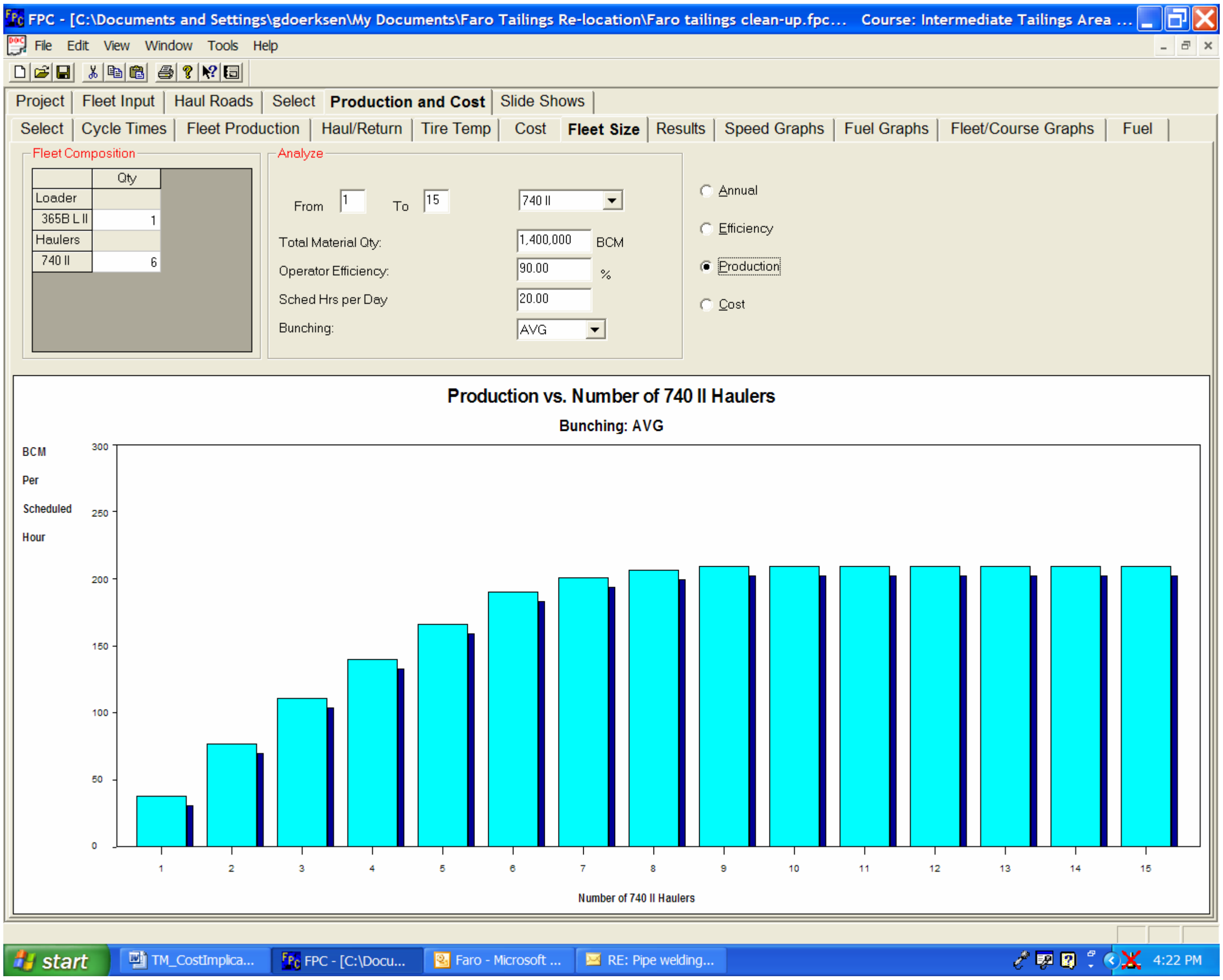
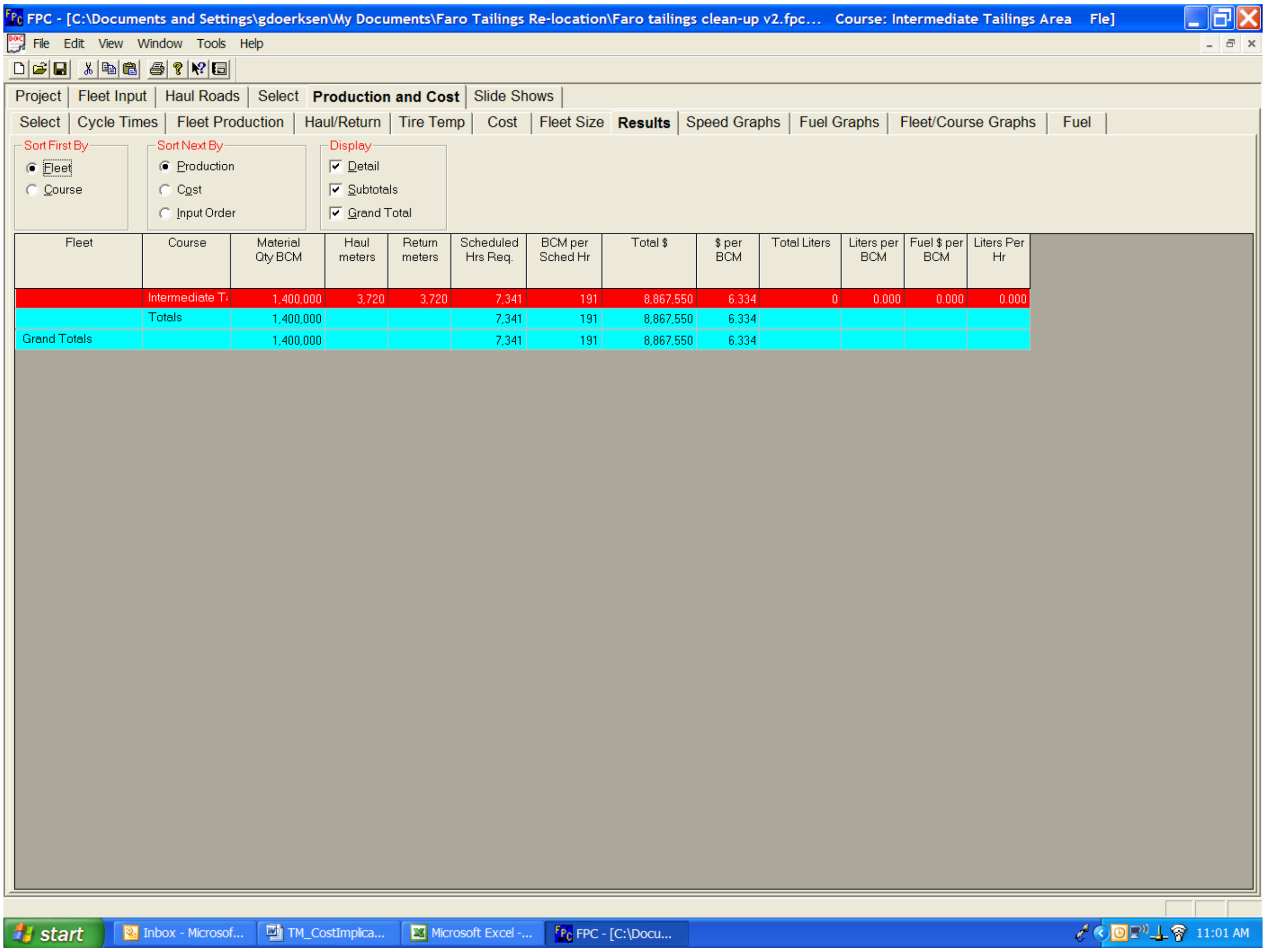


Figure A.16 FPC Productivity Graph showing a decrease in incremental productivity with greater than 6 trucks per excavator



Fleet	Course	Material Qty BCM	Haul meters	Return meters	Scheduled Hrs Req.	BCM per Sched Hr	Total $	$ per BCM	Total Liters	Liters per BCM	Fuel $ per BCM	Liters Per Hr
	Intermediate T	1,400,000	3,720	3,720	7,341	191	8,867,550	6.334	0	0.000	0.000	0.000
	Totals	1,400,000			7,341	191	8,867,550	6.334				
Grand Totals		1,400,000			7,341	191	8,867,550	6.334				

Figure A.17 FPC Fleet Cost Summary (1 excavator and 6 trucks)

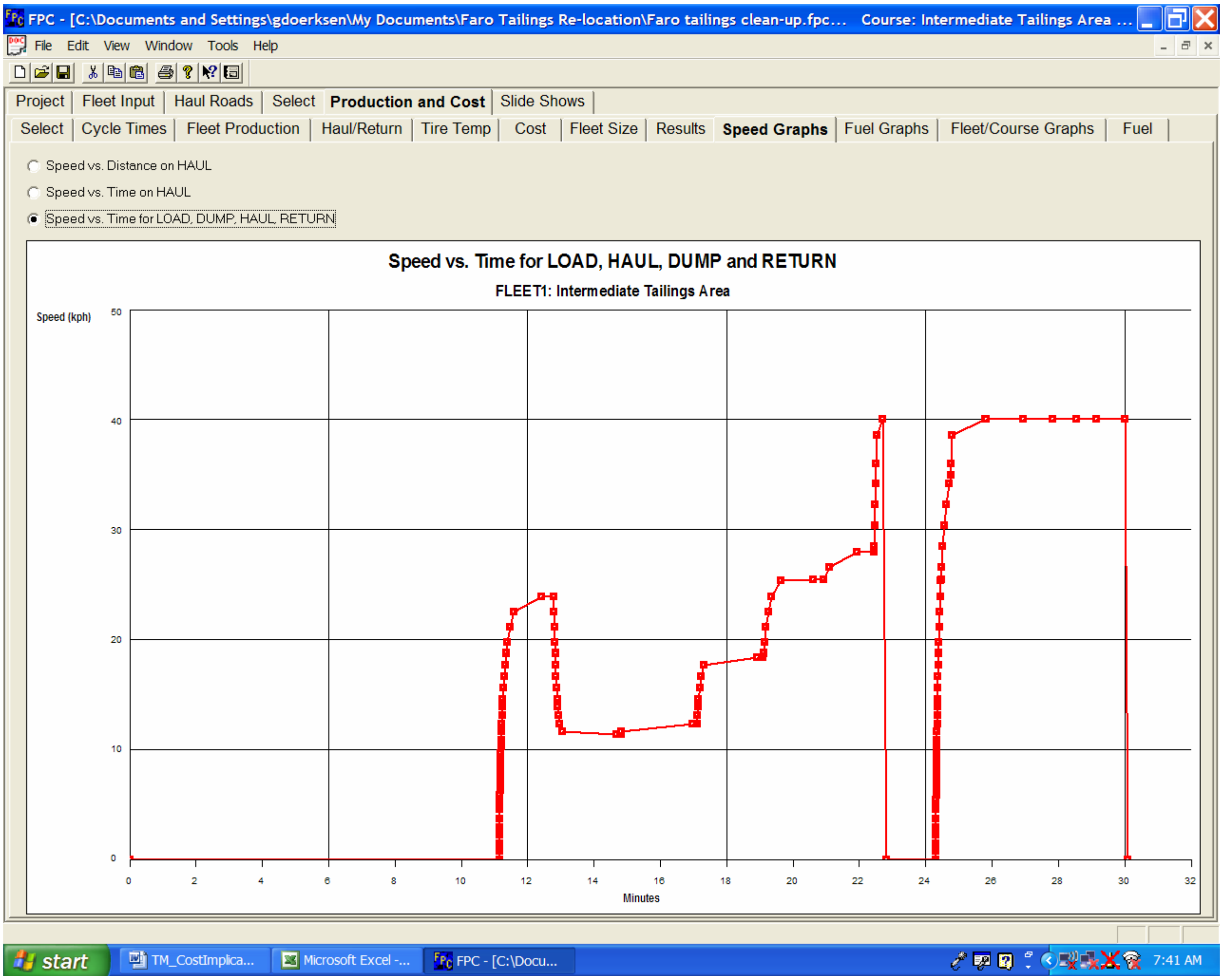


Figure A.18 FPC Truck Speed vs. Elapsed Cycle Time

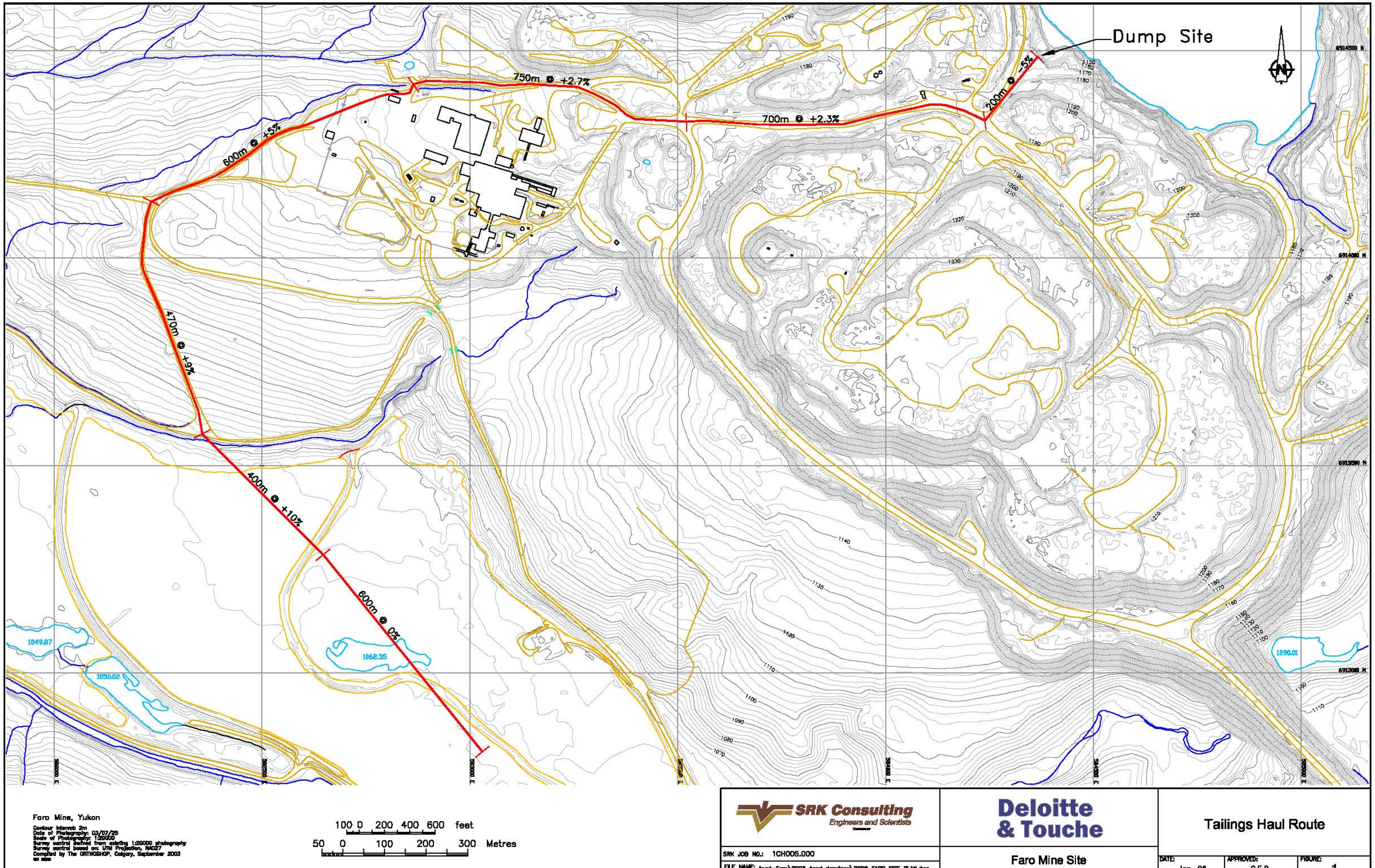


Figure A.19 Plan view of proposed haul road