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FORWARD

The management of oil sands tailings is beset with technical challenges and is an area of increasing public interest. Mature fine tailings (MFT) require long-term containment, and the appropriate management of capping waters represents the most visible of the challenges facing the industry. The vast quantities of tailings sand produced during bitumen extraction are used extensively to construct safe containment dykes that contain the MFT and capping water. Over the last 15 years, this sand has also been used to assist with the disposal of MFT in the form of consolidated/composite tailings (CT). Considerable technical work has been carried out over the last 40 years, but much of it is not easily accessible to the public, hindering the proper recognition of these extensive efforts and major advancements. Much of the scientific and engineering advancements in oil sand tailings management are published in diverse specialized conferences and journals, making it a challenge to review this important literature. In 1995, the Fine Tailing Fundamentals Consortium synthesized many of these studies. This was followed by the **International Oil Sands Tailings Conference 2008 (IOSTC'08)** at which an industrial and regulatory perspective on the needs for tailings research and management was presented in the theme lectures and proceedings. Two years later at **IOSTC'10,** approaches to meet the Energy Resources Conservation Board's (ERCB) Directive 74 were presented and discussed.

The aim of the **International Oil Sands Tailings Conference 2012 (IOSTC'12)** is to provide a further exchange of information between the people responsible for managing the oil sands tailings: researchers and providers of tailings management services who have experience with this industry. The presentations and conference proceedings will provide a venue to present the **Oil Sands Tailings Technology Deployment Road Map** prepared by the Consortium of Tailings Management Consultants (CMTC) on behalf of Alberta Innovates – Environment and Energy Solutions (AI-EES) and the Oil Sands Tailings Consortium (OSTC).

I want to personally thank the Oil Sands Tailings Research Facility (OSTRF) and the Canadian Oil Sands Network for Research and Development (CONRAD) for their encouragement and support for the conference. The conference would not have been possible without the dedication of Nicholas Beier, Vivian Giang, Ward Wilson and especially Sally Petaske who provided so much assistance and leadership.

A successful conference is only possible due to the presentations and the quality of the research presented in the manuscripts contained in the proceedings. The manuscripts become a lasting archival record and a snapshot of the state of knowledge in 2012. I want to thank our professional colleagues who willingly shared their experiences and insight with us. To all the authors, thank you for contributing your technical knowledge and for your efforts in submitting your manuscripts, especially in these extremely busy days when time is our most precious commodity. The proceedings contain information representing hundreds of years of collective experience. I know you will find insight and answers that will assist you in a better understanding of oil sands tailings.

David C. Sego

Chair, IOSTC 2012 Organizing Committee

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TABLE OF CONTENTS

SESSION 3 – CHEMICAL ADDITIVES

SESSION 4 – TAILINGS PROPERTIES MEASUREMENT

SESSION 5 – TAILINGS TECHNOLOGY ROADMAP 1

SESSION 6 - TAILINGS TECHNOLOGY ROADMAP 2 / TAILINGS MANAGEMENT 2

SESSION 7 – TAILINGS DEWATERING TECHNOLOGIES

SESSION 8 – TAILINGS TRANSPORT AND DEPOSITION SESSION 2

SESSION 9 – ENVIRONMENTAL ASPECTS

SESSION 10 – DESICCATION DEWATERING

Keynote

Presentations

COLLABORATION IN CANADA'S OIL SANDS: FLUID FINE TAILINGS MANAGEMENT

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ABSTRACT

The management of oil sand tailings continue to be one of the key environmental challenges facing the industry. In order to address this challenge, the industry has chosen to create the Oil Sand Tailings Consortium (OSTC). The OSTC is founded on four basic principles, which are intended to enable it to significantly advance the rate at which tailings ponds can be reclaimed. In addition, industry also needs to manage the requirements of the evolving regulatory regime.

Industry is advancing a "performance-based" or "outcome-based" approach as the best way to improve the management of oil sand tailings. In support of achieving an outcome-based governance approach, the OSTC has developed a set of technical guidelines. These guidelines include four deposit types that can be created from the fines management methods under active development and commercial use. In order to support the adaption of the four deposit types, performance factors have been proposed to ensure they can be effectively monitored and will achieve the desired outcome.

The Tailings Technical Guide, described in this paper, is intended to provide a path forward for both industry and regulators to ensure long term tailings management goals are achieved.

INTRODUCTION

The management of oil sand tailings continue to be one of the key environmental challenges facing the industry. During the early years of operation, industry efforts to manage tailings focused on ensuring safe containment of growing volumes of fluid fine tailings (FFT). As it became apparent that FFT volumes could not be safely managed in above ground structures, industry efforts shifted to tailings management methodologies whereby the fines were recombined with the coarse tailings. Current industry efforts are focused on developing methods to dewater the FFT such that they can be incorporated into the final mine closure landscape.

Three fundamental issues for managing fluid fine tailings throughout the operating period of oil sands mines must be addressed to create sustainable landforms for mine closure. These can be summarized as follows

1. The volume of MFT produced is substantial. At the time of writing there are approximately 850 million m^3 of fluid tailings held in above-grade containment dams.
2. The methods for transformation of fluid tailings into stable, sustainable elements of a closure landscape are all in various states of development, from preliminary research to commercial practice. None can be considered as mature (i.e. proven practice) with performance fully demonstrated for operation and closure.
3. Until recently, full commercialization of methods for fluid fine tailings management was slow. This has resulted in progressive reclamation respecting fluid tailings volumes being less than desirable to date.

In addition, the resulting process-affected water must also be managed. This will necessitate the commercial development of water treatment technologies that will ultimately enable the reuse and release of water back into the environment.

OIL SAND TAILINGS CONSORTIUM

A "coalition of the willing" came together in May of 2010 with an overall objective to jointly develop technologies to reclaim oil sands tailings ponds more rapidly. The coalition used a "principle-based" approach to put an agreement together to create the Oil Sand Tailings Consortium (OSTC). By December 2010, the OSTC was announced with the official agreement signed March 2011. All of the companies currently engaged in or considering surface mining of the oil sands are members of OSTC, including: Suncor Energy Inc., Syncrude Canada Ltd., Shell Canada, Canadian Natural Resources Ltd., Imperial Oil, Total E & P Canada Ltd. and Teck Resources Ltd.

Various collaborative projects have been undertaken since the early 2000's. However, none of them were of the magnitude of the commitment envisaged by the current OSTC Agreement. The core of the OSTC Agreement is founded on four principles: Eliminate monetary and intellectual property (IP) barriers; Share knowledge and support public transparency; Collaborate on tailings R&D; and Equitable cost sharing. These principles are described in the following four sections.

Eliminate Monetary and IP Barriers

Under the OSTC Agreement, participating companies have agreed to share all past, present and future tailings technology IP. This decision is a significant departure from the typical oil and gas industry practice. It required the more established companies (Suncor, Syncrude and Shell) who possess the majority of the industries' tailings IP and have collectively invested over $400 million over the 5 years leading up to the creation of the OSTC, to endorse the arrangement. Participation from all seven companies required a lower threshold dollar amount that the newer companies would contribute in order to support future tailings R & D.

Knowledge Sharing and Public Transparency

A number of workshops and site tours were held to facilitate the initial sharing of knowledge. The workshops covered technical experience with tailings, R&D-related information and operational experience accumulated over the past 40 years. Relationships developed among tailings professionals stemming from the workshops, led to the evolution of an oil sands tailings network. This network has improved communication between tailings personnel among the companies and helped support the technical development of junior staff.

The OSTC is also committed to being more transparent with stakeholders and to promoting broad collaboration, beyond the member companies. Through their effort to develop new tailings technologies, the OSTC will collaborate and share knowledge with universities, government laboratories, suppliers, vendors and third party technology developers.

Collaborate on Tailings R & D

True collaboration, as employed by the OSTC members, is not a common industry practice. Industry stakeholders expect more from the OSTC member companies when it comes to environmental issues. There is an expectation for the industry commit to working together in order to develop the oil sand resource using superior solutions sooner.

Collaboration is a cost effective approach to conducting business, especially when there are limited resources. Collaboration allows companies to:

- avoid duplication of work,
- share the risks associated with developing new technologies,
- improve the coordination of efforts, and
- build on each other's ideas to improve technologies and processes.

All of these points improve efficiency and economy as well as reduce time requirements.

Equitable Cost Sharing

Given the diversity of operating maturity at each of the member companies, an equitable funding approach to R & D was adopted. Companies would contribute at a level commensurate with their stage of development. Since there was no definitive formula established to define the monetary contribution, an annual review among the member companies is held to demonstrate that each company's contribution is equitable.

COSIA

On March 2, 2012, a new organization referred to as Canadian Oil Sand Innovation Alliance (COSIA) was established. It builds on the success of Canadian Oil Sands Network for Research and Development (CONRAD), Oil Sands Leadership Initiative (OSLI) and the OSTC. COSIA is now the umbrella organization for these groups constituting a much larger collaborative effort. The seven oil sand surface mining companies (OSTC members) and a similar number of in-situ operators make-up the COSIA membership. Its vision is to enable responsible and sustainable growth of Canada's Oil Sands while delivering accelerated improvement in environmental performance through collaborative action and innovation. Collaboration in COSIA will focus on four Environmental Priority Areas (EPA), including: tailings, water, land and greenhouse gases. The OSTC will now become the COSIA Tailings EPA but will still maintain the four founding OSTC principles summarized above.

OIL SAND TAILINGS REGULATORY REGIME

Two provincial government departments have primary responsibility for regulating the oil sands industry. They are Alberta Environment and Sustainable Resource Development (AESRD); and the Energy Resources Conservation Board (ERCB).

The Alberta Environmental Protection and Enhancement Act (AEPEA) is used by AESRD to execute its regulatory responsibilities in the oil sands. More recently, AESRD has released a document entitled the "Lower Athabasca Regional Plan" (LARP), which envisages a series of frameworks to address specific environmental issues, including oil sand tailings. A document referred to as the Tailings Management Framework (TMF) will set the Alberta government's policy for oil sand tailings management.

In February 2009, the ERCB issued Directive 074 (D074), which defined the requirements for regulating tailings operations associated with mineable oil sands. The directive specified performance criteria for the reduction of Fluid Fine Tailings (FFT) and the formation of trafficable deposits.

Tailings Roadmap Study

Concurrent with the introduction of D074, industry increased its efforts to accelerate commercial development of new tailings management technologies that were in various stages of development. Following the inception of the OSTC, a joint initiative between industry and government was undertaken to support a broader strategy for sustainable management of tailings produced by the oil sands industry. This collaboration is known as the "Technology Deployment Roadmap and Action Plan for 'End-To-End' Solutions for Oil Sand Tailings". The project was completed July 1, 2012 and the final report contains a comprehensive and up-to-date compilation of information along with key recommendations for furthering tailings research and development activities. The Technology Deployment Roadmap and Action Plan is the subject of a separate keynote address (Sobkowicz, 2012) and a special session at this conference and will not be discussed any further here.

Outcome Based Governance of Oil Sands Tailings

Industry is committed to improving the management of oil sand tailings and believes that the best approach to doing so lies in utilizing an "outcome-based approach". Using this approach, regulation of the industry would focus on the achievement of performance outcomes that are aligned with final Mine Closure Plans, using interim measures to ensure that various performance milestones are achieved en route to the final requirements.

Joint industry/government efforts are underway to improve the governance of oil sand tailings. The three initiatives listed earlier(i.e. the Tailings Management Framework, Directive 074 and the Tailings Roadmap Study), along with a "Tailings Technical Guide", are all viewed as important components of a governance process that will support an outcome-based approach. Figure 1 illustrates how these three initiatives are expected to be utilized to achieve an outcome-based governance approach for oil sand tailings

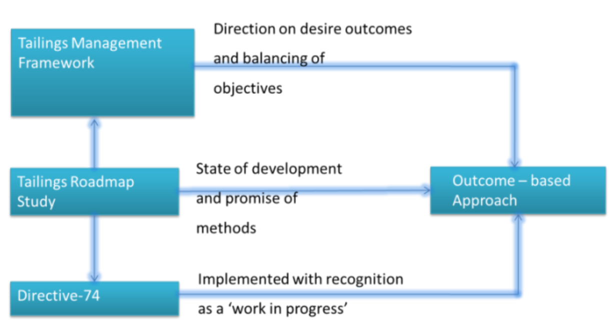

Figure 1. Outcome based approach for Tailings Governance (modified from Fair, 2012).

TAILINGS TECHNICAL GUIDE FOR FLUID TAILINGS MANAGEMENT

In support of improving the governance of oil sand tailings, the OSTC set out technical guidelines for managing FFT. This document entitled “Tailings Technical Guide for Fluid Tailings Management” was developed with oversight and guidance provided by two expert panels. The first panel was convened to review the technical content of the document. The panel members were David Carrier III, Richard Dawson, Ross Eccles, Norbert Morgenstern and John Sobkowicz. A second panel reviewed the document from a regulatory perspective. The panel members were Richard Dawson, Gerry DeSorcy, John Errington, Barry Hurndall, Bernie Roth and John Sobkowicz.

The Technical Guide is intended to support the Government of Alberta in developing a consistent policy for tailings regulation. It also provides a detailed up-to-date technical review of current practice for managing the different types of tailings deposits using best available technology.

Of most importance, this document proposes that site-specific volume profiles of FFT be established for each mine site. This approach provides a direct method to manage and steward the volume of FFT. Futhermore it will limit the accumulation and provide containment of FFT in a manner consistent with the goals of progressive reclamation as well as the desired reclamation and closure outcomes.

Under this proposal, oil sand operators would employ adaptive management to remain within their committed volumes. Adaptive management deals with inherent uncertainties associated with FFT generation, allowing operators to deploy available methods (and newly developed ones like those identified in the Tailings Roadmap Study) as required.

The following sections are excerpts from the Technical Guide describing the current state of practice and provide an accurate summary of deposit characteristics that can be formed and managed using current technology. Different technologies are available that form combined suites to meet various performance objectives within the overall tailings plan for a project.

OIL SAND TAILINGS PROCESSING

Various process methods are currently being utilized to release water from FFT. Figure 2 provides a perspective on volume reduction through dewatering of FFT, which compares percent solids by weight versus the equivalent volume percentage. FFT must attain a solids content of 75% to 80% (by weight) to develop sufficient long-term stiffness and strength (in the range of 50 kPa to 100 kPa), losing 67% to 75% of its water in the process. For tailings treatment technologies involving drying, FFT might further dewater as far as the shrinkage limit.

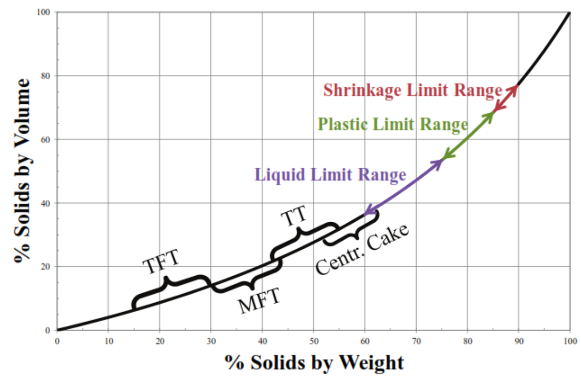

Figure 2. Tailings Volume vs. Tailings Mass (modified from OSTC, 2012).

Centrifugation of FFT

One process to dewater FFT uses flocculation and processing of FFT through a solid-bowl scroll centrifuge. Adding a coagulant such as gypsum can also assist the process. Solids contents of about 55% are produced in the centrifuge “cake.” The cake is deposited in relatively thin lifts of about 2t/m^2-y in constructed cells, in a manner similar to the handling of soft, wet overburden soils. Left for a winter freeze-thaw cycle, the cake will attain peak shear strengths of 5 kPa to 10 kPa, before an additional lift is placed. Alternatively, the cake is continuously deposited, at higher annual rates per area, into deep, in-pit deposits, relying on self-weight consolidation to effect further water release and volume reduction.

Thin Lift Dewatering

A second method employs in-line flocculation of FFT and discharge of the flocculated slurry in thin lifts into cells, where initial dewatering, effected by flocculation and drainage, can increase the solids content to around 60%. Further water removal is accomplished via evaporation and freeze-thaw effects. The volumes associated with oil sands mining and the low net evaporation rates in northern Alberta result in large area requirements to meet dewatering targets for reclamation. The dewatered material can be relocated to overburden cells after initial dewatering (similar to centrifuge cake), or alternatively, allowed to dewater further with evaporation or freeze-thaw to a point where it has sufficient strength to form an integral part of a disposal structure.

Thick Lift Dewatering

In-line flocculated FFT can also be utilized to form deep deposits (e.g., in a large in-pit cell). Water expressed from the deposit and precipitation is decanted from the surface. Surface dewatering can be assisted by rim-ditching the perimeter of the deposit or creating channels on the surface to direct water to a decant sump. Self-weight consolidation progressively increases the solids content of the deposit, driving water upward through the deposits (or both upward and downward if there is bottom drainage).

Thickened Tailings

A fourth dewatering method draws FFT directly from the extraction process (e.g., cyclone overflow), and then flocculates and thickens the FFT in a mechanical thickener. Thickening is generally employed to recover thermal energy but also has the benefit of partially dewatering the FFT, producing thickened tailings (TT). The TT can be placed in deep deposits, relying on consolidation for dewatering. Alternatively, it could be discharged in thin lifts, in a similar manner to that described previously.

CT or NST Tailings

Blending of sand slurry (typically at high solids contents) with FFT, using flocculants or coagulants to attain a non-segregating mix can also be used to promote fines capture and dewatering. Once mixed, the material is then discharged into a deep deposit. Where the fines are sourced as MFT, the resulting product is referred to as composite or consolidating tailings (CT). Alternatively, where fines are sourced as TT, the resulting product is referred to as non-segregating tailings (NST). The key objective of both methods is to reduce the water content and produce a sand-dominated mix, at a moderately high SFR. This results in a relatively quick volume reduction and increase in deposit strength (compared to lower SFR tailings deposits).

OIL SAND TAILINGS DEPOSIT TYPES

There are essentially four deposit types that can be created from the fines management methods under active development and commercial use. These deposits include: thin-layered, fines dominated deposits; deep, fines dominated deposits; fines-enriched sand deposits; and water-capped fine deposits. The following sections describe the different ways in which these four oil sands tailings deposit types are produced, their important performance factors and how the deposit performance can be assessed through the period of their placement to their readiness for reclamation.

Thin-Layered, Fined Dominated Deposit

This deposit type consists of a fine tailings stream that is discharged sub-aerially into a disposal site in thin lifts (typically 100 – 500 mm thick, depending on the process used to produce the discharge stream). It relies initially on dewatering by chemical and mechanical treatment, and later on, dewatering by drainage, atmospheric evaporation and freeze-thaw. It is viable on a commercial scale where sufficiently large areas are available for surface drying to the target solids content. These dewatered tailings can either be relocated to an overburden disposal site or left in place as part of a multi-layer deposit. In some cases, the soft material can be placed in polders in overburden disposal structures that provide the necessary integrity, without the need for a large reliance on the dewatering mechanisms described above. The precise tolerance for land use and for incorporation of weak material into overburden dumps will, by necessity, be a site-by-site determination, due to the highly variable conditions between leases.

There are three processes currently deployed or under development to create this deposit:

1. Thin-lift dewatering of in-line flocculated MFT, currently operating at a commercial scale at Suncor and in large scale prototype at Shell/MRM.

2. MFT centrifugation (MFT-C), undergoing large scale prototype testing at Syncrude.
3. Thin lift deposition of TT, which has been piloted at a relatively small scale.

Regardless of whether the deposit constitutes a multilayer deposit or is relocated to an overburden disposal structure, the primary performance factor is **undrained shear strength**. This indicator is suitable to monitor and demonstrate an acceptable trajectory towards reclamation.

Additional information and indicators would also be typically collected to support process control, other operational requirements such as mud-farming and geotechnical stability.

If thin-lift-fines-dominated deposits do not achieve their predicted performance, several design, operating and post-deposition contingencies are available or under development. The options currently available to operators are briefly summarized below:

- Increase drying time, or decrease lift thickness (i.e. larger cell)
- Utilizing alternative cell designs
- Improving process control and placement strategies (i.e. thinner lifts, mechanical spreading)
- Drainage layers to enhance dewatering
- Mud-farming to maximize evaporation
- Re-grading or re-constructing erosion and drainage features
- Over building to accommodate settlement

These methods should allow the fines-dominated deposits to achieve the desired performance.

Deep, Fines-Dominated (Cohesive) Deposits

This deposit type consists of a FFT stream that is discharged into a deep disposal site, which accumulates a significant thickness over time. The sand to fines ratio (SFR) of this material is typically below 2 and the fines are moderately to highly plastic. Initial water release is accomplished with a polymer flocculant process. The balance of water release/volume reduction occurs through self-weight consolidation and creep, (for very deep in-pit deposits, it might make sense to insert sand layers or geo-drains to enhance dewatering rates). Environmental effects (evaporation and freeze-thaw) play a minimal role in dewatering, except for surface crust development when filling is complete. Once a sufficient crust has developed, the surface is capped, typically with sand, to provide both a load, to assist the consolidation of the upper part of the deposit, and a substrate for topsoil. Deep fines-dominated deposits are generally favoured where in-pit area and volume are available. Less deep, out-of-pit deposits in a containment structure might be necessary for new mine start-up, or for example, could be attractive as polders in large out-of-pit sand storage areas. Deep, fines-dominated deposits represent a very efficient land end-use with geotechnically secure containment.

There are three processes currently deployed or under development to create this deposit:

1. TT made from sand depleted tailings streams fed to a thickener vessel and flocculated with a polyacrylamide flocculant. TT has been piloted by Shell and Syncrude and is under consideration by other operators.
2. Accelerated dewatering uses in-line flocculation of MFT followed by deposition in a deep containment area. Field pilot tests and a large scale prototype are underway at Syncrude.
3. MFT Centrifugation as described previously, but placed in deep deposits. Minimal data is available for this option at this time, but the deposit is expected to behave similarly to TT.

Consolidation time for deep fines-dominated deposits is affected by a number of factors, the most important of which are the character of the material, the rate of deposition per area and the overall depth of the deposit. It requires considerable technical effort to predict consolidation rates for design and to monitor performance through the operations and reclamation cycle. Simply stated, the major goals are to predict the time-dependent capacity of a containment area, (to properly size the area to accommodate the anticipated volume of fines-dominated material), and to analyze the settlement rates under surcharge for reclamation needs. Thus, the important performance factor is the **increase in solids content (or volume reduction) with time**.

Characteristic of deep cohesive deposits is the uncertainty over their precise rate of consolidation, volume reduction and strength attainment. Contingent measures can be applied at all stages, from planning and design through to reclamation. A conservative approach to design can limit the impact of consolidation underperformance. Improved process controls or reducing the rate of deposition can enhance the operational success. Once the deposit is placed, additional surcharging or addition of wick drains can be used to improve the rate of consolidation or volume change.

Fines-Enriched Sand

Fines-enriched sand typically has an average SFR from about 3 to 5 (can also be somewhat higher), a pipeline solids content ranging from 55 to 60%, and a solids content after deposition in excess of 70%. These deposits are typically formed using processes such as CT and NST that are intended to capture the fines within sand voids during deposition, thus managing the inventory of FFT with minimal re-handling. To form these deposits, adequate sand and containment are required. Coupling the process with bitumen production often leads to operational constraints that must be considered before selecting this type of deposit.

CT is currently the commercial process used for forming fines-enriched sand tailings deposits. NST is a variation of CT where the fines are supplied from a thickener rather than MFT from a tailings settling pond. The CT process is currently, or has been, operated at a commercial scale at Syncrude and Suncor. Shell and CNRL both plan to implement CT or NST technologies as part of their tailings management plans.

The deposit can be characterized by two primary metrics for assessing conformity of the operation with the plan: **SFR distribution after deposition and solids content trajectory in response to surcharge** (e.g., additional CT layers or sand cap). Field sampling, laboratory measurements and *in-situ* measurements are required to support the performance monitoring.

Mitigation plans need to be part of the tailings management plan, and include proposed “triggers” to guide when mitigation would be implemented. If fines-enriched sand deposits do not achieve the required performance, depending on the stage of the deposit and deviations encountered, options could include:

- Improving process and operational controls
- Improving depositional technology
- Further dewatering at the tailings facility prior to deposition
- Promote increased dewatering in the deposit through a combination of wick drains, coke-capping, sand raining capping, or winter capping.
- Re-grading or re-constructing erosion and drainage features
- Over building to accommodate settlement

These methods will allow the fines-enriched sand to be managed and monitored, and achieve the performance criteria established at each site.

Water-Capped Deposits

This deposit type consists of placing FFT that has naturally densified to >30% solids content into an engineered mine pit where a water cap is established to form a lake. Once acceptable surface water quality is attained, in-flow to and out-flow from surrounding terrain is established to emulate a natural lake system. In a variation of this method, fine tailings would be densified before placement, thereby increasing the disposal capacity of the mine pit. While these deposits consolidate and gain strength over a long period as soft lake bottom mud, they are not intended to support terrestrial reclamation features.

One of the main types of FFT that are envisaged for water-capped deposits is MFT. The two key issues associated with incorporating un-amended MFT in a reclamation landscape are:

- The volume of MFT produced is substantial
- MFT is a fluid with a composition that is predominantly water; densification to fully consolidated clay is currently projected to take hundreds of years

When the water-capping concept was introduced, researchers recognized several key functional aspects that required empirical knowledge and demonstration before such a reclamation component could be accepted. These included: a) stability of the pond layers, b) interaction of the pondwater with the groundwater, c) flux across the water cap/tailings interface, d) littoral zone development, e) toxicity to aquatic life, and f) ecological development. Over the last two decades a program of progressive monitoring and experimentation, using laboratory, field and modeling methods at Syncrude were implemented to address these questions. Where possible, questions have been addressed with scientific study in real systems at a reduced scale from a full-size lake. Syncrude is currently in the process of initiating a full-scale, demonstration system at the Mildred Lake site.

As **water quality** is fundamental to attainment of the proposed reclamation end point, it is the primary performance factor for this technology. A measurement and monitoring program is also required to validate application of the technology. It should compare actual facility performance with the developmental “trajectory” of the six functional aspects proposed by the operator.

Fluid Fine Tailings Management

To this point, the various methods were addressed for treatment and deposition of FFT, so that the treated material is, in many cases, no longer a fluid and in all cases incorporated into the closure landscape. However, containment and control of FFT through the period of active mining and at mine closure is also a key risk management matter. The following discusses the management of the remaining volumes of FFT, that are not captured in one of the previously discussed deposit types and that are considered acceptable for storage on the lease without further treatment.

Through the life of mine, fluid fine tailings and overlying oil sands process-affected water (OSPW) must be safely contained. This aspect is currently well managed through internal procedures overseen by the Alberta Dam Safety Branch and industrial practice. However, in the long-term, it is accepted that there will be no above-grade storage of FFT. Volumes of FFT will be maintained within a profile consistent with site plans submitted for project approval and updated with mine plans as mining progresses.

To the extent practical, this profile should reflect the principle of progressive reclamation, meaning that volumes should not be accumulated such that there is a large and costly liability at mine closure. Adaptive management measures will be available to maintain FFT volumes within committed limits. In practice, this will mean adjusting the rate of FFT treatment and disposal methods to control final volumes of FFT. Implicit in this approach is that reactive, unplanned increases in the storage containment volume will not be the response to under-prediction of FFT generation. Proven methods will be developed for treatment and disposal of FFT consistent with reclamation planning and execution during operations and at mine closure.

To demonstrate that past and predicted future accumulations of FFT are within submitted tailings plans and committed limits, measuring and reporting are required by the operators. To be assured that this is the case, operators must fully understand their entire tailings mass balance including:

- the rate at which the FFT is being generated
- the rate at which FFT consolidates in the ponds
- the volumes of FFT that are treated and stored in an approved Designated Disposal Area (DDA).

Given the degree of uncertainty in net generation of FFT, the primary design contingency is to take a conservative approach to forecasting FFT generation. The degree of conservatism will be determined by each operator, recognizing:

- Degree of variability in the ore body
- Site-specific constraints on the practicality and cost of contingency volume
- Ability to respond in operations by expanding dewatering processes that can offset any underestimates of containment and accumulated FFT volumes.

The response in operations to an increase in FFT beyond the planned trajectory, once all operational improvements have been exhausted, would be to expand the resources for dewatering of FFT, such as discussed in previous sections of this paper. As the end of mine life is approached, if the volume of FFT to be stored on the site were to exceed the amount that could be acceptably accommodated according to the closure plan, the FFT could be treated to increase its density and therefore storage efficiency.

Summary of Deposits

The Tailings Technical Guide sets out guidelines for managing FFT through appropriate treatment and disposal in a DDA. For each site, operators must consider land availability and disturbance, geotechnical conditions, resource distribution, general site geology, containment availability and mine advancement to develop the optimum tailings management strategy. Table 1 shows a summary of the different attributes for each of the four deposit types. The different characteristics for each deposit type highlight the importance of matching performance measures with particular deposit types.

PERFORMANCE MEASURES

The current ERCB D074 requires operators to reduce FFT through fines captured in DDAs and to “form and manage” these DDAs using strength criteria Essentially, the D074 performance criteria favors thin-layered, fines-dominated deposits despite the high cost, large footprint and high energy intensity requirement. Consequently, the regulation has resulted in tailings management practices that are out of alignment with advancing technologies and improved practices.

The Tailings Technical Guide establishes the requirement to adapt D074 regulations to suit current best available practices and emerging technologies. These proposed D074 revisions focus on the specific performance criteria for FFT reduction and for managing DDAs so there is consistent alignment with emerging technologies and current practices. Table 2 provides a summary of performance measures for each of the potential deposit types.

Industry continues to support a regulatory framework that is performance-based and requires an annual measurement and reporting cycle to demonstrate compliance. Revising the regulations so there is better alignment with deposit types and technology developments will maintain this requirement while recognizing values that are important for sustainable oil sands mining including resource recovery, cost, progressive reclamation, energy use, and land disturbance.

CONCLUSIONS

The OSTC’s Tailings Technical Guide provides an up-to-date technical overview of current practice in oil sand tailings management orientated towards the different types of deposits formed and managed using best available technology. These deposit types include: thin-layered, fines-dominated deposits; deep, fines-dominated deposits; fines-enriched sand deposits; and water-capped fine deposits.

The Technical Guide also suggests updates to D074 that would promote better tailings management given recent technology developments and changes in current practice. These changes are proposed in the context of the original intent to provide a performance-based regulation that builds on a foundation of continuous improvement.

Detailed site-specific mitigation plans are important to the proposed adaptive management approach. In many cases, the contingencies are still in a research or developmental stage. Hence the need for an adaptive management plan, whereby new insights are continuously incorporated in future designs and applications.

Industry acknowledges, and continues to be greatly influenced by D074 and has expended significant effort (resources, time and money) on solutions to the tailing challenges that the industry faces.

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Table 1. Oil Sands Tailings Deposit Attributes.

Deposit Type		Cost	Area Footprint	Containment Volume Requirement	Energy Intensity	Critical Technical Components
Thin-Layered Fines Dominated	No Re-handle	High	Very Large	Minimal	High	-Initial Dewatering -Drying area/time -Deposit working
	Re-handle	Very High	Large	Minimal to Low	Very High	
Deep Fines Dominated		Moderate	Low-Moderate	Low	Low	-Initial dewatering -Consolidation
Fines Enriched Sand		Moderate	Moderate	High	Low	-Minimizing segregation
Water-Capped In-pit Deposits		Low	Low-Moderate	Medium	Low	-Water quality with time

Table 2. Deposit Performance Factors.

Deposit Type	Primary Performance Factor	Comment
Thin-Layered Fines Dominated	Strength	Consistent with current D074 criteria
Deep Fines Dominated	Volume change with time	Focuses on dewatering behavior consistent with FFT reduction
Fines Enriched Sand	Volume change with time and Sand/fines ratio	Focuses on dewatering behavior and fines segregation behavior. Fines enriched sand deposits have overall (sand and fines combined) lower bulking factors.
Water-Capped In-pit Deposits	Water quality	Key metric in support of lake ecosystems.

THE OIL SANDS TAILINGS TECHNOLOGY ROADMAP AND ACTION PLAN: INTRODUCTION AND KEY RECOMMENDATIONS

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ABSTRACT

In 2011, Alberta Innovates – Energy and Environment Solutions (AI-EES) awarded a contract to the Consortium of Tailings Management Consultants (CTMC) to prepare an Oil Sands Tailings Technology Roadmap. The project objective was "…to create a technology deployment roadmap and action plan that will assist regulators and industry to create and implement technology solutions that will meet the goals of Alberta Environment's (AENV's) Draft Tailings Management Framework and ERCB's Directive 074".

The Project had four main components: 1) to identify and describe all known tailings management technologies, through all stages of the mining life cycle, 2) to define the important tailings reclamation objectives to which successful tailings technologies should contribute, 3) to evaluate the identified tailings technologies to determine their strengths and weaknesses, in light of these objectives, and 4) to identify technologies and/or suites of technologies which could improve the ability of tailings management practices to meet the previously defined goals, and the pathways by which they could be brought through the research and development process to commercial implementation.

This paper describes the objectives of the Project, the technical process whereby these goals were met, and the key recommendations arising from the Project work. Associated papers will describe in more detail specific components of the Project.

INTRODUCTION

General

Oil sands mines near Fort McMurray, Alberta, Canada produce large quantities of solid and fluid tailings. Mining and tailings production have been ongoing for over 40 years, and are expected to continue, at increasing rates, in the coming decades. There are ongoing concerns about the geotechnical risks, environmental risks, and long-term liability related to tailings production. In particular, there are concerns related to production, storage, and reclamation of fluid tailings.

In response to some of these concerns, Alberta Innovates – Energy and Environment Solutions (AI-EES), in partnership with the Oil Sands Tailings Consortium (OSTC), in 2011 awarded a contract to the Consortium of Tailings Management Consultants (CTMC) to undertake an integrated project called "The Technology Deployment Roadmap and Action Plan for 'End-to-End' Solutions for Oil Sands Tailings". The project objective was "…to create a technology deployment roadmap and action plan that will assist regulators and industry to create and implement technology solutions that will meet the goals of Alberta Environment's (AEW's) Draft Tailings Management Framework and ERCB's Directive 074".

Project Involvement

Specific Alberta government departments and organizations participating in the study were:

- Alberta Innovates – Energy and Environment Solutions
- Alberta Energy
- Alberta Environment and Sustainable Resource Development
- Energy Resources Conservation Board
- CANMET

Member companies of the OSTC involved in the study were:

- Canadian Natural Resources Limited
- Imperial Oil
- Shell Canada Energy
- Suncor Energy Inc.
- Syncrude Canada
- Tech Resources Limited
- Total E&P Canada Ltd.

A consortium of leading oil sands tailings management engineering firms was formed to carry out the project proposed by AI-EES. This unified team of tailings management consultants and academics provided a forum for developing a Roadmap that was based on the best available expertise and enjoyed broad endorsement. The CTMC includes:

- AMEC
- BGC Engineering
- Golder Associates
- Klohn Crippen Berger Ltd.
- Norwest Corporation
- Thurber Engineering Ltd.
- The University of Alberta Geotechnical Group.

Project Goals

The CTMC Project Execution Plan (2011) defined the Project goals. The "Technology Deployment Roadmap and Action Plan for 'End-To-End' Solutions for Oil Sand Tailings" (TTD Roadmap) is an initiative of both the government and industry to support a broader strategy for sustainable management of tailings produced by the oil sands industry. The Tailings Roadmap / Action Plan initiative will provide a framework to government and industry that will:

1. Help achieve more timely deployment of the end-to-end tailings technologies, and share the results and knowledge of tailings deployment activities.
2. Document the current state of tailings reclamation technology to define technology pathways to reach the end goal.
3. Serve as a basis for accessing government and industry funding to accelerate commercial scale demonstration of technology, and promote sharing and technology transfer.
4. Identify technology options and establish a framework for operators to conduct detailed feasibility studies and deploy technology, and allow regulators to verify the performance during this process.
5. Promote a collaborative approach to oil sands tailings technology that expedites technology deployment, reduces environmental impacts beyond the boundaries of the mine lease and enhances public trust.
6. Provide a medium for sharing the results and knowledge of effective tailings deployment initiatives.

The Project Report contains 5 volumes, each volume with its own set of Appendices. Volume 1 summarizes the results of all of the Project work, and draws various conclusions and recommendations for the Project, from which this paper is derived. Volumes 2 to 5 contain separate, detailed reports that have been issued for each component of the Project.

All Project Report volumes are available on-line at:

http://ai-ees.ca/reports.aspx or http://www.cosia.ca/media-resources/resource-library/

Project Components

The project was divided into four main components, each of which had a component lead, assistant leads and supporting specialists that provided input to all aspects of the work scope. The specific tasks of the four components can be summarized as follows:

- Component 1 (C1) searched for, collected and compiled information on technologies in current use or in a research or development stage that had potential application in the oil sands industry. They also compiled a summary of current tailings management practices. BGC Engineering and Norwest Corporation carried out this component.
- Component 2 (C2) developed a set of Tailings Management and Reclamation (TMR) Objectives and Sub-objectives, to be used in developing criteria for the evaluation of each of the technologies. Klohn Crippen Berger Ltd carried out this component.
- Component 3 (C3) set up and carried out a review process to evaluate each of the technologies, in light of the TMR Objectives, and to identify gaps, strengths and opportunities within the technologies. Golder Associates Ltd. and the University of Alberta carried out this component.
- Component 4 (C4) identified "highlighted" tailings technologies that would improve existing tailings suites or contribute to new suites, and then developed TTD Roadmaps for these highlighted tailings technologies, to bring them to a stage of commercial imple-

mentation. Thurber Engineering Ltd. and AMEC carried out this component.

The Project work was carried out under the direction of a Research Advisory Committee (RAC), consisting of representatives of government, industry and the CTMC, led by a Research Director (Dr. John Sobkowicz, P.Eng.).

Throughout this paper, each component of work will be referred to as “C1” for “Component 1”, etc.

PROJECT HIGHLIGHTS

The Technology Discovery

C1 identified 549 tailings technologies that have potential use in the oil sands industry. With refinement, the initial 549 technologies were reduced to 101 unique technologies (plus some technology variations and enhancements); these are discussed in more detail in a companion paper in this conference (“The Oil Sands Tailings Roadmap and Action Plan: Oil Sands Tailings State of Practice Overview”).

The technologies were categorized by their position in the mining life cycle (mining, extraction, tailings processing, tailings capping/deposition, water treatment and reclamation) and their location in the Research and Development cycle.

In the early stages of this work, a few characteristics of the groups of tailings technologies became apparent:

- There are just a few technologies in the mining category that can be used to reduce the amount of fines reporting to tailings.
- Similarly, there are only a few opportunities in water-based extraction to influence tailings behaviour in a meaningful way.
- Other, non-water based extraction methods provide an opportunity to avoid creating tailings slurries, if some of the environmental and economic hurdles can be overcome.
- The majority of technologies catalogued by C1 are in the tailings processing and tailings capping/deposition categories, and these technologies were the subject of close examination later in the project.
- Only a few reclamation technologies were identified that could significantly impact tailings behaviour. Most of these are already in commercial use, but there are several in the research / development phase that were identified.
- There are some technologies (mostly chemical amendments) whose main purpose is to enable or enhance the effectiveness of other tailings technologies.
- There are a number of water treatment technologies that can be applied to a variety of tailings situations, mostly to treat various types of tailings process water, but these are not specific to any particular tailings treatment technology.

C1 also summarized current mining and tailings practice in the oil sands industry, and defined eight technology suites. These suites encompass an “end-to-end” set of technologies (from 7 to 10 in each suite) that are being commercially operated and/or commercially demonstrated to produce, treat, dispose of and reclaim tailings. They are listed below (named to indicate the main tailings processing or disposal technology):

- Conventional hydraulic fill construction.
- Water-capped MFT.
- Composite tailings (including both CT and NST).
- Conventional tailings thickening.
- In-line thickening of tailings, with thin lift deposition/dewatering.
- In-line thickening of tailings, with thick lift deposition/accelerated dewatering.
- Centrifuged MFT.
- Coke capping.

In all, there are 22 tailings technologies already in commercial use – many of these (such as conventional beaching or CT) are mature, whereas others are really in an advanced development stage (running at a commercial scale, such as thin-lift dewatering), being implemented in response to recent changes in regulations.

The Technology Evaluation

Evaluation of the technologies identified by C1 was an ongoing process throughout the remaining project, with:

- C2 establishing a set of tailings management and reclamation (TMR) objectives.

- C3 developing criteria to evaluate the technologies for their ability to meet the TMR objectives, and then carrying out that evaluation.
- C4 further identifying technologies with the potential to improve existing commercial technology suites or create new suites and then, based on the C3 evaluation results, assessing the benefits and risks associated with each technology in order to prioritize them for inclusion in the TTD roadmaps.

The TMR objectives were developed by C2 and confirmed with OSTC and government representatives in several workshops. The eight main TMR objectives are listed below; a full list of the objectives and all related sub-objectives is given in a companion paper in this conference ("The Oil Sands Tailings Technology Roadmap Project: The Identification of Key Objectives for the Evaluation of Tailings Technologies").

1. To minimize production and long-term storage of fluid fine tailings.
2. To manage tailings in a manner that minimizes the impacts of process affected water on the environment.
3. To facilitate progressive reclamation and achieve a trafficable surface as soon as possible following the cessation of deposition.
4. To reduce ongoing operations liability and long-term closure liability.
5. To minimize the footprint of permanent tailings facilities.
6. To minimize cost of construction, operations and reclamation without compromising safety.
7. To use robust technologies.
8. Potential to reach commercial implementation.

C3 developed a set of evaluation criteria, tied to the TMR objectives, and a process for evaluating each of the 101 unique technologies. The evaluations were carried out in a number of sessions by groups of oil sands experts, selected for their knowledge in different aspects of the mining life cycle. The evaluation results represented a consensus opinion of each group, and while not "perfect", reflected the best judgement of the evaluators.

The primary purpose of the evaluation was to identify the strengths and weaknesses of each technology (in regards to meeting the TMR objectives), so that C4 could better understand and utilize these technologies in developing the TTD Roadmaps, and appropriately fit each technology into a complete technology suite.

The results of the evaluations are discussed in a companion paper in this conference ("The Oil Sands Tailings Roadmap and Action Plan: Assessment and Evaluation of Tailings Technologies").

The early C4 work focused on identifying technologies from the C1 catalogue that had potential to improve existing commercial suites or contribute to new suites. Some of these technologies were already at a commercial stage (and hence did not need any further roadmap consideration); some were in a research or development stage. Tailings technologies in the latter group (research or development) were compiled into a list of "highlighted" technologies, as shown in Table 1.

The last stage of the evaluation work was to assess the benefits, risks and relative costs of each of the highlighted technologies, which was done to prioritize them for consideration in the TTD roadmaps.

Table 1. "Highlighted" Technologies

Technology Category	Technology #	Description
Mining	T-606	Mobile In-Pit Crusher
Extraction & Bitumen Recovery	T-600	Fine Sizing (Crushing)
	T-602	Secondary (Deep Cone) Separation Vessel
	T-604	High Temperature Heating (for Solvent Removal)
	T-607	Mobile versions of conventional separation/ extraction equipment
	T-024	Alberta Taciuk Process
	T-186	Solvent Extraction
	T-548	Retort Based Bitumen Extraction

Techno-logy Category	Techno-logy #	Description
Deposition and Capping	T-603	Control of Biogenic Gas during MFT Spiking
	T-040	Thin Lift Drying: Robinsky Cones
	T-032	Accelerated Dewatering
	T-037	Thin Layer Freeze-Thaw Cycling Dewatering (T-037)
	T-039	Accelerated Evapotranspiration using Vegetation
	T-062	(Variation) Co-Mixing Overburden and Centrifuge Cake
	T-065	Interlayered Centrifuge Cake & Sand – Variation of Interlayer MFT & Sand
	T-088	Shock Densification
	T-090	Vertical Drains
	T-099	Stacker Hydro-cyclones (including fully mobile versions)
	T-188	Under-drained Tailings
	T-270	Conventional Hydraulic fill (Improvements to…)
	T-438	Subaqueous Capping
	T-510	Tailings Discharge Variations
	T-550	Tailings Surface Sealants
Tailings Process-ing	T-601	Waste Cooling (in Retort Extraction Processes)
	T-609	Mobile Centrifuge
	T-612	Vibrating Screens (for dewatering sand)
	T-018	Hydrodynamic Cavitation
	T-020	FTT – Bitumen, Solvent and Heavy Mineral Removal

Techno-logy Category	Techno-logy #	Description
	T-060	MFT Spiked Whole Tailings
	T-067	Cross Flow Tailings Filtration
	T-069	Solid Bowl Scroll Decanter Centrifuge (including a mobile version)
	T-080	Vacuum Filtration (including a mobile version)
	T-085	Thermal Drying
	T-185	NST Production from Cyclone Underflow, Thickener Underflow and MFT
	T-197	Super CT
	T-206	NST from sand and TT
	T-208	Paste Thickener
	T-267	Froth Treatment Tailings Thickening
	T-529	Oleophillic Processes/Technologies
Reclama-tion	T-138	Water-Capped Lake (over MFT)
	T-605	Water-Capped Lake (over mostly TT)
New Technolo-gies	T-610	High Density MFT Harvesting
	T-611	MFT Tank Thickening
	T-614	Shear Conditioning of Soft Tailings
	T-608	Geotextile Drainage Layer
	T-609	Mobile Centrifuge

Sixteen technologies (two similar) were selected as high priority for inclusion in the TTD roadmaps. These were technologies that had some combination of high benefit, low risk and/or low cost (but not necessarily all three). Some of the technologies have a wide application in a variety of technology suites; others have a much more specific application or are of very limited use.

The high priority technologies are listed below by mining lifecycle category. The technologies are listed in order of technology number; no priority or preference is implied by their arrangement. Technologies with restricted application are marked with an asterisk (*) and those that are of very limited use with a double asterisk (**).

The following are the high priority technologies that might be used as an alternate to current water based extraction methods:

- T-024 / T-548 Alberta Taciuk Process / Retort Based Extraction

The following are the high priority technologies from the tailings processing stage of the mining life cycle:

- T-060: MFT Spiked Whole Tailings*
- T-069: Solid Bowl Scroll Decanter Centrifuge
- T-085: Thermal Drying*
- T-197: Super CT
- T-208: Paste thickener
- T-267: Froth Treatment Tailings Thickening*
- T-529: Oleophillic Sieve

The last technology (T-529 Oleophillic Sieve) is included in this list not for its potential as an extraction technique, but because of its potential for secondary processing of tailings for bitumen removal (and the resulting improvement in subsequent tailings treatment steps). Several other technologies could potentially also be used for this purpose, e.g., Oleophillic beads (also T-019), Hydrodynamic Cavitation (T-018) or FTT – Bitumen, Solvent and Heavy Mineral Removal (T-020). There may also be other as yet unidentified technologies that could be used to remove bitumen from oil sand tailings.

The following are the resulting high priority technologies from the deposition and capping stage of the mining life cycle:

- T-032: Accelerated Dewatering
- T-062: Co-mixing MFT & Overburden*
- T-090: Vertical Drains
- T-099: Stacker Hydro-cyclones
- T-188: Under-Drained Tailings*
- T-510: Tailings Discharge Tremmie

The following are the resulting high priority technologies from the reclamation stage of the mining life cycle:

- T-138: Water Capped MFT Lake
- T-550: Tailings Surface Sealants**

The Technology Deployment Roadmaps

The final result of the Project was the production of nine Tailings Technology Deployment (TTD) roadmaps. Each roadmap focussed on technologies in a research or development state, which could potentially be used to improve an existing commercial technology suite or contribute to the formation of a new technology suite. Each roadmap is thus technology suite-centered.

A list of the nine roadmaps is given in Table 2. The TTD roadmaps are discussed briefly in a companion report in this conference ("The Oil Sands Tailings Technology Roadmap Project: The Identification and Improvement of Tailings Technology Suites, and Pathways for Technology Development"). To fully appreciate the definition and discussion of the TDD roadmaps, the reader should download V5 of the Project Report from one of the aforementioned websites. The major conclusions and recommendations related to the roadmaps are presented in a following section.

MAIN CONCLUSIONS

The main conclusions of the study are listed below (further, more detailed conclusions are given in Volume 1 of the Project Report):

Table 2. Roadmap List

Road-map No.	Related Tailings Technology Suite Name
1	Centrifuging MFT with conveyor/stacking
2	Composite Tailings
3	In-line thickening with accelerated dewatering
4	In-line thickening with thin lift evaporative drying
5	Thickening
6	Water capped end pit lake
7	Improvement to Water-Based Extraction
8	Non-Aqueous Solvent Extraction Retort Based Extraction Parallel High/Low Fines Suite
9	In-Pit Tailings Stream

1. There is still no "silver bullet" tailings technology, i.e., a single technology or suite of technologies, which will solve all the Oil Sands tailings challenges with a single effort.
2. A set of 9 TTD Roadmaps has been developed by the Project (full Roadmaps summarized in Section 4, with full Roadmaps in Appendix E). These Roadmaps consider improvements to existing technology suites and contributions to potentially new suites, and identify technologies with a high priority for Research and Development (R&D) to effect these changes.
   a. Many improvement opportunities have been identified across a number of technology suites, both existing and new (see next item).
   b. A significant number of technologies have been highlighted to address these opportunities, and represent substantial potential for further development in the Oil Sands. Technologies considered applicable to each technology suite are shown on Table 6.1.
   c. The status of development of individual technologies has been plotted on a detailed generic model graphic, to map out the steps required in their future research and development.
3. There is a major opportunity to increase the performance and decrease the cost of existing commercial tailings technologies and suites employed at the oil sands operations. These opportunities are discussed in detail in TTD Roadmaps 1 to 6. Some of the major opportunities are:
   a. Reduce segregation during beaching (for Conventional Tailings, CT and TT). Potential improvement technologies are mostly centered on increasing the density of the tailings stream – by thickening the fines stream (directly or as a technology component; various types of tank or in-line thickeners) or by removing water from the sand stream (hydrocycloning or filtration). For these technologies, chemical amendments and technologies that remove bitumen from the tailings streams have potential as enablers. Other improvement technologies focus on reducing the amount of shear during deposition (e.g., tremie diffuser).
   b. Increase deposit strength and improve trafficability, or alternatively, cap very soft deposits (for Conventional Tailings, CT, TT, Accelerated Dewatering and Centrifuging MFT). Potential improvement technologies focus on removing water through drainage (e.g., vertical drains or horizontal drainage layers), by drying (using thin-lift deposition) or by freeze-thaw; chemical amendments to increase strength; or using soft soil capping technologies (hydraulic or mechanical).
   c. Improve fines capture/storage (for Conventional Tailings and CT). Potential improvement technologies focus on introducing additional fines into the tailings stream, either by fines spiking or by using a more dense fines stream as a technology component (e.g., for CT, making it with highly thickened fines).
   d. Address geotechnical risks (for Conventional Tailings and CT). Potential improvement technologies focus on densifying the material to increase strength or decrease the risk of liquefaction (using technologies that promote drainage or increase density before discharge).

e. Reduce long-term settlement (for TT, Accelerated Dewatering and Centrifuged MFT). Potential improvement technologies focus on improving the initial density of the tailings stream (through improved thickening, which generally relies on better chemical amendments), or speeding up consolidation through enhanced drainage (e.g., Vertical drains).

f. Address long-term environmental impacts (mentioned for Conventional Tailings and CT, but likely applies to most of the commercial technology suites). Potential improvement technologies focus on controlling release water chemistry (which might be addressed by appropriate selection of chemical amendment technology or by water treatment), or recovering bitumen and/or heavy metals from the tailings streams.

g. For technologies that rely partially on drying to remove water from tailings streams (In-line flocculation with TLD, Accelerated Dewatering, and Centrifuging of MFT), there are specific improvements that are needed, such as reducing drying area requirements, improving consistency of the processed tailings stream, and overcoming rheology effects (too steep or too short of tailings runout). Not all of these issues are applicable to all of the technology suites mentioned, but potential improvement technologies focus on improved thickening (usually, with better chemical amendments), pre-treating the initial tailings stream to remove bitumen (e.g., using oleophillic sieve or similar technology), modifying discharge methods (e.g., use of central-point discharge), or enhancing the effects of drying with freeze-thaw techniques.

h. Separate fines rich from fines poor ores and then treat the two ore streams separately. Potential improvement technologies include selective mining and the use of non-water based extraction technologies.

i. If tailings streams are separated for whatever reason, (e.g. using a cyclone), keep them separated for processing and deposition. For example, this is already done in producing TT, but there may be other improvement opportunities of this nature.

j. The water-capped MFT technology suite has its own specific improvement opportunities, many of which are related to achieving a higher density of the water-capped MFT or capping alternate, higher density fine tailings. Potential improvement technologies include centrifuging or thickening (in-line or tank) of the MFT, or high density MFT harvesting from existing ponds.

4. The typical time frame required to progress a technology through the Research stage is from 2 to 3 years, and through the Development stage from 2 to 5 years. Additional time may be required after initial commercial implementation to bring the technology to a mature state – in the order of from 1 to 5+ years. Timelines for bringing potential tailings technologies to full commercial implementation are thus quite long (from 5 to more than 10 years); this fact must be recognized when planning tailings management improvements (to reach desired reclamation and closure objectives, or in responding to regulatory changes).

MAIN RECOMMENDATIONS

The work of all parties associated with the Project demonstrates the very important benefits of full cooperation and communication amongst operators, and between operators, regulators and consultants. The efforts to date are to be lauded, but this is not the time to relax focus on solving the very real oil sands tailings issues. Rather it is a time when this past success should be leveraged, and for this reason, the following major recommendations are offered:

1. As a follow-up to the TTD Roadmaps included herein, the following next steps should be taken:
   a. The Roadmaps should be assessed by individual operators for applicability to their specific site(s), mining practices, and tailings inventories, so that company priorities for R&D can be set.
   b. More detailed R&D plans should be generated from the selected Roadmaps and specific R&D technologies. There is a role for both operators and regulators to play in this process (further discussion below).

2. In regards to the previous point, the TTD Roadmaps and related information should be regularly revised, so that they do not become dated. This should include:
   a. Setting up a process to receive additional tailings technology as it becomes available.
   b. Soliciting information from credible and experienced vendors who have so far not become involved in oil sands tailings.
   c. Lowering the threshold for potential vendors to participate in the Roadmap process. This should include defining what "entrance" information is needed and standards for data submitted to the government and operators (including such items as protocols for sampling and laboratory testing).
   d. As the technology database is updated, the technology evaluations and TTD Roadmaps should also be updated to reflect the new information.
3. The Roadmaps given in this report have focused largely on Tailings Processing and Tailings Deposition & Capping technologies. More focused study and attention should be given to potential technologies from the other mining life cycle categories (Mining, Extraction and Reclamation) that could contribute to better tailings management. Some examples include (but are not limited to) the following:
   a. Water based bitumen extraction methods are mature technologies and are key to current oil sands economics. They are unlikely to be replaced without significant development of new methods, with proven economics and commensurate (step change) benefits to tailings, but only if environmental and economic factors can be addressed. This is a major research initiative.
   b. A "position paper" that defines the current state of Reclamation technologies, and casts a wide net in searching for "new" technologies that have not been used in the oil sands industry to date, should be prepared.
4. The full potential of chemical amendments as technology enhancers has only begun to be addressed. Greater R&D focus should be brought to bear on this issue, particularly by leveraging the knowledge of vendors who are active in industries outside of oil sands, but which has not been tapped to date.
   a. Recommendations 2b) and 2c), regarding lowering the entrance threshold for companies who want to contribute to solving tailings problems, are particularly germane to this point. If the industry expectation is that proponents take a particular tailings technology through steps 1 to 3 (or 4) of the R&D model prior to approaching oil sand operators, then the types of tests and standards for carrying them out should be clearly defined.
   b. The OSTC should consider establishing or maintaining large lab and pilot scale facilities, which could be used by vendors or other third party proponents to demonstrate the effectiveness of their chemical amendment(s), after they have passed their "entrance exam".
   c. The development of partnerships between Oil Sands industry operators and tailings technology providers should be further encouraged by establishing collaborative investments and testing programs, which would realize accelerated benefits from new technologies.
5. A more flexible regulatory approach that places more emphasis on achieving TMR Objectives, and less emphasis on short-term goals, would result in more effective and less costly tailings reclamation efforts. Government regulators should give serious consideration to updating current regulations. Several examples are as follows:
   a. Current regulations are focused on the shear strength of the soft tailings only, whereas trafficable surfaces require compatibility between the cap and the underlying soft tailings. Current plans and regulations for stabilization, capping, and reclamation of soft tailings deposits should be re-examined for compatibility with material properties (both capping materials and underlying tailings), the scale of the operation, costs, and the desired landscape performance. Various methods for hydraulic capping should be included in this evaluation.
   b. In a similar vein, current regulations assume particular capping and reclamation methods but exclude others due to restrictive criteria. Consideration should be given to technologies that are able to cap very soft deposits and promote early consolidation and strength gain, as opposed to the more traditional

approaches that require access by mechanical equipment.

6. It became clear when carrying out the Project work that issues of water balance and water chemistry are becoming critical and need attention. Initial efforts at defining water treatment needs were addressed in the Project, but this issue is larger than just water treatment. A similar focus should be directed, in the future, at addressing water storage and water release issues as was given to tailings management issues herein, with a similar definition of objectives for surface and ground water, and means of achieving those objectives.

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OIL SANDS MINE CLOSURE – THE END GAME: AN UPDATE

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PREAMBLE

The first version of this presentation (Morgenstern, 2012)) was presented at the David C. Sego Symposium held at the University of Alberta on April 26-27, 2012. It was received with considerable interest and I was encouraged to present it to a larger audience. This update integrates some comments that I have received and expands on the central theme.

OIL SANDS MINING – THE END GAME

WIKIPEDIA defines End Game as the ending scenario of a particular game; when and how it will end, most prominently used in chess. In Oil Sands Mining (OSM) the ending scenario is Closure leading to Certification whereby custodial transfer returns the lease to the Crown with, for all practical purposes, the prior Operator being relieved of on-going obligations and responsibilities. This presentation will summarize my sense of how this End Game evolved and the difficulties to be faced in applying it in future practice. It will raise the question whether alternate End Games are more practical and desirable. The purpose of the presentation is to expand the discussion around the question: What is the optimal End Game for OSM?

THE CHALLENGE OF CLOSURE

NASA supports a wonderful web site that presents sequential imagery of the oil sands mining leases from 1984 to 2011, inclusive.

(http://earthobservatory.nasa.gov/features/worldofchange/athabasca.php)

As of September 2011, roughly 602 square kilometres of land had been disturbed for oil sands mining. The distribution and holding of available mining leases has recently been consolidated among corporations so that the type of industrial stakeholders going forward is now clearer. A recent estimate of the surface mineable area is 4,750 square kilometres and 99% is under lease (see Figure 1). The ultimate landscape, based on experience to date, will be dominated by mine waste and water management structures with a significant but smaller area, devoted to plant sites of various kinds. The mineable area straddles the Athabasca River.

A casual assessment of the dominant landforms reveals the widespread presence of process-affected water, either on the surface or in the groundwater. Its conjunction with both major and minor waterways raises issues rarely encountered in mine closure.

Bringing this landscape to Closure, consistent with regulatory requirements, is a unique challenge. Put simply, we, the multiple stakeholders, are engaged in the largest project of ecosystem reconstruction that, arguably, has ever been undertaken. It therefore behooves us to make every effort to ensure that the enterprise is well-directed for success. While to some, the scale and complexity of Closure of the lands disturbed by oil sands might be excessively daunting, I regard it as a precious opportunity to contribute to enhancing the sustainable development of the oil sands area by creating the most-effective land management system that is consistent with our policies. This discussion of alternate End Games is intended to be a contribution in that direction.

EXPERIENCE WITH CERTIFICATION

The current End Game is custodial transfer of disturbed lands back to the Crown, following certification. Once reclamation to appropriate standards is complete, monitoring activities begin and it can take 15 or more years to effectively establish an acceptable ecosystem. Reclamation certificates are only issued when long-term monitoring demonstrates that the reclaimed land meets the objectives of equivalent land capability. The Regulator has now established a classification of disturbance,

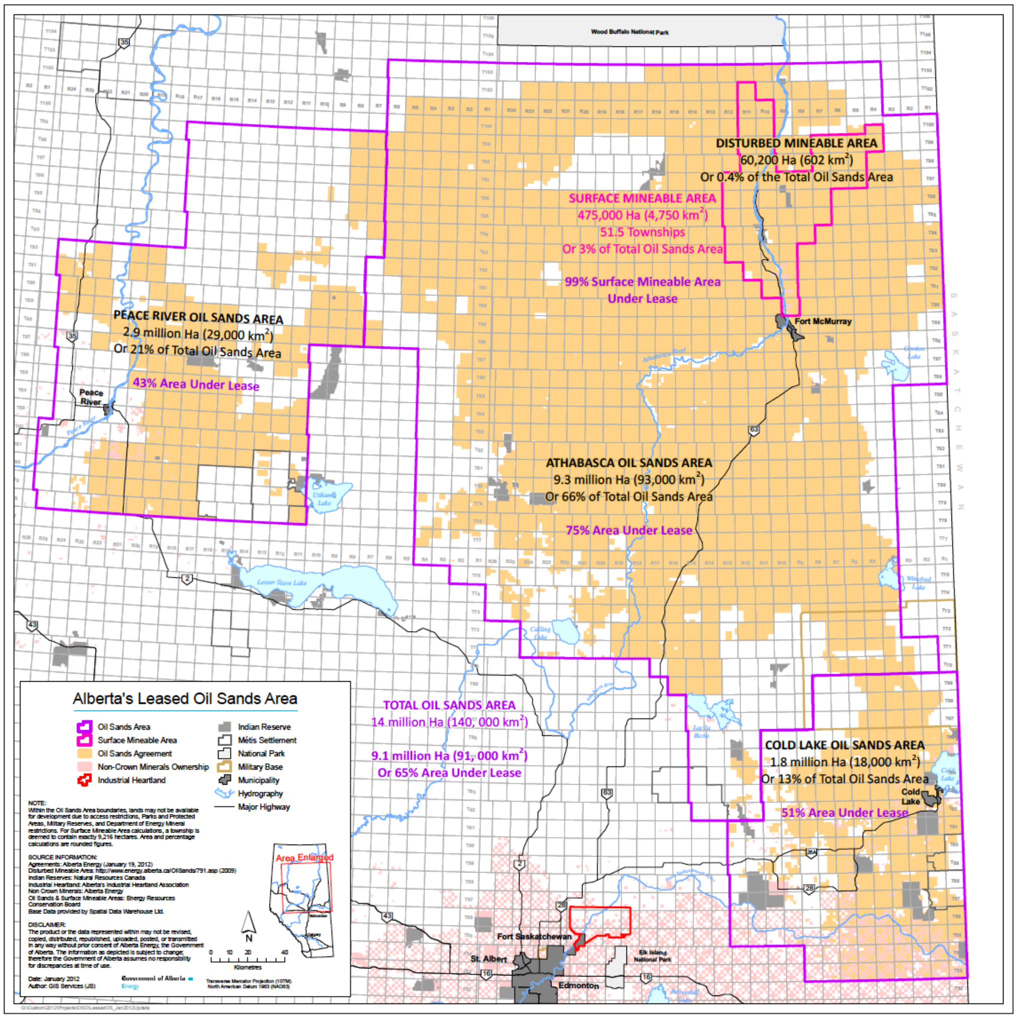


Figure 1. Alberta's Leased Oil Sands Area (from Alberta Energy, Alberta Leased Area).

ranging from Cleared to Certified. This provides for accounting of the efforts devoted to intermediate stages of reclamation. As shown in Figure 2, as of December 31, 2010, 71,497 ha had been disturbed with 104 ha certified. Since that time the disturbed area has increased, but with no increase in certified areas. To my knowledge, there are no plans in the industry to apply for additional certification in the near future.

Figure 3 (Macyk and Drozdowski, 2008) summarizes the major regulatory changes and documents to guide reclamation practice and mechanisms for measurement of reclamation success. It seems most likely that the early lease approval process drew upon the then existing practice for approval of coal mining. Within the time period of approval of commercial development of early oil sand leases, (Suncor and Syncrude) experience had already been gained in reclamation and Certification within the coal industry. At the time coal mine operations in support of thermal power production west of Edmonton were disturbing, reclaiming and certifying about 100 hectares a year. The certification process took about one year for each block and, at least for awhile, became relatively routine. It was not unreasonable to assume that this process could be extended to the oil sand industry.

At the time of the initial permits, forest management for commercial purposes was a high priority within the Regulatory community and the achievement of "equivalent land capability" was often interpreted to mean just the return of an area capable of supporting commercial forest equal to the original area. As the industry matured, it became increasingly less clear as to what "equivalent land capability" means. The ambiguity is discussed in some detail in (Jones and Forest, 2010).

The sole Certified area is called Gateway Hill. This landform began life as an out-of-pit overburden dump early in Syncrude's operations. The area was first planted in the early 1980's and Certified in 2000, approximately 30 years after its initial formation. As someone involved in advising Syncrude on reclamation-related matters at that time, I remember supporting the recommendation to apply for certification in order to gain experience with the process.

The experience was disappointing. Gateway Hills is the simplest landform conceivable in the industry, yet certification took about ten years from the decision to certify and receiving certification. The incremental costs required to meet the approval standards were surprisingly high and the requirements being imposed appeared to lack transparency. It was my impression that the Regulatory system was not prepared to engage in Certification of oil sand landforms in the same manner that it had engaged in the coal industry. However all participants in this certification exercise benefitted from the experience, as did the industry as a whole.

Reclamation is primarily regulated under the Environmental Protection and Enhancement Act (EPEA) originally administered by Alberta Environment and now administered by Alberta Sustainable Resource Development (ASRD). Since 2007, EPEA approvals for oil sand mines include the following conditions, amongst others:

- "The approval holder shall reclaim the land so that the reclaimed soils and landforms are capable of supporting a self-sustaining, locally common boreal forest, regardless of the end land use."
- "The approval holder shall revegetate the disturbed land to target the establishment of a self-sustaining locally common boreal forest, integrated with the surrounding area….".

This welcome broader perspective embraced by EPEA approvals places greater emphasis on integration at lease boundaries, on the return of regional ecosites, improved soil salvaging and on wetlands reclamation. To facilitate achieving these objectives a number of technical guidelines have been produced, and reporting requirements have been amplified. A valuable summary of these developments is provided by Richens and Purdy (2011) while Houlihan and Hale (2011) have published Figure 4, that summarizes the perception of the regulatory process and associated time lines that lead to Certification.

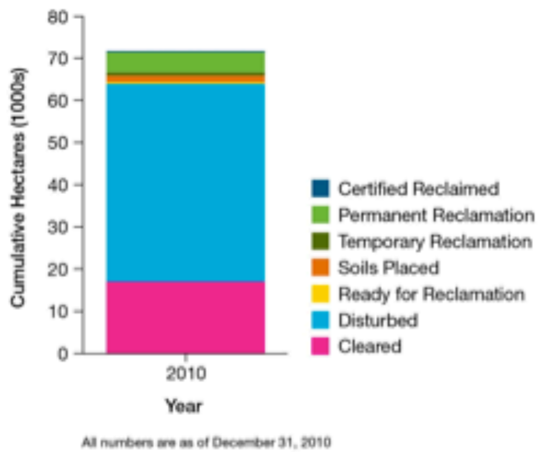
With respect to the Certification process itself, Richens and Purdy (2011) reveal that starting in 2009 Regulators worked with a consultant to develop a comprehensive and updated "Administrative Guide to the Oil Sands Reclamation Process". This evidently incorporates learning from the past experience and provides, among other things, details on the reclamation standards, the review process, as well as an appeal process. This document will be an important contribution to improving the success of the Certification End Game, but at the time of writing it has not been finalized and published. Poscente and Charette (2011) provide an indication of the complex structure required to reduce the Certification process to a set of indicators.

While the broader vision of ecosystem reconstruction embraced by more recent development approvals is welcome, as are the research efforts culminating in technical guidelines to facilitate improved practice, the resulting process leading to certification is untested and encumbered by considerable uncertainty. This is reflected in the report on the Oil Sands Mining Reclamation Challenge Dialogue sponsored by the Oil Sands Research and Information Network (OSRIN) and published by Jones and Forrest (2010). Figure 5, from this report, captures the "oil sands reclamation system" that was the theme of the Challenge Dialogue, while Figure 6 captures a process-oriented view of the path to Certification. Significant gaps and uncertainties exist even within this simplified vision. Moreover within the real system, dominating issues like the ultimate fate of process-affected water, remains virtually unattended.

The challenges associated with Certification are amplified in another Report issued by

The Reclamation Process

In 2009, new definitions were introduced to better track the level of land disturbance and reclamation progress to date.



Certified Reclaimed - 104 hectares If an area meets stringent requirements for reclamation, regulators will issue final certification and the land is returned to the Crown as public land. To date, one area called Gateway Hill is certified reclaimed.

Permanent Reclaimed - 4,835 hectares (3,643 hectares terrestrial; 1,192 hectares aquatic and wetlands) Landform design, soil placement, and revegetation are complete (for both land and aquatic ecosystems). Companies must use local plant species to target the return of local boreal forest ecosystems. Soils are tested and tree and shrub growth is monitored for 15+ years. When ecological trends are achieved, the company can apply for reclamation certification.

Temporary Reclaimed - 780 hectares Some areas are reclaimed and revegetated to grasses for the purposes of stabilization and erosion control. These areas may also see future distrubance.

Soils Placed - 1,534 hectares Soils have been placed as directed by each facility's reclamation and soil placement plans, as approved by regulators.

Ready for Reclamation - 394 hectares Areas that are no longer required for mine or plant purposes and are therefore available for eclamation. Reclamation activites have not begun.

Disturbed - 46,859 hectares Land is still part of the active operations of a facility.

Cleared - 17,055 hectares Land is cleared of vegetation, but the soil is relatively undisturbed. In forested areas, the trees are harvested and some of the smaller wood may be conserved for use in reclamation. Total active footprint as of December 31, 2010 for all oil sands mining activities, including land cleared, disturbed and reclaimed was 71,497 hectares

Figure 2. The Reclamation Process (Alberta Government Oil Sands webite). (http://oilsads.alberta.ca/reclamation.html)

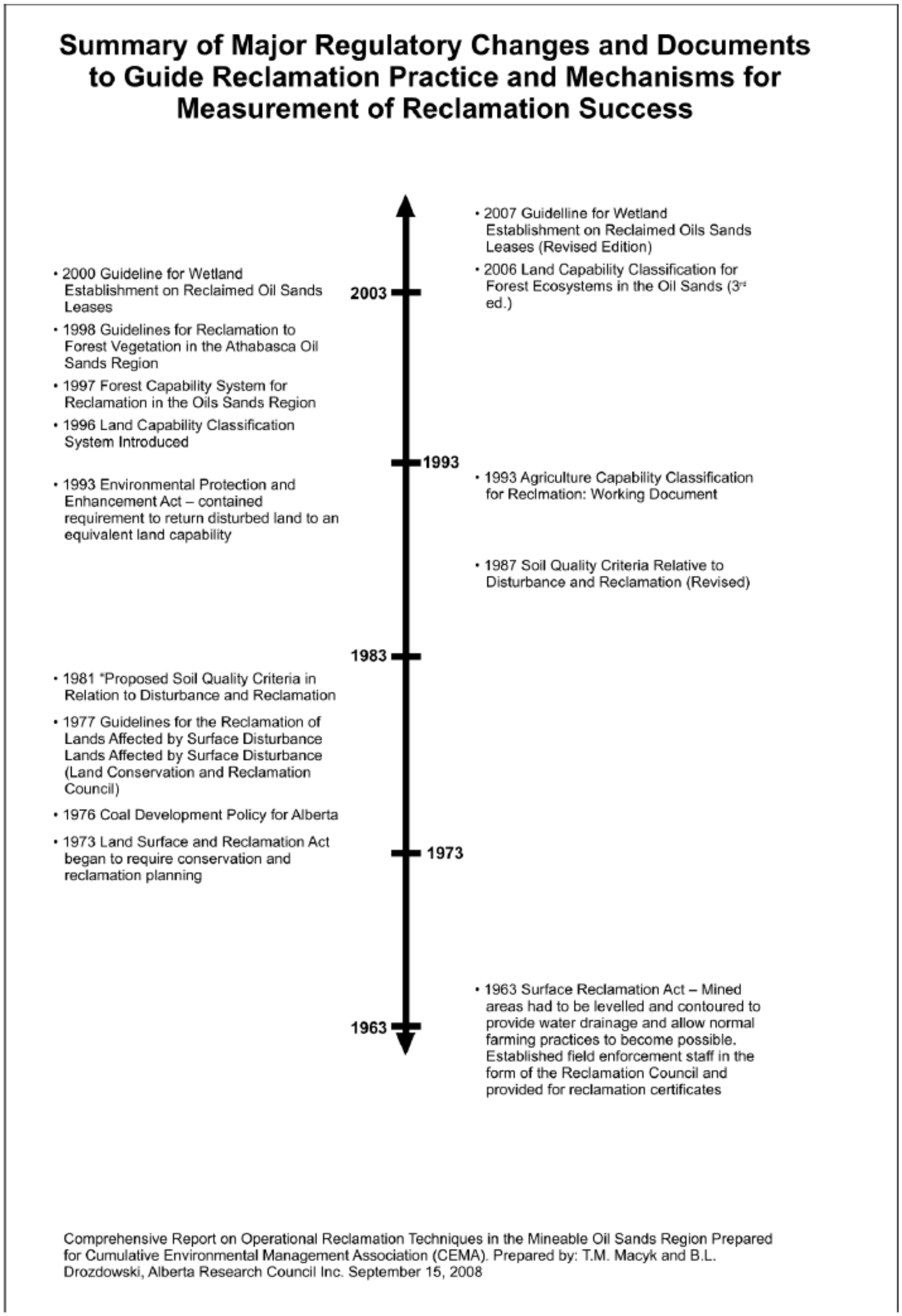


Figure 3. Summary of Major Regulatory Changes and Documents to Guide Reclamation Practice and Mechanisms for Measurement of Reclamation Success. (R.K. Jones and D. Forrest, October 2010).

OSRIN arising from a Workshop on the Information that Professionals Would Look for in Mineable Sand Reclamation Certification (Creasey, R. 2012). This Workshop involved the participation of 50 technical specialists from a variety of disciplines representing about 850 years of experience. Some of the conclusions arising from the Workshop are disconcerting as quoted below:

"The workshop also sought to determine how the confidence in decision making is affected by the use of field equipment/tools and the value of background data and reports in increasing confidence. Given the extensive experience of the workshop participants, it was surprising to see how little confidence they had in using only their knowledge and experience to make reclamation certification decisions, although this likely reflects:

- The increasing complexity of oil sands reclamation
- The difficulty in describing what success is
- The diversity of reclamation substrates, especially the uncertainty surrounding tailings (both in terrestrial and pit lake settings)
- The wide range of regulator and stakeholder expectations
- Concerns that judgment will vary too much among people and over time to be a reliable method for such an important decision
- The liability that is attached to the decision."

It is my view that if the reclaimed landscape consisted primarily of landforms like Gateway Hill, the updated End Game would have a chance of success. However, the reclaimed landscape is dominated by surface and subsurface water considerations, and given the current limited progress in dealing with these issues, it is my view that the updated End Game will also not be fit for purpose. If this view is widely shared, it raises a serious concern for all stakeholders.

IS CERTIFICATION POSSIBLE?

Butler and Bentel (2011) have made a valuable contribution in their recent paper on mine relinquishment (Certification). They write as follows:

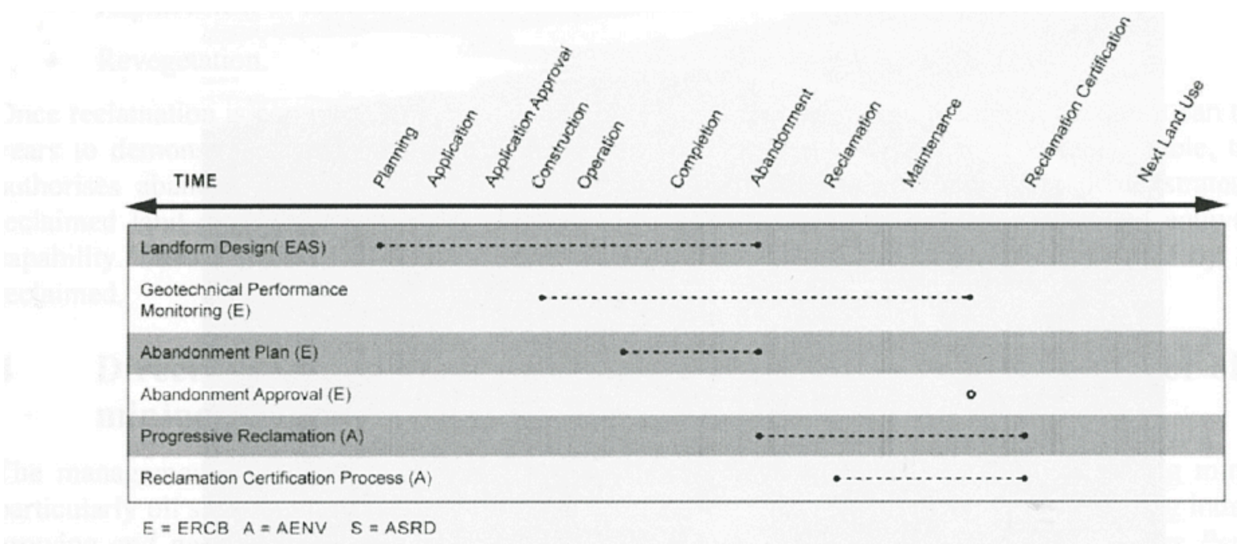


Figure 4. Regulatory process and timelines for oil sands and coal mines (Houlihan & Hale, Mine Closure 2011).

"Mine relinquishment in general terms is the completion of the reclamation activities, meeting agreed closure objectives and return of the site to the landowner. Relinquishment planning and completion is a concept that governments and communities speak to as an expectation and that the mining industry aims to achieve. But there are few standards or guidelines that provide the how to, and only a precious few mine sites have attained all or even partial release. As regulatory requirements for mine reclamation activities and reporting evolve in detail and scope, and bonding becomes the norm, reaching the end point remains elusive for many sites. Instead the current end point for most is care and maintenance. Is relinquishment possible, or practicable, or is it becoming less probable?"

The paper offers some guidance on improving the potential for relinquishment. The guidance is of value whether or not it results in certification.

Based on my general experience with the mining industry beyond the oil sands the likelihood of achieving Certification resulting in custodial transfer is remote. Ultimately, closure is all about water. In the metals industry, all the acid generators, metals leaching, suspended solids etc. issues are related to water. I know of no recent mines, other than in dry climates, that are proposing custodial transfer solutions and many that made enormous efforts in this regard but were ultimately disappointed. Following a visit to 100 mine sites McKenna (2002) found that the only mines not likely to need perpetual maintenance were very small and had very good chemistry.

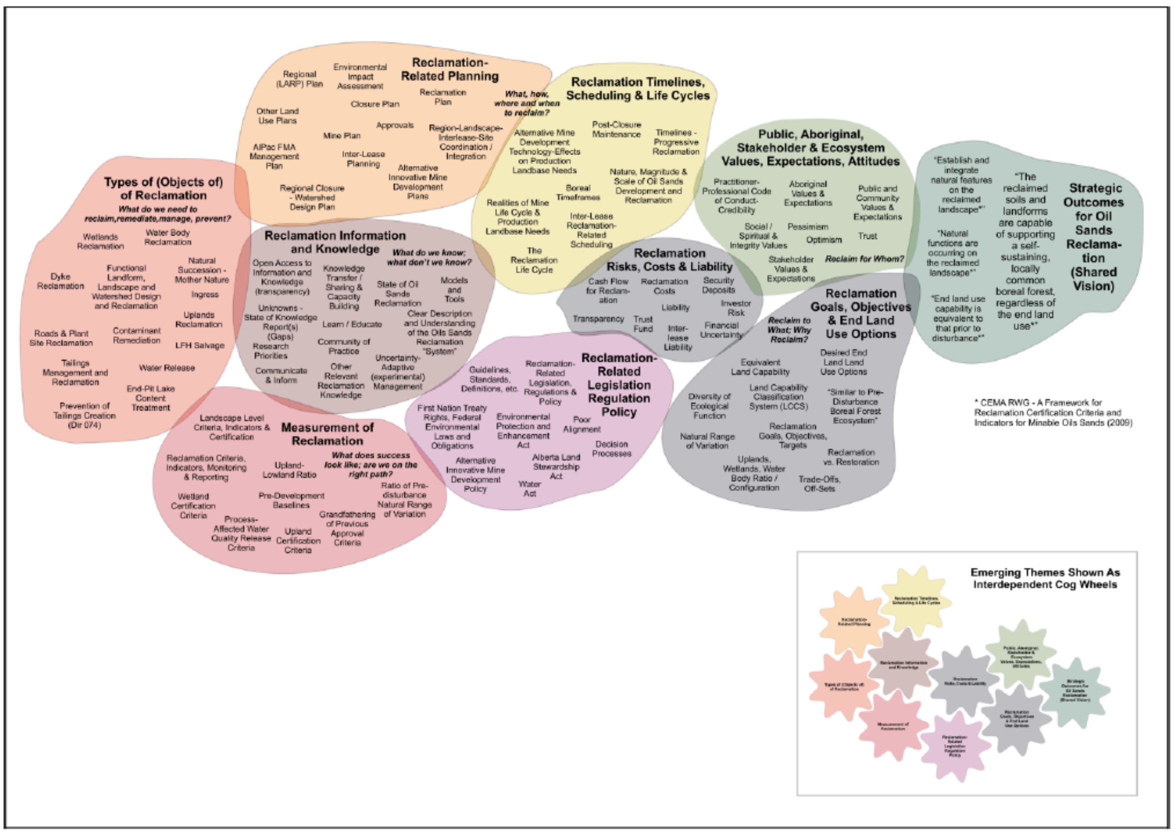


Figure 5. Emerging discussion themes and key discussion points – the "oil sands reclamation system". The inset shows the themes as interdependent cog wheels. (R.K. Jones and D. Forrest, October 2010).

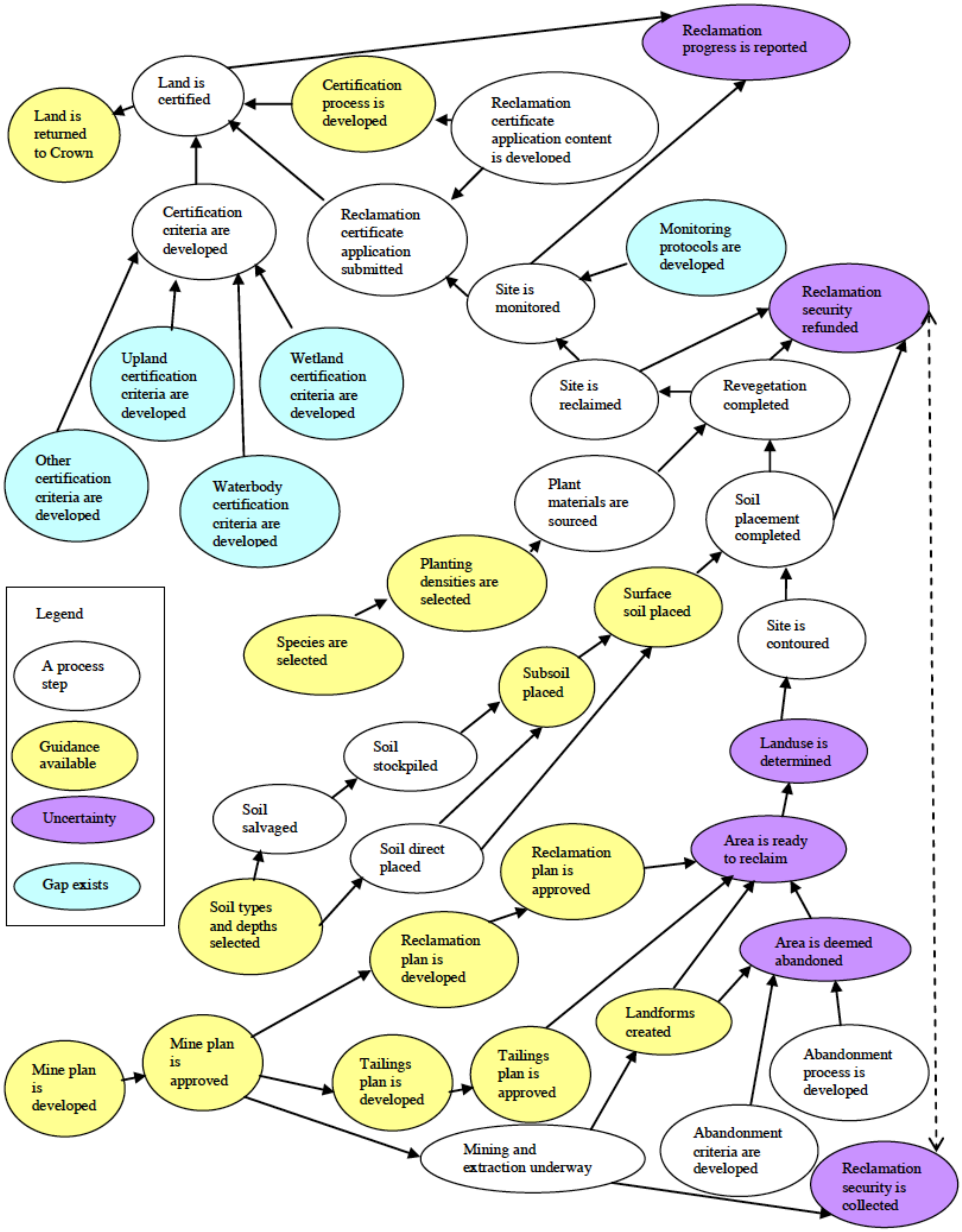


Figure 6. **Simplified process-oriented view of the "reclamation system". (R.K. Jones and D. Forrest, October 2010).**

Kennecott Minerals Co. Ridgeway Mine is an interesting example. It is a gold mine that completed operations in November, 1999. The reclamation challenge involved the tailings impoundment, two pit lakes and wetland systems. The corporation was a model in engaging the local communities to evaluate options which involved conventional reclamation, creation of wetlands, a pit lake development plan of about 10-14 years, followed by a 30 year post reclamation monitoring period that begins when pit lakes fill to ultimate level and the establishment of an Environmental, Education and Research Centre. Even after this lengthy period all with these and other community-centered contributions, it is not evident that walk-away relinquishment will be achieved. This is also illustrated by Kennecott's experience at Flambeau.

In OSM, much emphasis has been on terrestrial reclamation, but it should be recognized that a number of other challenges inhibit reaching Certification. The following illustrate these challenges:

i) Some tailings structures (e.g., South Tailings Pond at Suncor) are being constructed on sand channels. They are being managed extremely well to avoid process-affected-water discharging off lease boundaries. (e.g., Holden et al, 2001). The operator has agreed to manage the seepage until it meets discharge criteria. The time required for this is almost indeterminate, but in any case will be decades.

ii) It is known that pore water associated with the construction of Tar Island Dyke is seeping into the Athabasca River. No impacts to the aquatic ecosystem have been found. However, it could take about 100 years for this seepage to end. Does this preclude Certification until the draindown is over?

iii) There is currently a lack of public trust in the assessment of the impact of the oil industry on the water quality in the Athabasca River, notwithstanding a positive assessment by the Royal Society Study (Royal Society of Canada, December 2010). This arises from both technical and non-technical reasons. Regardless of how this trust was lost, it precludes the likelihood of any early resolution of discharge criteria for process-affected water to the river.

iv) Water treatment is entering the business case for several operators. It is easy to imagine that once these plants start-up, they will be incorporated as essential in the very long term to enhance water quality stored on sites.

v) Many reclaimed tailings ponds will involve long term settlement that will require adjustment to drainage features. Does this preclude Certification until the process has ended?

vi) About 30 end-pit-Lakes are proposed in the OSM industry. CEMA (2012) have produced an excellent technical guidance document on this important landform that anticipates a timeline, replete with adaptive management, of about 100 years from lease development to certification. Even this recognizes the need for more complex governance than simple Certification as experienced so far.

vii) There is no denying the importance of wetland reclamation in OSM ecosystem reconstruction. Foote (2012) makes a compelling case that it is not possible to satisfactorily predict post mining reclamation of wet lands in advance which emphasizes the limitations of the End Game based upon certification.

THE IDEAL END GAME

Prior to recommending alternate End Game strategies, it is of interest to list the characteristics of an ideal strategy. The following come to mind:

1) It must honour government policy
2) It should expand stakeholder participation. For example, Fort McKay have continually expressed their desire to be involved in the Certification process (Buffalo, K., et al, 2011).
3) It should provide world-class scientific and managerial leadership in large-scale ecosystem reconstruction.
4) It should support industrial opportunities arising from its mandate.
5) It should recognize its obligation to maintain archives of the reconstructed landforms and other relevant data.
6) It should recognize the impact of changing societal expectations with time.

7) It should provide confidence in Operators that the temporal limit to reclamation efforts and related financial obligations can be determined in a well-defined manner.
8) It should strive to ensure that reclamation related expenditures are optimally cost-effective.
9) It should embrace response to extreme events (flood, rain, fire) in its vision.

ALTERNATE END GAME STRATEGIES

It is my view that an alternate End Game strategy to be preferred over the current Certification process is one based on perpetual care. Fortunately, the Native Orphaned and Abandoned Mines Initiatives (NOAMI) (Holmes and Stewart, 2011) has recently supported the production of an excellent resource document on "Policy Framework in Canada for Mine Closure and Management of Long Term Liabilities" (Cowan Minerals Ltd., 2010). It is essential reading for anyone wanting an up-to-date assessment on these issues, as experienced by both national and foreign jurisdictions.

In assessing the role of perpetual care, long-term monitoring and maintenance, the Report states the following:

> "We know, however, that there are elements of sites closed out under acceptable technical standards and guidelines that now require long-term monitoring and maintenance in order to ensure the safeguards remain intact and are performing as intended in the closure process. These safeguards can range significantly depending on the complexity of the original mining operations but generally deal with items such as:
>
> - pit wall stability for open pit mines
> - shaft cap stability
> - tailings and tailings dam stability
> - waste rock/cover stability
> - tailings cover integrity in relation to design for acid generation
> - continued water quality within accepted standards (which may include treatment)
> - crown pillar stability
> - protection against vandalism
> - aesthetics

The following issues are identified:

1) Many current mining operations throughout the world are subject to closure or rehabilitation plans to ensure sites are restored by site owners, to pre-existing or other reasonable land use when mining operations cease. However there are elements of closed sites with currently acceptable rehabilitation practices in place that will require some form of monitoring and maintenance to ensure physical and chemical integrity of the site.
2) Providing sufficient funding to address these long-term needs is in its infancy. Estimates of these costs vary across jurisdictions and occur prior to relinquishment of lands to responsible authorities.
3) Providing for sufficient funding for unforeseen incidents.
4) Stakeholders need to be assured that funding and the appropriate application of funds is sufficient, directed to site needs, and physical/chemical issues dealt within an effective and efficient manner.
5) Site information is complete, securely stored, maintained and easily accessible for use.

Policy Guidance included in the Report is as follows:

- Ensure all closed out site features that may present a future hazard and cost are identified in all the closure plan process.
- Develop a site land return process that focuses on these features/hazards to provide a degree of certainty of impacts, potential for occurrence, level of risk acceptance and method of costing. This should include worst case scenarios to assist in emergency response planning and costing.
- Establish or identify a jurisdictional body that coordinates agency/stakeholder inputs and has authority to negotiate final assurance requirements and develop appropriate inspection programs.
- Establish a recognized authority for receipt of assurance and tracking and consistent application of funds for monitoring, maintenance and emergency requirements. This should include funds dedicated to site specific features as well as funds established for unforeseen incidents.
- Ensure funds are held in dedicated accounts with appropriate investment growth potential.

- Establish a secure archiving/filing system to store mine site data for ready access.
- Ensure all land use restrictions are applied, recorded, enforced and appropriately identified in all land use planning systems such as GIS."

It is evident that an End Game based on perpetual care can go some distance to meet the ideal outlined above. The Cowan report provides details on financial estimating to support perpetual care and hence constitutes a starting point for these considerations.

It is of interest to know that the Province of Saskatchewan has already taken action in this regard by the establishment of its Institutional Control Program (ICP) Saskatchewan MERP(2011). The elements of the ICP are as follows (Butler and Bentel, 2011):

> "Institutional Control consists of these actions, mechanisms and arrangements implemented in order to maintain control or knowledge of a remediated site after project closure and custodial transfer to some form of responsible authority. The two primary components of the ICP are the Institutional Control Registry and the Institutional Control Funds: the Monitoring and Maintenance fund and the Unforeseen Events Fund. The Registry maintains formal records of closed sites, manages the funding and performs any required monitoring and maintenance work. Registry records include the location and former operator, site description and historical records of activities, site maintenance, monitoring and inspection documentation and future allowable land use for the site. The Monitoring and Maintenance Fund pays for long term monitoring and maintenance; the Unforeseen Events Fund will pay for unforeseen future events. Examples of these may include damage resulting from floods, tornadoes or earthquakes. To address the province's risk of accepting sites into custodial responsibility and the costs of future monitoring and maintenance and unforeseen future events, dedicated site specific funding is established by the site holder responsible for an individual site. The funds would be managed by the province but are legislated and independent from the provincial revenue (Saskatchewan Ministry of Energy and Resources, 2009)."

Butler and Bentel (2011) also cite the example of the Contact Lake mine, which is a gold mine that closed in 1998, and which achieved custodial transfer in 2009 with the assistance of the Institutional Control Registry. The details of the process where-by this mine achieved custodial transfer merit study.

NEWS RELEASE ON JUNE 1, 2015

The Government of Alberta announces the formation of the Wood Buffalo Land Management Corporation (WBLMC). The WBLMC is a crown corporation that assumes responsibility for managing the surface rights of lands disturbed by oil sands mining in a manner compliant with Government reclamation policy. It is governed by an independent Board of Directors and funded by agreements made with Mine Operating Companies to assure long term care and maintenance of disturbed lands. A joint agreement with several of the First Nations group has been signed to ensure participation in the WBLMC.

The WBLMC has entered into a joint agreement with EPCOR-NORTH to administer a distributed water treatment network and engage in research and development related to more cost-effective regional water management.

It has also entered into a joint agreement with the University of Alberta and Keyano College to initiate a training and research centre on Ecosystem Reconstruction. This will alleviate the shortage of skilled eco-system engineers required by the WBLMC and operating companies.

A number of Mine Operating Companies have agreed to develop their closure plans in a more integrated manner based on watershed disturbance. This welcome approach arises from the technical directions proposed by the WBLMC.

The head office of the WBLMC is in Fort McMurray with subsidiary offices in Edmonton and Calgary.

CONCLUSIONS

The current policies of custodial transfer by Certification are untested on the complex issues associated with the reclamation of lands disturbed by OSM. Alternate strategies that could implement government policy merit consideration. Strategies based on perpetual care are attractive in this regard and should be assessed.

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Session 1

Tailings Deposition 1

FINES CAPTURE IN A LONG TAILINGS BEACH AT THE SHELL MUSKEG RIVER MINE EXTERNAL TAILINGS FACILITY: HYDRAULIC AND DEPOSITIONAL ASPECTS

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ABSTRACT

At Shell Albian Sands oilsands operations, bitumen extraction produces three tailings streams, namely coarse sand tailings, thickened tailings, and tailings from the solvent recovery unit. The three streams differ in flow rate, concentration, and particle size distribution of the solids fraction. Since 2004, these three tailings streams have been continuously discharged in the same area, the north-east corner of the External Tailings Facility where they have formed a long beach. The average fines content observed in this deposit is around 25% using 44 micron as the fines cut off. It may have the highest fines capture for an un-engineered tailings deposit in the oil sands industry. It is important to understand the depositional and hydraulic aspects associated with sub-aerial tailings placement on this beach and provide insights into the possibility of maximizing fines captured in beaches by conventional tailings operation techniques. Several years of sample profiles and testing are provided, along with general observations and potential implications for the oilsands industry.

DEFINITIONS

In this paper, the following definitions are used:

- Beach is a solid tailings deposit;
- Fines are defined as the mineral particles smaller than 44 microns;
- Fine tailings are tailings deposits or slurries having a maximum diameter equal to or smaller than 44 microns;
- Coarse tailings are tailings deposits or slurries having a minimum diameter larger than 44 microns;
- Clay-size tailings or clays are the mineral particles smaller than 2 micron;
- Bitumen is the hydrocarbon measured from Dean Stark extraction;
- Particle Size Distribution (PSD) is determined by means of Laser Diffraction (LD) according to the Shell Canada tailings investigation standard working procedures (2010);
- Fines content is the ratio between mass of fines and mass of mineral solid in a tailings deposit;
- Fines capture is the ratio of mass of fines in a beach to the mass of fines discharged by a tailings line; and
- Bitumen content is the ratio between mass of bitumen and the total mass of a tailings deposit.

INTRODUCTION

The Athabasca Oil Sands Project (AOSP) is a joint venture among Shell Canada Energy (60%), Chevron Canada Limited (20%) and Marathon Oil Sands L.P. (20%). The AOSP came on stream in 2003 with a design capacity of 155,000 barrels per day (b/d). Tailings, the residual by-product that remains after the bitumen is separated from the mined oil sands ore, are an important matter for the oil sands mining industry. Tailings are composed of residual bitumen and solvents, water, sand, silt and clay particles. Upon deposition into the tailings pond, coarser fraction settles close to the discharge point and forms a solid deposit (beach), while the finer part of the tailings remains suspended in the run-off and accumulates in tailings ponds. In the Western Canadian oil sands industry, it is common practice to assume that 50% of the total fines in the ore feed is captured in beaches and dykes, whereas the remaining 50% accumulates in fluid ponds as fluid fine tailings. The 50% of the total fines captured in beaches and dykes is generally in deposits having fines content not exceeding 15%. This assumption together with the time required for the suspended fine tailings to settle from suspension has important implications for tailings space planning.

Since start of operations in 2003, Shell Canada has conducted annual tailings investigations to monitor the volume and the properties of the fluid tailings at the External Tailings Facility (ETF) and to assess the geotechnical stability of the beaches. From 2009, the annual tailings investigation scope has included additional sampling and testing techniques for the tailings beaches to increase the understanding of the depositional features of these deposits. The results of the investigations indicate that the fines content of the North-East corner (NE beach) of the ETF exceeds the industry assumption of 15%. This paper shows the results of the investigations at the NE beach, offers preliminary interpretation of the depositional mechanism, and discusses the consequences of this on tailings space planning.

HISTORY OF TAILINGS DEPOSITION AT THE NORTH-EAST CORNER OF THE ETF

Current bitumen extraction at Muskeg River Mine (MRM) produces three tailings streams, namely coarse sand tailings (CST), thickened tailings (TT), and tailings from the solvent recovery unit (TSRU). Despite its name, CST tailings are, a fine grained sand tailings stream with, a relatively low fines content. CST is the underflow stream of the hydro-cyclone used in the extraction process. The hydro-cyclone overflow is, after other treatments, sent to the thickener. The TT tailings are the thickener underflow and are classified as, a medium-coarse silt slurry. The TSRU tailings is the stream produced from the solvent recover unit, and it is a medium silt slurry with an average bitumen (asphaltene) content around 7%. Since start-up, the three tailings streams have been discharged into the External Tailings Facility (ETF). TT and TSRU were initially pumped into two dedicated ponds (Figure 1), whereas CST was used as construction material to raise the dykes or discharged into the main pond.

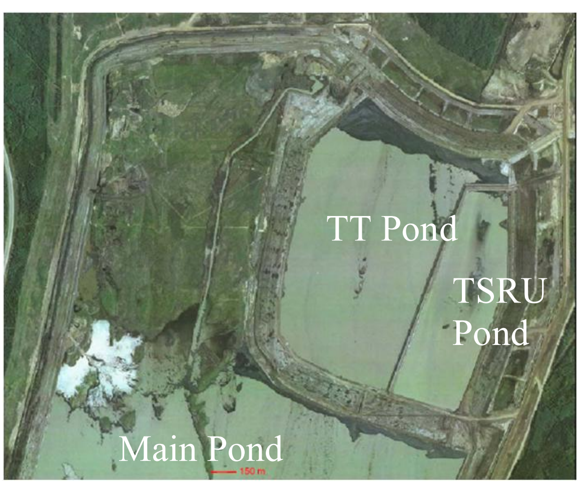

Figure 1. North portion of the ETF in 2003.

At the end of 2004, it was decided to halt the raising of the dykes delimiting the TSRU and TT ponds. As result, a long beach began forming (Figure 2), initially helped by the presence of the overtopped dykes that acted as groins limiting the flow and the movement of sediments. Figure 3 shows the current (June 2012) situation, with a long beach reaching a length of 1,400 m from the tailings discharge points located on the external dyke.

Figure 2. North portion of the ETF in 2005.

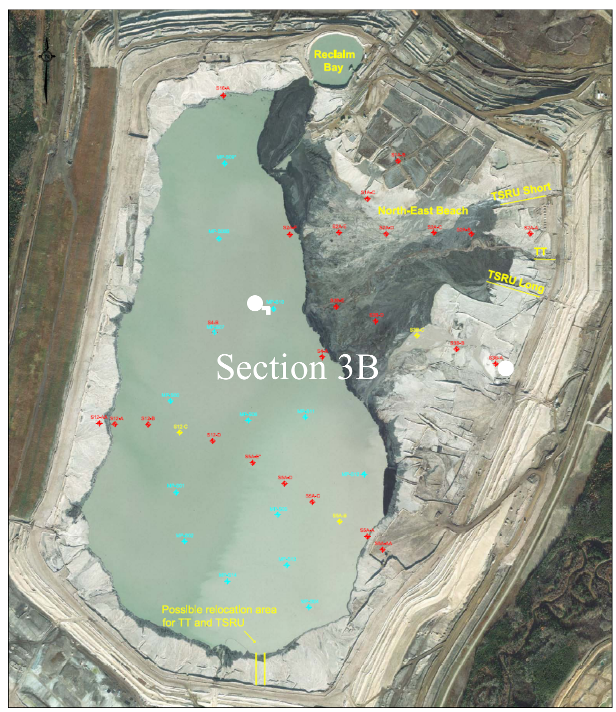

Figure 3. ETF in 2011, testing and sampling locations, and section 3B.

TAILINGS FLOW AND DEPOSITION

Tailings discharge at the NE beach of the ETF has formed two geomorphological zones (Schumm, 1977). The first zone extending few hundred meters from the discharge point (production zone) is a sediment transfer zone. Tailings discharged sub-aerially from pipes form a crater as result of the impact of the slurry jet on the existing Beach-Above-Water (BAW). Subsequently, the slurry flows from the crater onto BAW forming a sinuous single channel (Figure 3) which is dominated by sediment transport. Deposition in this zone is limited to relatively large particles.

The second zone is the sediment deposition zone where the channel divides into many separated sub-channels forming an alluvial fan (Figure 3). The formation of tailings alluvial fans in a tailings pond is well described in Parker et al. (1998a and 1998b). The flow of water and sediments over the fan surface is channelized into a number of braided anabranches. At any given time, the anabranches occupy only a small fraction of the area, but tailings are deposited across the entire fan as the channels shift and avulse. This is also triggered by the relocation of the tailings discharge lines in the area. Inspection of Lidar images indicates that the entire area of the fan is reworked in a matter of five-six months even when tailings lines are not relocated.

The statistics of the tailings lines discharging at the NE beach are in Table 1. The CST stream is, a hyper-concentrated slurry transporting fine sand tailings generally poorly sorted. The TT and the TSRU lines discharge more diluted slurries containing well-sorted mineral solid ranging from medium-coarse silt (TT) to medium silt (TSRU). Whereas CST discharge is not continuous at the NE beach, TT and TSRU lines have been permanently located in this area in the previous years. Table 2 shows the mineral load discharged at the NE beach from 2008 to 2011. During this period, the average fines content of the tailings discharged by the three streams was 35% of the total mineral solid. Around 65% of the fines mass discharged have been captured in NE beach. For sake a comparison, the fines capture efficiency observed at the NE beach is equivalent to a fines capture of 81% for a Composite Tailings (CT) deposit at Sand-to-Fines-Ratio (SFR) of 4.

A qualitative hydraulic characterization of the NE beach was initiated during the 2012 annual tailings investigation and included the survey of channel dimensions, the measurement of flow velocities and suspended load sediment concentration. Run-off channels from each of the three tailings streams were selected and flow and sediment measurements were executed along the channels at cross-sections located at different distances from the discharge points (Figure 4). Since the CST tailings line was not discharging at the NE beach during the execution of tailings investigation, measurement for this tailings stream were executed at a different location, the North-West corner of the ETF where the BAW is much shorter (around 150 m length). Whenever possible, flow depths were measured with a staff whereas channel widths were estimated from GPS readings. Flow measurements were taken at two depths at the centre line of each cross-section to obtain an estimate of the depth-averaged velocities. A FlowTracker Handheld Acoustic Doppler Velocimeter version 3.7 from SonTek was used to measure flow velocities. Surface velocities were also estimated by measuring the travelling time of a floating object. Grab samples were taken to estimate the bed material particle size distribution whereas suspended load was determined by means of a bottle sampler and analysis of the total suspended solids. The data discussed herein are restricted to the flow measurement as the sediment characterization is still under progress. Observed channel widths in the sediment deposition zones ranged from 1.5 m to 4 m, whereas channel depths typically ranged from few centimeters to 0.3 meters. Typical measured flow velocities were between 0.3 and 0.6 m/s for the TT stream and between 0.2 and 0.4 m/s for the TSRU stream. From these velocities, the average channel shear velocity and the average boundary shear stresses were computed. A preliminary assessment of the boundary shear stresses suggests that the measured velocities are compatible with the high deposition rate of fine tailings observed at the NE beach. Future work aims to validate the flow and sediment load measurements.

SEDIMENT CHARACTERISTICS OF THE NE BEACH

Section 3B (Figure 3) crosses the NE beach from South-East to North-West and includes five boreholes drilled from the BAW and borehole drilled from the pond into the BBW. Figure 6 shows the boreholes logs and the location of dykes and surfaces (BAW, BBW, and pond level) along section 3B. The section presented herein refers to June 2011. At each borehole location, continuous sampling was conducted with a sonic sampler advanced from the beach surface or pond surface until refusal. At the same location, Cone Penetration Testing (CPT) including continuous passive gamma and pore pressure readings was conducted until refusal. High passive gamma count in conjunction with drops in pore pressure below the equilibrium pore pressure are a typical signature of layers rich in asphaltene and bitumen and are used to facilitate the interpretation of the tailings depositional environment.

To simplify the interpretation of the tailings deposit (Figure 5), three groups of materials are defined:

- High fines low-bitumen tailings (HF-LB zones in Figure 5);
- High fines high-bitumen tailings (HF-HB zones in Figure 5); and
- Low fines low-bitumen tailings (LF-LB zones in Figure 5).

In agreement with history of the beach, two different depositional environments can be observed. The lower part of the sequence is characterized by HF-HB and LF-LB East of the TSRU dyke and by HF-LB mixed with LF-LB West of the TSRU dyke. East of the TSRU dyke, TSRU tailings were discharged with CST whereas CST and TT tailings were discharged West of the TSRU dyke. In the higher part of the sequence, LF-LB is prevalent in the first 500-600 m from the discharge point, whereas HF-HB is dominant in the more distal part of beach, where a long and continuous depositional length became available after overtopping of the TSRU and splitter dykes. It is likely that the splitter dykes acted as sediment traps preventing transport of part of the fines to the central part of the pond. After that moment, the alluvial fan started forming with channelization into a number of braided anabranches characterized by low flow velocities. This caused progressive increase of the tailings fines content moving from the dyke towards the pond. The fining (hydraulic sorting) of a tailings fan is analytically described in Wright and Parker (2006) and experimentally in Kupper et al. (1992). It is also interesting to note the strong correlation between fines content and bitumen content. If the samples below the elevation of the original splitter dykes are not considered, the population correlation coefficient between fines and bitumen is around 0.7. It is hypothesized that TSRU tailings settlement increases substantially after the first 500-600 m from discharge, when its temperature decreases and the density increases. The presence of the TSRU on the channels beds might also facilitate the settlement of fines from other streams, due to increase of the roughness and the molecular structure of the material. The flocculant used for treatment of the TT tailings might also have an effect on the high fines content of the NE beach. More work is required to validate these hypotheses.

Figure 6 shows the tailings strength derived from CPT soundings executed along section 3B. Tailings East of the TSRU dyke are sand dominated and the strength is expressed in terms of friction angle whereas deposits west of the TSRU dyke are fines dominated and therefore the undrained strength can be derived from the sounding. In general, strength decreases moving from the discharge point towards the pond to the West. Location S3E is about 100 m from the shoreline and the average fines content of the borehole exceeds 50%. At this location, the average undrained strength in the first 30 m is around 15-20 kPa. Measurements of the pore pressure at this location indicate that the equilibrium pore pressure is only about 20% larger than the hydrostatic pore pressure, indicating that excess pore pressure dissipation is quite efficient, most likely favored by the inter-bedding of fine and sandy layers.

Figure 4. Hydraulic measurement at NE beach.

Table 1. Statistics of the tailings lines discharging at the NE beach.

Stream	Flow Rate (m^3/s)	Density (kg/m^3)	SC (%)	D_{10} (μm)	D_{50} (μm)	D_{90} (μm)	Particles <2 μm (% of SC)	Particles <44 μm (% of SC)
Coarse Sand Tailings	1.2	1506	54	74	225	567	1	8
Thickened Tailings	0.7	1130	18	3	35	168	7	60
TSRU	0.4	1060	10	3	20	41	12	74

Table 2. Mineral solid discharged at the NE beach from2008 to 2011.

	CST			TT			TSRU			NE Beach		
Year	Total (Mt)	< 2 μm (Mt)	< 44 μm (Mt)	Total (Mt)	< 2 μm (Mt)	< 44 μm (Mt)	Total (Mt)	< 2 μm (Mt)	< 44 μm (Mt)	Total (Mt)	< 2 μm (Mt)	< 44 μm (Mt)
2008	7.2	0.07	0.6	4.7	0.3	2.8	1.5	0.2	1.1	13.4	0.6	4.5
2009	9.1	0.09	0.7	4	0.3	2.4	1.6	0.2	1.2	14.7	0.6	4.3
2010	3.8	0.04	0.3	1.6	0.1	1	1.2	0.1	0.9	6.6	0.3	2.2
2011	1.9	0.02	0.2	2	0.1	1.2	1.4	0.2	1	5.3	0.3	2.4

MANAGEMENT AND RECLAMATION OF THE NE BEACH

The features of the NE beach discussed in previous sections present a number of opportunities and challenges. One of the most significant opportunities is the possibility of planning for a smaller amount of fine fluid tailings. The decreased amount of fines accumulating in the tailings pond is caused by the increased amount of fines captured in the beaches. In addition, shorter settling time is observed for fine fluid tailings at the Shell Albian Sands operations. It is currently estimated that that fine tailings run-off in the pond goes from a highly dispersed phase (solids content <1%) to a less diluted phase (solids content >5%) in 3-4 months. These two occurrences reduce the need for large fluid volumes to maintain and preserve a sufficient clear water cap on the pond, which is an essential condition to run the bitumen extraction. The reduced need for large fluid volumes has a positive impact on the capital and operating costs and ultimately on the environment. On-going work indicates that at the ETF the beach capture used for planning is around 65% compared to the traditional 50% used in the Oil Sands industry. For the remaining life of the ETF, the tailings fan aggradation needs to be managed to avoid siltation of the water intakes located in the north of the pond. Aggradation rate needs to match pond elevation and dyke raise rates. The amount of sediment deposited on the alluvial fan can be controlled by using the techniques described in Parker et al. (1998a) when preparing the tailings staging plans.

Another opportunity is the possibility of early reclamation. Current plan is to keep the pond in operation for other few years. Shell has recently started preliminary engineering to assess the feasibility of the current approved reclamation plan. One of the first findings is that the NE beach tailings deposit is readily reclaimable by means of conventional engineering methodologies. The assumption that it is possible to place a hydraulically deposited sand layer over the softer portion of the NE beach is supported by evidence and was confirmed by numerical modeling. At the end of operations, the pond level will be maintained constant for a period of time and a crust will form on the NE beach. It is likely that higher water releases will occur in the weaker zones of the NE beach slowing the crust formation process. Successive sand placement surcharges the deposit resulting in an increase in water release and extending the period of water release out in time. Given the one-dimensional nature of the mechanism involved, the net effect of water release will be to maintain full saturation in the cap and to increase time to develop the crust. Surface management techniques, such as ditching, reduce the water table in the sand cap but increase the effective loading on the tails, resulting in more water release in the short period of time. It is essential to find the balance between water release, surface drainage, rate of placement of sand cap, and natural dewatering. From a practical perspective and given that the current observed average solids content is around 65%, numerical modeling indicates that 1-2 years are required to form a crust in the weaker zones of the beach before pacing a sand cap at a rate of 2-3 m/year. A sand cap of 5-6 m, likely sufficient to establish the foundation of reclamation material, can be placed in 2-3 years. It is also expected that surface drainage is needed for 2-3 years before initiating landform grading and placement of reclamation material. This makes the total amount of time required before placing the final reclamation cover between 5 and 8 years. Engineering work is currently ongoing at Shell to refine the preliminary planning premises given above and to initiate detailed reclamation design.

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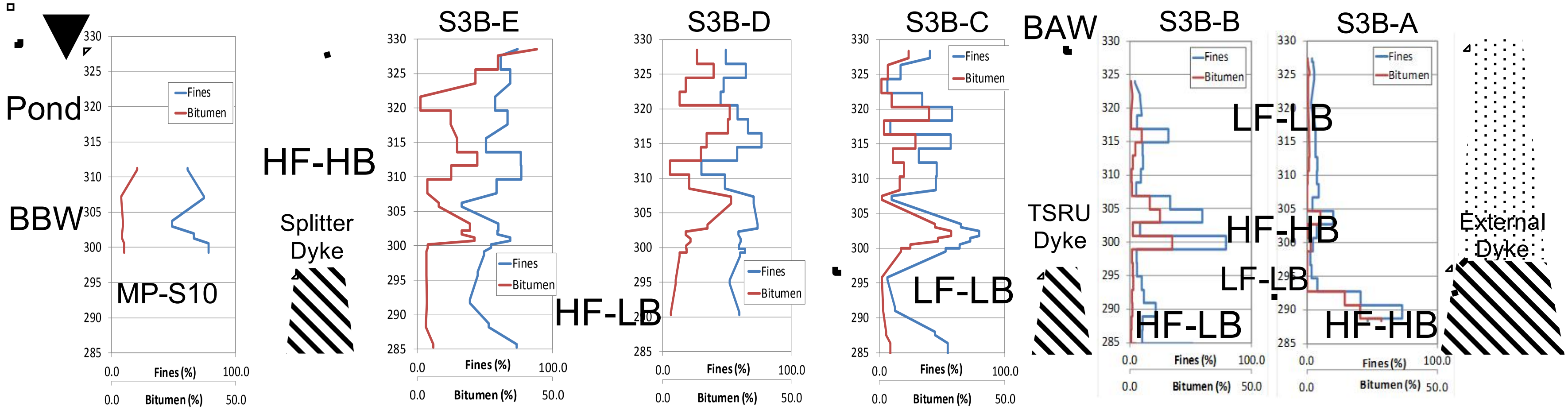

Figure 5. Fines and bitumen content along section S3B.

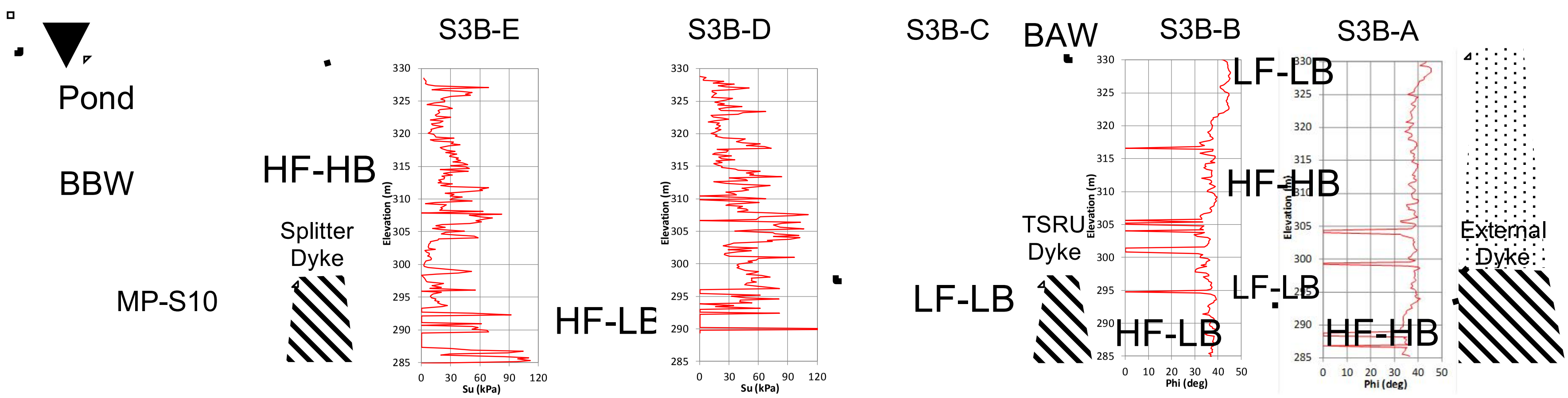

Figure 6. Strength profile along section S3B.

SAND AND FINES MIXING IN FLUID TAILINGS AT THE SHELL MUSKEG RIVER MINE EXTERNAL TAILINGS FACILITY

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ABSTRACT

Shell Albian Sands oilsands operation north of Fort McMurray, Alberta, has been in production since 2002. Since start-up, compacted coarse (sand) tailings has been used to construct the perimeter dykes of the External Tailings Facility while a mix of fine and coarse mineral tailings have been directly discharged into the container within the perimeter dykes. This tailings operation produces solid tailings beaches around the inside of the perimeter dyke and soft fluid tailings beyond the beaches, in the pond. The first fluid tailings investigation was conducted in 2003, with successive annual follow-up programs through 2011. The data collected since start of operation indicate that the depositional environment of the pond and the sediment transport mechanism provide substantial mixing of sand and fines deep in the pond at rates that are quite different compared to older tailings facilities at neighbouring oilsand operations. During the first two years of operation, the fluid tailings consisted almost exclusively of fines particles, using 44 microns as the cut off. Since 2005, coarse tailings has been transported by density flow far down in the fluid tailings where it continuously mixes with fine tailings. This has created a mixture similar to engineered ‘composite tailings’ and presents challenges and opportunities with regard to the reclamation of the fluid tailings and compliance with the 2009 tailings regulation.

INTRODUCTION

The Athabasca Oil Sands Project (AOSP) is a joint venture among Shell Canada Energy (60%), Chevron Canada Limited (20%) and Marathon Oil Sands L.P. (20%). The AOSP operations at the Muskeg River Mine (MRM) plant started in 2003. Since start up the tailings stream remaining after the bitumen extraction has been deposited into an out-of-pit earth structure (tailings impoundment) known as ETF (External Tailings Facility). Tailings are composed of residual bitumen, solvents, water, sand, silt and clay particles. Upon deposition into the tailing ponds, coarser fraction settles close to the discharge point and forms a solid deposit (beach), while the finer part of the tailings remains suspended in the run-off and is transferred to the central parts of the pond, forming a fluid fine tailings stream.

Shell Canada conducts annual tailings investigations to monitor the volume and the properties of the fluid tailings at the ETF. Another purpose of these annual investigations is to have an understanding of the rate of dewatering (sedimentation /consolidation) within the ETF to plan for future reclamation activities. The present paper reviews the results obtained at selected sounding locations in the recent years, from 2009 to 2012. Review of the sounding location profiles indicates that a thick layer of “sand dominated” deposit with significant fines content is forming at lower to middle levels of the pond. At some locations, the sand to fines ratio (SFR) values for samples taken within these layers resembles the SFR values used for production of Composite Tailings (CT). This observation can be regarded as an opportunity for future reclamation of the ETF.

TAILINGS POND INVESTIGATION

To track the changes of tailings characteristics with time, sampling and in-situ measurements have been conducted annually at specific locations within the ETF. Figure 1 illustrates position of eight of these sounding locations. In Figures 2.a to 2.c the variations of Solids Content (SC), Fines Content (FC) and Fines/(Fines+Water) ratio from 2009 to 2012 are presented for MP-S06 as an example.

It can be seen that for sounding location MP-S06, the SC varies from 27 to 81%, FC from 96 to 32%, and F/(F+W) from 24 to 60% from top to bottom of the sounding. Significance of the high F/(F+W) ratios observed at lower half of the pond in creating a relatively non-segregating sand-fines mixture is discussed in the following section.

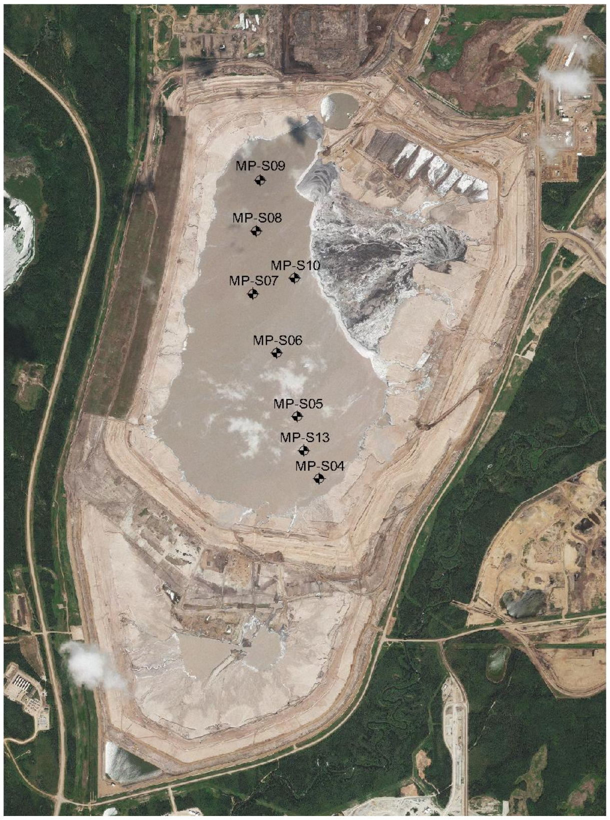

Figure 1. Distribution of Sampling locations within ETF.

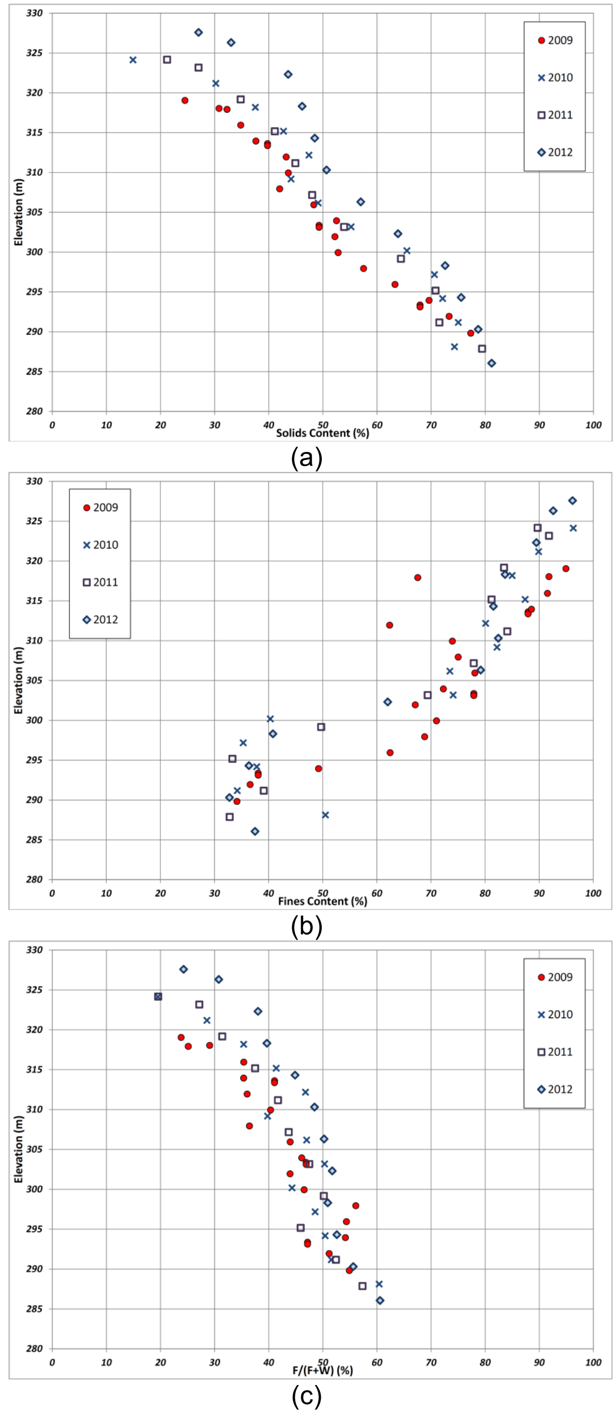

Figure 2. Variations of Solids Content (a), Fines Content (b) and F/(F+W) ratio (c) at sounding location MP-S06 located at central zone of the MRM ETF.

TAILINGS CHARACTERISTICS

The Slurry Properties Diagram (Ternary Diagram) (Scott and Cymerman, 1984; Morgenstern and Scott, 1999) is a useful tool for characterization of oil sands tailings. Figure 3 to Figure 6 show the position of samples taken from the eight sounding locations within the ETF on the Slurry Properties Diagram, from 2009 to 2012. In order to better follow the changes in tailings characteristics at different pond elevations, the samples shown on the ternary diagrams are distinguished according to their position with respect to the pond bottom; i.e. samples within each 5 m layer from the pond bottom to top are marked using a similar symbol.

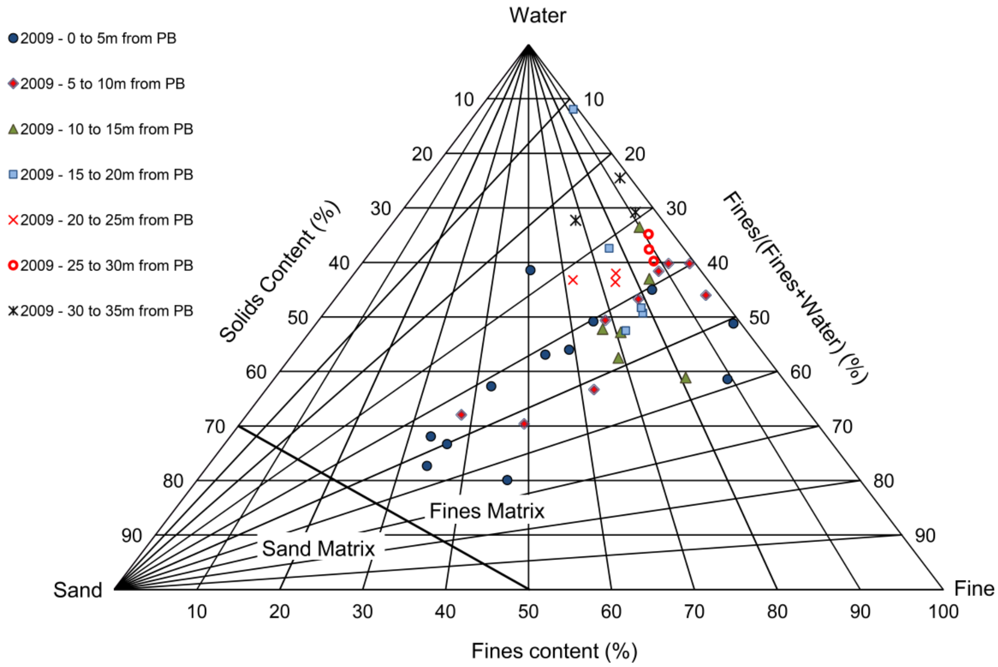

Figure 3. Position of the 2009 samples on the Ternary Diagram. Samples collected at eight sounding locations are distinguished based on to their distance from the pond bottom.

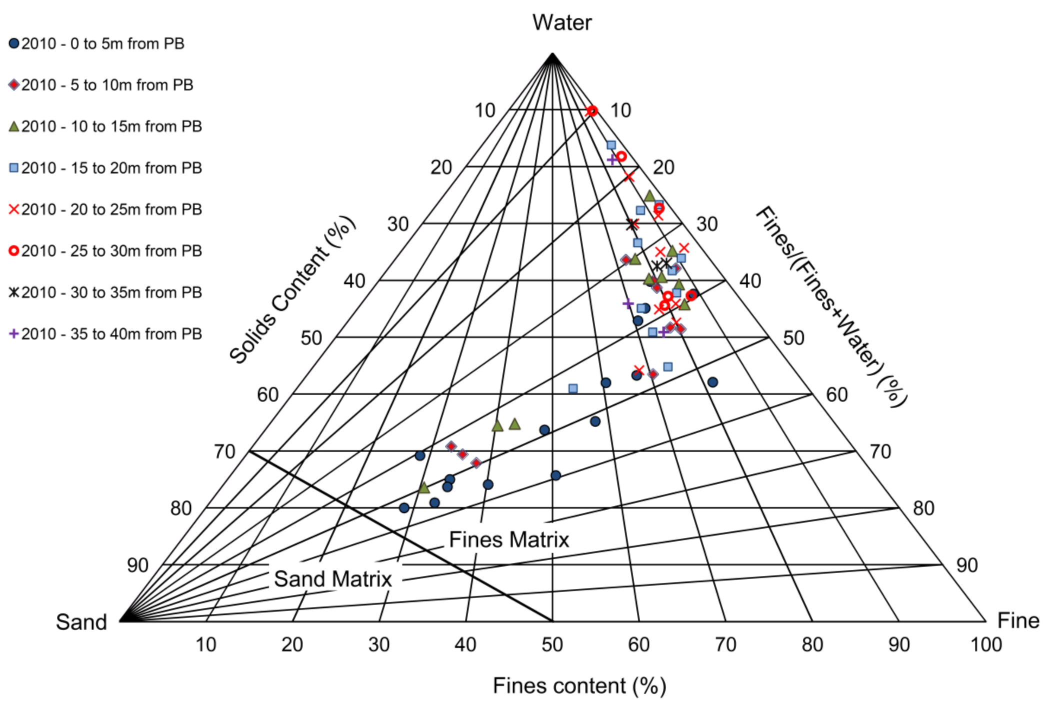

Figure 4. Position of the 2010 samples on the Ternary Diagram. Samples collected at eight sounding locations are distinguished based on to their distance from the pond bottom.

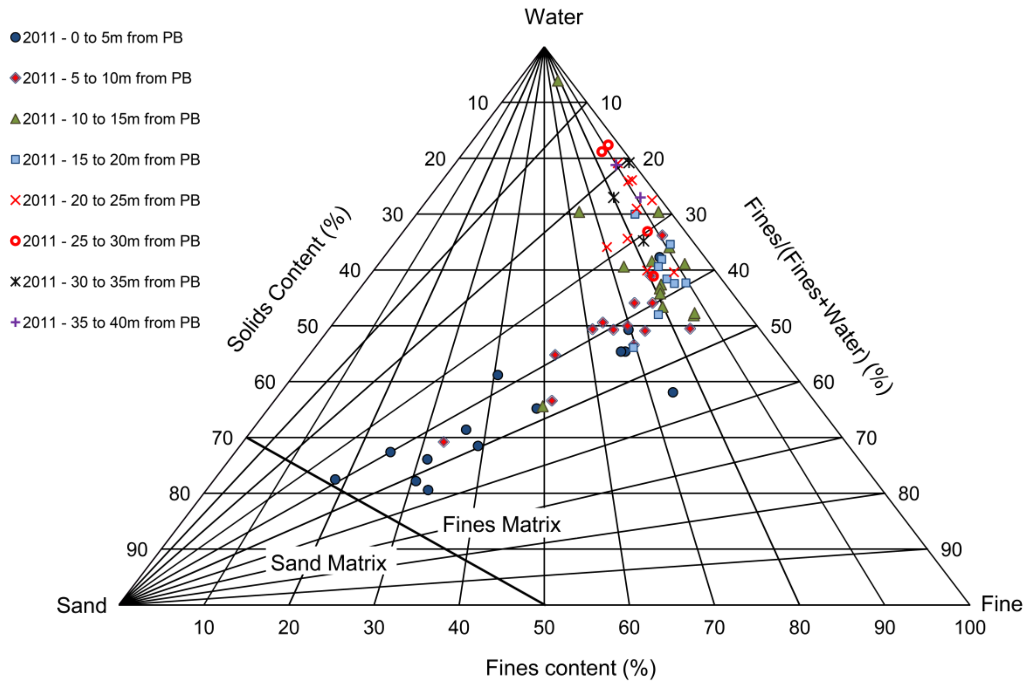

Figure 5. Position of the 2011 samples on the Ternary Diagram. Samples collected at eight sounding locations are distinguished based on to their distance from the pond bottom.

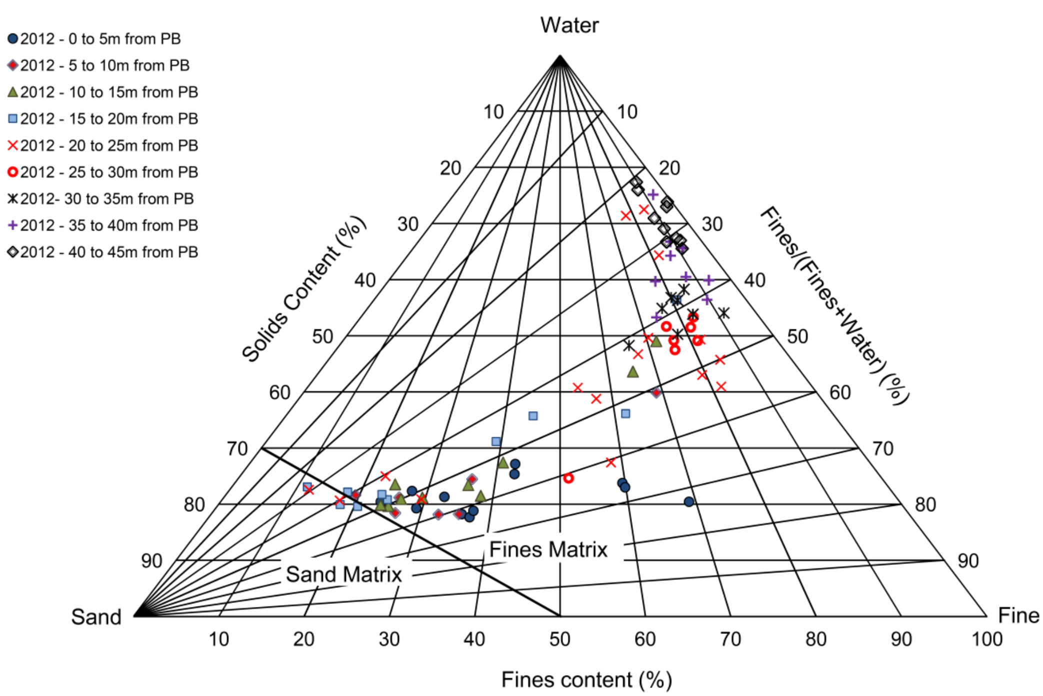

Figure 6. Position of the 2012 samples on the Ternary Diagram. Samples collected at eight sounding locations are distinguished based on to their distance from the pond bottom.

From Figure 3, it can be seen that in 2009 some samples within the lower 10m of the pond have solids content values higher than ~60%, with fines content values higher than 33% (i.e. SFR=2 and lower). The 2010 site investigation results illustrated in Figure 4 indicate that samples in the lower 15m of the pond show solids content values higher than 60% to 80% and a limited number of samples show SFR values close to 2.5. Based on the 2011 investigation results shown in Figure 5, couple of samples in the lower 5m of the pond show SFR values from 3 to 4. The SFR of 4 is the common value used for production of CT (Composite Tailings) in oil sands operations. The results from the 2012 investigation are presented on Figure 6. It can be seen that in the lower 20m of the pond, the number of samples having solids content values higher than 75% and fines content values from less than 20% to 40% (SFR=1.5 to 4) has significantly increased. Formation of such a thick layer of high solids content deposit, which captures a significant percentage of fines, can be considered as an opportunity for faster reclamation of the ETF pond.

Figure 2.c shows the variation of F/(F+W) at sounding location MS-06. It can be seen that the F/(F+W) ratio varies from about 20% at the top elevations to about 60% close to the pond bottom. Higher values of F/(F+W) ratio, particularly at the lower half of the pond create a thick carrier fluid which prevents segregation of the sand particles and their further transport to the pond bottom. It should be noted that a recent study by Mihiretu (2009) showed that a F/(F+W) ratio of 30% is sufficient to create a non-segregating sand-fines mixture at SFR values of 2 and lower without chemical treatment.

In a separate publication by the present authors (Esposito and Nik, 2012) the history of tailings deposition into the ETF and the different tailings streams directed into this pond is reviewed. As stated in this publication, one of the tailings streams deposited into ETF is thickener underflow or TT (Thickened Tailings). Presence of the polymers used for treatment of the TT stream results in flocculation of the fines (clays) and formation of larger particles. As a result, the dewatering rate and the viscosity (shear strength) of the carrier fluid are increased. This can potentially reduce segregation of the sand particles.

While in other oil sands operations it is observed that most of the sand particles settle close to the discharge point and form a BBW (Beach Below Water) or a sand beach below Mature Fine Tailings (MFT), at MRM ETF it is observed that a significant percentage of the sand is transferred to the central zones of the pond, forming a mixture with the suspended fine particles. The following section will discuss the possible reasons for transfer of sand to the central zones of the pond.

TRANSPORT OF FINE AND COARSE TAILINGS

Van ‘t Hoff et al. (1992) and De Groot et al. (1998) described the formation of underwater slopes from discharge of hyperconcentrated sandy slurries. It was found that the formation of underwater slopes is mainly governed by the soil mechanics properties of the sand. Dependency of the slope on the specific slurry flow rate is instead weaker. The sand deposited underwater is always very loosely packed. Loose saturated granular materials show strain-softening response under controlled monotonic strain conditions and are therefore susceptible to liquefaction (Atigh and Byrne, 2004). Such soils may become unstable under controlled loading conditions. Typical monotonic loading conditions can be the deposition of additional material in the upper part of the beach or the reduction in effective stresses associated with pore-pressure rise. Based on field measurements of pore pressure during tidal variations in the Fraser river delta, Atigh and Byrne (2004) found that even under falling tide levels a drop in effective stress can occur as a result of the presence of gas in the pores. The strain softening under monotonic loading conditions is generally a local condition influencing only the most superficial part of the submerged slope (Stoutjesdijk et al. 1998). If liquefaction occurs, sand goes in suspension increasing the local density of the carrying fluid and density flows can be initiated in the form of turbidity currents.

Flume experiments and field observations during the reclamation works in the Netherlands (de Groot, et al., 1988) indicated that at low specific flow rates and with fine sand, conditions similar to those observed at the ETF, slopes develop in a discontinuous and rhythmic fashion. First steep slopes develop caused by sedimentation mostly in the upper zone of the slope close to the waterline. When the slope reaches a critical height or when hydraulic condition changes (for example changes in pond elevation), it becomes meta-stable and turbidity currents may be triggered. The density flow transports sand to the most distal zones of the pond until the density difference is insufficient to sustain the flow. This mechanism flattens the slope and, as consequence, sedimentation resumes mostly in the zone close to the waterline. The sequence repeats continuously and brings high quantity of sand in the distal part of the pond.

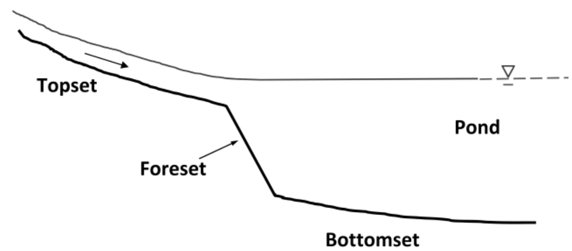

Figure 7. Schematic illustration of the Topset, Foreset and Bottomset. Flow is from left to right (modified from Kostic et al., 2002).

Except for the North-East sector of the ETF characterized by a beach length above water exceeding 1000m, the beach-above-water of the ETF has an average length of 150m (Figure 1). On these shorter beaches, Coarse Sand Tailings (CST) is usually discharged sub-aerially. Despite its name, the solid transported by the CST line is a fine sand tailings, generally poorly sorted. A fraction of the sand of the tailings slurry discharged on these shorter beaches deposits out fluvially to form a topset deposit (Figure 7). By means of the mechanism described previously, the sand is transported down the beach-below-water face to form the foreset deposit of a prograding clinoform. The foreset is deposited at a spatially constant avalanche slope. The slope of the underwater beach in June 2011 is showed in Figure 8. The slope of the foreset is generally between 8% and 14%. The slope of the bottomset is generally lower ranging between 1 and 5-6%. The fines that remain in suspension form a surface plume in the pond out of which the fines gradually settle out. If higher concentrations of fines reach the pond, the fines-laden plume can become sufficiently heavy to plunge and form a bottom turbidity current (Kostic et al., 2002). The fines turbidity current can then override the slopes of the beach below water.

The beach-below-water of the North-East portion of the ETF is also characterized by similar slopes. Due to its length between 1000 and 1400m, it is expected that the tailings lines discharging in this area bring less sand to the foreset. This is confirmed by the higher amount of relocations required to operate the CST lines in the area. The TT line and the Tailings Solvent Recovery Unit (TSRU) tailings lines discharge in this area and discharge a higher quantity of fines. The TT tailings are the thickener underflow and are classified as a medium-coarse silt slurry. The TSRU tailings is a medium silt slurry with an average bitumen (asphaltene) content around 7%. Fines transported by these lines on this beach find an area characterized by a relatively low ratio of boundary shear velocity to fine particle fall velocity. These lines therefore deposit most of the sand in the first 400-500 m of the beach above water and substantial quantities of fines in the distal portion of the topset.

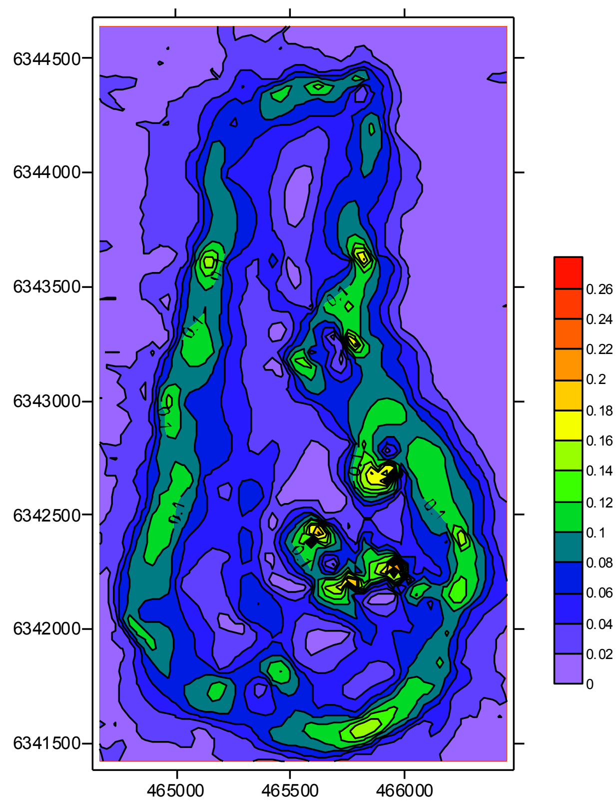

Figure 8. Underwater beach slopes measured in June 2011.

DISCUSSION AND CONCLUSIONS

The annual tailings investigations at MRM ETF indicate that a thick layer of a mixture of sand an fine particles, at solids contents higher than 60% and SFR values from 1.5 to 4 is gradually forming.

At lower elevations this mixture resembles the characteristics of CT, captures a significant percentage of fines and can be considered as an opportunity for faster reclamation of the ETF pond. The mixture has the ability to support a sand cap placed hydraulically by means of conventional technologies. In addition, this material is difficult to pump and, as a consequence, capping in place is the reclamation methodology of choice.

At higher elevations however, low SFR values (i.e. higher fines contents of the deposit) and low solids content pose some challenges for the reclamation of the facility. Fluid tailings with a solids content less than 35% is a likely candidate for pump-and-treat technologies such as centrifugation or Atmospheric Fine Drying. Fluid tailings between 35 and 45-50% can still be capped in place, although advanced technologies might be required. Sand raining or hydraulic placement of TSRU tailings are two of the solutions that will be considered in the near future.

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AN ANALYTICAL MODEL FOR TAILINGS DEPOSITION DEVELOPED FROM PILOT SCALE TESTING

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ABSTRACT

Measurements and observations from bench- and pilot-scale flume and shear cell testing of non-segregating tailings (NST) are incorporated into a semi-empirical analytical model for the prediction of fines segregation for various deposition methods and tailings characteristics. The model allows exploration of the effects of tailings mixture characteristics and tailings placement scenarios on fines capture, and enhances tailings basin management planning.

This paper presents: 1) the theoretical basis of the analytical model; 2) an evaluation of the capability of the model to predict the results observed in flume tests; and 3) predictions of fines capture for schematic tailings basin configurations representing an external tailings facility and an in-pit disposal area.

INTRODUCTION

Tailings in the oil sands industry are traditionally discharged via an open-ended pipe located on a tailings basin dyke. From that point, the tailings typically flow down a sub-aerial beach to the pond edge, where the tailings slurry flows below water, and eventually come to rest. The flowing tailings are subject to shear against the bed throughout the deposition process, causing fines to segregate and sand to settle in the deposit. The un-captured fines end up in the tailings basin as fluid fine tailings (FFT), and may eventually become mature fine tailings (MFT).

The study features laboratory testing, bench-scale testing (shear cell) and pilot-scale experiments (30 m^3 flume) to evaluate three NST placement methods. Pilot-scale studies were conducted to evaluate fines segregation for three different methods for placing NST; open-end pipe (OE), buried pipe (BP) and tremie-diffuser (TD). The primary goal of this study was to determine whether these deposition methods affected the segregation behaviour, and resulting fines capture, of NST as will be produced in the Canadian Natural's planned NST plant. The deposition methods were evaluated in the flume, with the results of the flume tests incorporated in an analytical segregation model, in order to extend the flume test results and evaluate the fines capture when depositing into an External Tailings Facility (ETF) or In-Pit.

Model development extrapolates the laboratory, bench, and pilot data to field scale in order to predict the fines capture expected from a sub-aerial discharge for two NST mixtures (strong and weak), three discharge methods (OE, BP, TD) and both ETF and In-Pit deposition scenarios. Definition sketches of ETF and In-Pit are given in Figure 1.

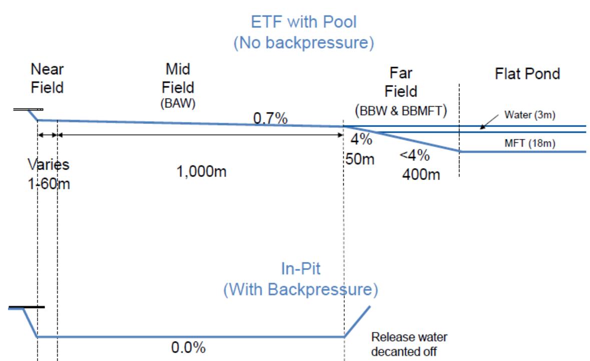

Figure 1. Definition sketch: ETF and In-Pit Scenarios.

A combination of flow types is possible on tailings beaches. Laminar flows on tailings beaches are sometimes referred to as sheet flow. Pirouz et al. (2006) present measurements including velocity and concentration profiles, from a flume situated on a tailings beach fed by prototype tailings. Fitton

et al. (2006) propose that the repetitive channeling flow often observed is turbulent, but that with the commonly observed concave beach profiles (Blight et al. 1985) the flow velocity must drop towards the toe, giving rise to repetitive, fanned, depositional laminar sheet flows. For thickened oil sand tailings, turbulent flow should be unlikely, though not impossible. The study presented here is focused primarily on laminar slurry flows.
Sand settles in laminar shear flow of colloidal-based carrier fluids, as originally disclosed by Thomas (1979). Shear flow testing of mixtures is a method to calibrate shear-settling relationships. The principles of shear cell testing and typical results are described in van Kesteren et al. (2008) and Pennekamp et al. (2010). Shear cells are utilized in the current study for quantification of settling sand under non-sheared conditions (static settling) and shear flow conditions (dynamic settling).

For the purpose of analytical modeling, the following nomenclature is used:

- The Non Segregating Tailings (NST) slurry is defined as a mixture of sand, silt, clay, bitumen and water, in which the solids are not segregating with respect to each component fraction (silt, sand, clay and bitumen). The NST is assumed to be homogenous at the discharge.
- Sand, silt, clay, and bitumen particles comprise the solids fraction of the slurry.
- Bitumen is a fluid, but is considered a component of the fines fraction, and is assumed to be present in negligible quantities.
- The solid particles of the fines fraction comprise silt and clay (all material < 44 microns).
- The carrier fluid is defined as the mixture of fines and water.
- The shear rate profile in the flow of NST is determined by the rheology of the NST.
- The rheology of the carrier fluid determines the sand settling rate, given a shear rate profile in flowing NST.
- The hindered settling of sand is a function of both the volume concentration of sand in the NST and rheology of the carrier fluid.
- The composition of the mixture is characterized by FOFW [weight ratio of fines/(fines+water) expressed as %] and SFR (weight ratio of sand/fines).
- At zero shear rate a unique static segregation boundary exists for a given combination of SFR and FOFW. At higher SFR or higher FOFW the sand does not settle, while at lower SFR or lower FOFW the sand settles.
- When, due to sand settling, sand concentrations increase near the base of the flow, two different types of beds can result: A gelled bed (Talmon and Mastbergen, 2004), in which the carrier fluid has a high apparent viscosity, where the porosity with respect to the sand remains above the maximum skeleton porosity, or a sand skeleton, which occurs when the porosity with respect to the sand drops below the maximum skeleton porosity.

ANALYTICAL MODEL

Segregation Model

The core of the modeling effort presented here is referred to as the segregation model. It is designed to address the behaviour of various NST mixtures in idealized tailings basin deposition scenarios. The model captures the underlying physics and hydrodynamics that govern NST flow and deposition, and is intended as a tool for estimating fines capture and MFT generation as a function of NST characteristics, deposition method, and basin configuration. This basic physical description can be used to evaluate the relative performance of proposed deposition methods, tailings basin configurations, and NST rheologies. The model developed to date is a simplification of reality and does not include all possible factors that could affect fines capture and MFT generation, some of which may be important. In particularly, the model simplifies or neglects the effects of:

- tailings slurry channelization;
- three-dimensionality of flow on wide beaches;
- morphological evolution of the tailings beaches (increase/decrease in slopes with time); and,
- backwater/backpressure effects.

Deposition Physics

The segregation model involves a series of computations that incorporate a variety of input parameters, and ultimately results in predictions of fines capture and MFT generation as a function of discharge method and disposal area configuration. A summary description and governing equations are provided in this Section.

In the flowing NST, the shear rate near the bed is highest, and therefore a depletion of solids develops in the lower portion of the flow. Shear rates in the central and upper part of the flow are lower, and the concentration in these parts of the flow changes more slowly. A definition sketch of the flow including a control volume describing the mass balance of solids is given in Figure 2.

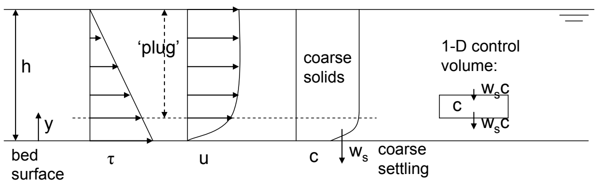

Figure 2. Definition sketch: shear settling in laminar open channel flow, including a mass balance of the suspension.

Flow and Shear

The NST slurry generally behaves as a plastic with flow properties characterized by yield strength and viscosity. The flow behaviour of the NST slurry is, however, assumed to follow a Ostwald-de Waele power-law model (Coussot, 1997). The model is constructed in a manner that allows calibration of the velocity profile (the relationship between velocity and depth in the flow, quantified by a parameter obtained from the rheological measurement). This is an important capability of the model, as the character of this relationship governs the degree to which flowing NST is subjected to shear (the vertical gradient in velocity).

For rheological properties of the slurry mixture the Ostwald-de Waele power-law model is applied:

$$\tau = K_m \left(\frac{dU}{dy} \right)^n \tag{1}$$

Where, τ is shear stress, U is flow velocity, y is vertical co-ordinate, K_m is the viscosity index of the slurry and n is the flow index.

The velocity profile of the suspension is described by:

$$\frac{U}{\bar{U}} = \frac{1+2n}{1+n}\left[1-\left(1-\frac{y}{h}\right)^{1/n+1}\right] \tag{2}$$

Where, h is flow depth, and $\bar{U}$ the depth-averaged flow velocity.

The shear profile is a direct consequence of the assumed velocity profile, as it is computed simply as the vertical gradient in velocity. In relatively 'plug' like flow, the upper portion of the slurry is all moving at the same velocity and there is significant change in velocity in a relatively thin layer just above the bed and below the plug. Consequently, shear rate is high only in this thin layer, and most of the flow thickness is subjected to negligible shear and segregation. The character of the velocity profile is related to the NST rheology—stronger NST (i.e. higher SFR and/or FOFW) leads to more plug like profiles—but little data exists to define this relationship empirically.

For a given NST rheology and depositional geometry, the model computes an equilibrium flow and shear profile for every point from discharge to final deposition. The equilibrium flow depth is commonly approximated in engineering analyses, and involves a force balance between gravitational driving forces and frictional resistance. In general, 'stronger' NST mixtures (higher FOFW and SFR) result in deeper NST flows and slower flow velocities and, therefore, less segregation.

The equilibrium depth of laminar 1-D Ostwald-de Waele power-law flow is:

$$h = \sqrt[2+1/n]{\frac{1+2n}{n}\, q \left(\frac{K_m}{i\rho g} \right)^{1/n}} \tag{3}$$

Where, q is the volumetric flow rate per unit width, ρ the slurry density, g the gravitational constant, and i is slope.

Sand Settling Velocity

The degree to which NST flow is sheared is one of the critical predictions made by the segregation model because the rate at which sand settles from the NST flow is proportional to shear rate (Thomas 1979, Talmon and Huisman 2004, and Wilson and Horsley 2003). The model applies sand settling data in two ways (dynamic and static settling). The first is the sand settling vs. shear rate relationship, which is applied in all situations where NST flow is subjected to shear, in order to calculate the rate at which sand is removed from the flow—and therefore the vertical rate at which the static bed of the flow accumulates sand. The second involves estimation of cut-off SFR values at which sand in NST would either statically segregate (fall out when the slurry is not moving) or would lead to a deposit with impractically low strength development potential. The latter is applied in situations where flowing slurry comes to rest (e.g., the flat pond bottom in the ETF) to check whether sand should stay in suspension.

The dynamic settling velocity of a solitary particle in viscous shear flow is determined by the stress distribution over the particle that co-rotates with the shear flow. This settling velocity is, suitably and pragmatically described by a Stokes type of equation (Talmon and Huisman 2005):

$$w_0 = \frac{\alpha}{18}\frac{(\rho_s - \rho_f)gd^2}{\mu_0} \tag{4}$$

Where, d is solids diameter, α is an empirical constant derived from shear cell testing, ρ_S is solids density, ρ_f is fluid density and μ_0 is the apparent viscosity of the carrier fluid.

The apparent viscosity of the carrier fluid varies with vertical position and is described by:

$$\mu_0 = K\left(\frac{\partial U}{\partial y}\right)^{n-1} \tag{5}$$

Where, K is the viscosity index of the carrier fluid. The rheological parameters of the carrier fluid K and n have been determined as a function of FOFW.

The relative viscosity (K_m/K) is determined as a function of FOFW and SFR for yield strength measured from vane shear tests. The coefficient α is determined by shear cell testing. Shear cell testing is described in van Kesteren et al. (2008). Applying Richardson and Zaki (1954) theory gives the settling velocity in a concentration of coarse solids:

$$w_s = w_0(1-kc)^{\beta} \tag{6}$$

Where, k is an empirical constant (often taken as k=1) and β an empirical exponent, for viscous Newton fluids: β =4.65 according to Rowe 1987). This function is calibrated from shear cell testing and rheological characterizations of the actual thickened tailings (TT) utilized in this study with results: k=1.72, β =3.1, α=1 and $d = 2*d_{50}$.

Concentration Profiles

The shear profile and sand settling velocities discussed above are used in combination with the velocity profile to estimate the vertical distribution of sand in NST flow. The profile is estimated as a function of the evolution of the NST between the discharge pipe and a particular point in the model, as well as the shear rate and settling velocity computed at that point. The velocity gradient is typically steeper closer to the bed, which results in relatively low sand concentrations (by volume) near the bed and an upward decrease in sand concentration with distance from the discharge. Figure 3 shows the general evolution of the vertical sand concentration profile for a control volume of slurry travelling across the subaerial ETF beach. Note that as the slurry enters the beach at t^+ =0, the sand concentration is vertically uniform. As time progresses, however, sand is lost from the base of the flow.

Substitution of Eq.(1), Eq.(5) and Eq.(6) in Eq.(4) gives for the settling velocity of solids just above the bed surface:

$$w_{sb} = \frac{\alpha}{18}\frac{\left(\rho_s - \rho_f\right)gd^2}{K}(1-kc)^{\beta}\left(\frac{\tau_b}{K_m}\right)^{1/n-1} \tag{7}$$

where τ_b is the shear stress acting on the bed surface. Assuming a linear shear stress profile: τ = (1-y/h)τ_b, the settling velocity of solids in the profile is described by:

$$w_s = w_{sb}\left(1 - y/h\right)^{1/n-1} \tag{8}$$

The mass balance of settling solids in the slurry reads (longitudinal gradients are neglected):

$$\frac{\partial c}{\partial t} = \frac{\partial w_s c}{\partial y} \tag{9}$$

To circumvent complexities from the non-linear (1-kc) hindered settling term, this term is assumed to remain constant. For an initial uniform solids concentration (c_0 at t=0), the exact solution of this linearized equation is:

$$\frac{c}{c_0} = \left(t^+ \frac{1/n-2}{(1-y/h)^{2-1/n}} + 1\right)^{\frac{n-1}{1-2n}} \tag{10}$$

Where, t^+ is dimensionless time:

$$t^+ = \frac{w_{sb0}\, t}{h} \tag{11}$$

Where, $w_{sb0} = w_{sb}(c_0)$, is the hindered settling velocity of solids at the bed surface where the shear stress equals τ_b and the solids concentration is equal to the initial (uniformly distributed) solids concentration c_0. Equation (11) has been derived by an analysis of the characteristics of the hyperbolic mass balance equation. Concentration profile development according to Eq.(10) is shown in Figure 3.

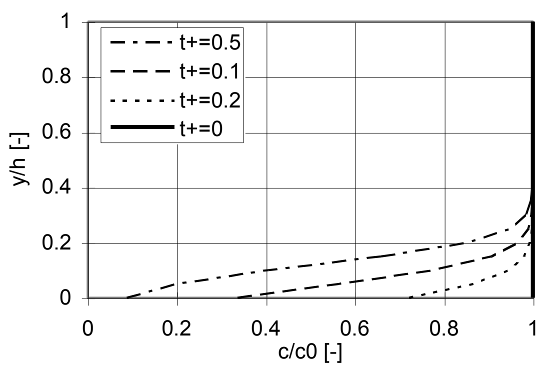

Figure 3. Theoretical suspended concentration profiles for n=0.05.

1-D morphological model

The bed interface has a vertical velocity, called the sedimentation velocity: v_{sed}. Continuity of the suspension is described by (stationary):

$$\frac{\partial qW}{\partial x} = -Wv_{sed} \tag{12}$$

Where, W is the width of the flow, x is downstream distance and v_{sed} is the sedimentation rate.

The sand continuity equation of the suspension is (stationary):

$$\frac{\partial qW\overline{c}}{\partial x} = -Wv_{sed}c_{bed} \tag{13}$$

Where, $\overline{c}$ is the depth averaged concentration, calculated from an analytical integration of the concentration profile Eq.(10). Combination of Eq.(12) with Eq.(13) gives the sedimentation rate:

$$v_{sed} = \frac{\partial \overline{c}}{\partial x}\frac{q}{\overline{c} - c_{bed}} \tag{14}$$

Inputs needed for this calculation are:

- a formulation of the width of the flow as a function of distance: W=f(x),
- the sand concentration of the bed layer: c_{bed}.
- the mean solids concentration of the suspension $\overline{c}$ as a function of distance.

The running time t, which is needed for calculation of $\overline{c}$ is:

$$t = \int_0^x \frac{1}{\overline{U}}dx = \int_0^x \frac{h}{q}dx \tag{15}$$

Outputs are the flow rate per unit width, q, as a function of distance and the sedimentation rate v_{sed} as a function of distance.

Deposit Characteristics

The final component of the core segregation model involves estimating deposit characteristics (and therefore fines capture and MFT generation) as a function of the physical processes discussed above. The model allows for three basic types of material that derive from the original source NST:

1. A settled sand deposit;
2. NST with non-segregating properties; and
3. Segregated MFT.

The first two accumulate captured fines. The third represents those fines not captured. A settled sand deposit results from those situations in the model where NST flow is subjected to shear to such a degree that sand is removed from the flow and left on the bed. This settled material is assigned a loosely-packed sand porosity, or higher porosity, based on measurements of settled sand in flume tests. All pore spaces are assumed to be saturated with carrier fluid having the same FOFW as the NST carrier fluid. This, in turn, allows computation of fines capture in this type of deposit. Due to the shear-related removal of sand from the NST flow, the characteristics of the flow itself change with distance from the outlet. In scenarios where the SFR stays above the static segregation boundary found in shear cell testing, the NST maintains its non-segregating properties from outlet to final deposition. In these cases the material is assumed to deposit as NST, and all non-segregated fines are assumed to be captured. However, in certain scenarios, the NST is “degraded” (due to the settling and removal of sand) to the point where the slurry would continue to segregate even when it is not moving, or it would develop strength over an impractically long time period.

MFT is the final material accounted for by the segregation model, which produces segregated MFT via two processes; when NST is degraded to the point that it segregates upon stopping, as discussed above, or upon encountering either roll wave (low Richardson number) or turbulent (high Reynolds number) conditions in the ETF pond. In either of these cases, the slurry that encounters these conditions is assumed to segregate into a settled granular sand layer and MFT due to dilution with pond water.

FLUME TESTS

Flume tests were conducted on-site. Tailings representative of planned Canadian Natural NST were employed, created by mixing borrow sand (d50=220 micron) with TT created from Canadian Natural floatation tails. The flume is 25 m long, 1 m wide and 1.2 m deep. With obstructed flume exit (effectively an In-Pit configuration), different discharge options were tested first: OE, BP and TD. After these flume tests were completed, two tests were conducted to simulate flow on an ETF beach, see Figure 4. In these flume tests the obstruction at the end of the flume was removed and all NST reaching the end of the flume was recycled back through a mixing tank and re-deposited at the upstream end. The test conditions are summarized in Table 1.

Time series of in-situ conductivity were measured by CCM probes installed in the flume. Measured mud lines are shown in Figures 5 and 6. In FL09 (weak NST) a wedge-shaped sand layer developed near the discharge.

Table 1. Test conditions Flume test

	FL09 Weak NST	FL10 Strong NST	
i [-]	0.0066	0.0066	floor slope
SFR [-]	3.8	5.9	
FOFW [%]	17	22.8	
Solids [%]	50	67	
Q [m^3/s]	0.005	0.005	flow rate
V [m^3]	15	19.6	total NST

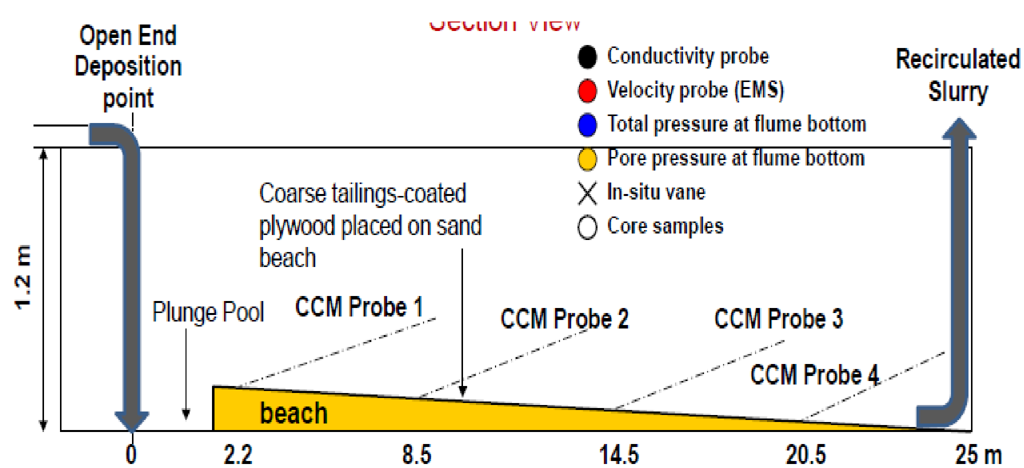

Figure 4. ETF flume configuration.

An example of a velocity profile measured in FL10 is shown in Figure 7. The velocity profile was translated from flume bottom to free surface over approximately two minutes. Samples were collected from the bed, the feed line, and the discharge end of the flume.

FLUME SIMULATIONS

For model validation purposes, the segregation model was adapted to simulate the particulars of the recirculating flume geometry. Modifications were relatively minor, and primarily involved accounting for the fact that all NST reaching the end of the flume was recirculated. The bed accumulation in the flume was computed as an average value along the flume length. That is, the model was not utilized to predict flow or bed behaviour at particular locations along the length of the flume—it was configured to simulate the bulk behaviour of NST in the flume with time.

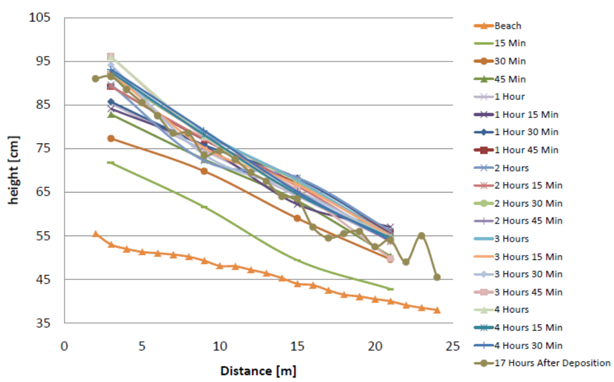

Figure 5. Mud-lines as a function of time in FL09.

In FL09 (weak NST) the conductivity probes measured high sand concentrations, of the order of those that characterize loose packing of a granular bed, near the floor of the flume. This was confirmed by sampling of the bed at the conclusion of the test, which showed. The test showed the slurry segregated quickly. Figure 8 shows the comparison between segregation model predictions of mud-line elevation and slurry sand concentration, and real-time measurements of mud-line and sand concentration during the flume test.

No granular bed developed in test FL10, with strong NST deposited in the same manner as FL09. A bed layer developed with what we characterize as gelled characteristics. Calculation results are shown in Fig.9. Despite accounting for relatively channelized flow (flow was confined to channels of approximately 0.5 m width during this test), the model predicts a monotonous decrease in mud-line elevation that contrasts with the more constant flume measurement. This difference is due to the fact that the model does not include gelled bed dynamics. That is, the model assumes a rigid bed layer and the flume test data show a smooth transition between equilibrium flow suspension and static bed.

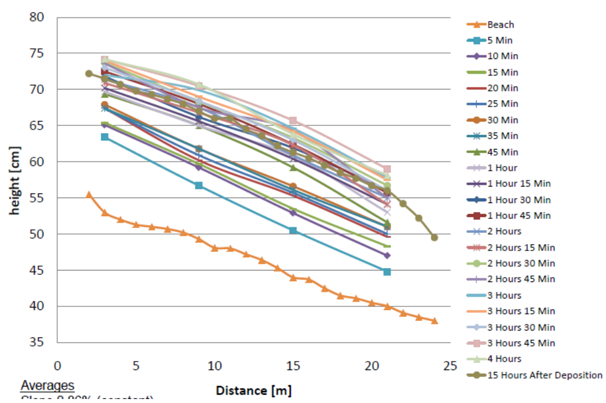

Figure 6. Mud-lines as a function of time in FL10.

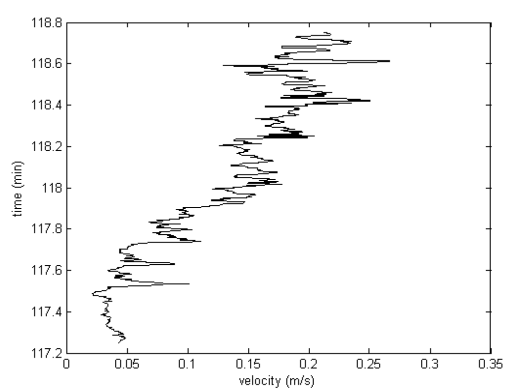

Figure 7. Velocity profile measured in FL10.

Gelled layers add to the complexity of the flow and deposition modeling. Such layers are an exception rather than a rule. They have been encountered before in a small number of pipeline flows and open channel flows of engineered slurries. There is some laboratory data available on comparable segregation in pipeline flows. Hill (1996) reported on segregated concentration profiles in pipeline flow. Hill and Shook (1998) show results where the bed layer has concentrations below 50% by volume. Gillies et al. (2001) published on segregation effects in pipeline flow. Graham et al. (2002) concluded from ERT measurements in a closed pipe that wholesale sliding of the bed layer occurred but also that some shear could have been present in the bed. Pullum et al. (2010a) reported on the implication of bed-presence in prototype tailings pipelines with segregating tailings slurries on pressure losses, and Talmon (2010) noted the inability of such bed layers to develop Coulomb friction, like granular deposits do.

There is published laboratory data on open channel laminar flow of segregating synthetic thickened oil sand tailings (TT): Sanders et al. (2002), Spelay et al. (2006) and Spelay (2007). These experiments measure the flow in a slightly tilted horizontal conduit of circular cross-section of which some ceiling sections are cut away. These studies show measured concentration and velocity profiles that show the formation of a bed layer with a concentration lower than that of a granular layer in which the coarse solids are in mechanical contact. Spelay (2006; 2007) took syringe samples and found that the clay content in the lower layer remained the same but the coarse grained content was significantly higher than in the overriding slurry. Comparable sampling was conducted in gelled bed layer tests described in Talmon and Mastbergen (2004), with a similar result.

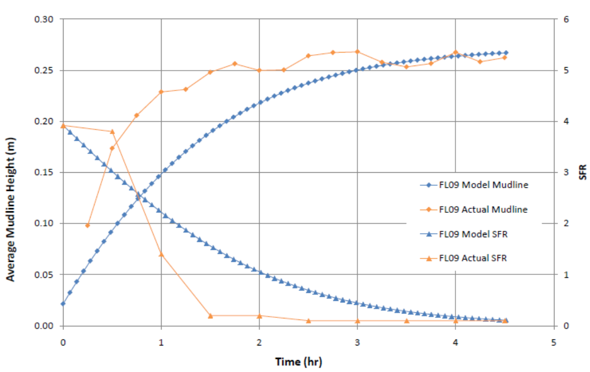

Figure 8. Weak NST (FL09) actual and predicted mudline and SFR.

It appears the FL10 flume test likely resulted in the formation of a gelled bed. These dynamics are not included in the present version of the model, so model predictions do not match FL10 well.

PROTOTYPE CALCULATIONS

Disposal Area Geometry

To date, modeling efforts have focused on two general tailings disposal configurations - ETF and In-Pit. The model is used to simulate the behaviour associated with a single discharge point, which conveys approximately one-tenth of the total proposed, real-world, NST production. The estimates of fines capture, however, apply to the entire basin, as they are expressed as a fraction of NST discharge.

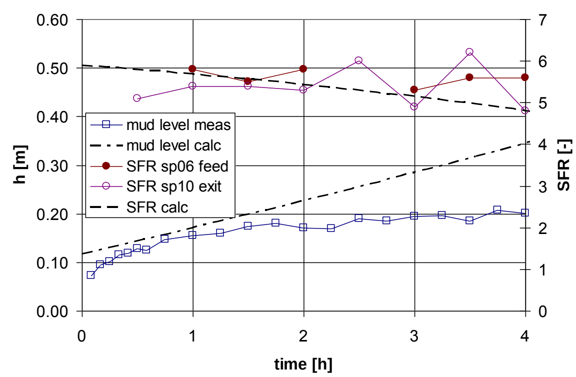

Figure 9. Strong NST (FL10) actual and predicted mudline and SFR.

ETF

The ETF is assumed to have a 1,000m long beach with a uniform slope of 0.007 (Figure 1). At the end of the beach the NST flows into ponded water over MFT. In this pond, the NST flows under water for 50 m on a slope of 0.04 (4%), and under MFT with a density of 1,200 Kg/m^3 for 450 m on a slope of 0.04. Upon reaching the flat pond bottom, the model assumes that the NST comes to rest.

In-Pit

Unlike in the ETF, NST flow slope in the disposal area is nearly always influenced by the presence of previously deposited NST in the basin. NST flow would only be subjected to relatively steep pre-existing mine pit slopes during the first months of basin filling, and those effects can be minimized by depositing at the lowest spot. The slope over which

NST will flow is assumed to be equal as the theoretical slope that would be formed during an NST 'slump test.' The slope is therefore variable, depending on source NST rheology. Within the range of planned Canadian Natural NSTs, this generally results in slopes seven to tenfold shallower than those expected on the ETF beach. The significantly lower slope leads to significantly higher equilibrium flow depths and therefore considerably reduced shear and segregation.

Deposition Method Specifics

The TD discharge structure (van Kesteren et al. 2008) was the only method of the three tested that lead to laminar slurry flow in the near field. BP and OE discharge structures lead to turbulent flow, which flume test results indicate leads to erosion of the substrate. In the modeling program, the TD discharge is assumed to produce a progressively widening and thinning flow, which ultimately enters the pond with a width of 240 m (over 1000 m travel distance across the subaerial beach). The BP and OE discharge methods have shown a tendency to channelize NST flow in flume tests and are assumed to do so in the wide near the discharge, and widen to 10 m near the ETF pond. The BP discharge is assumed to be 1.5 m wide near the discharge, and widen to 7 m near the ETF pond. model. Based on flume observations and scaling, the OE discharge channel is assumed to be 2.5 m wide near the discharge, and widen to 10 m near the ETF pond. The BP discharge is assumed to be 1.5 m wide near the discharge, and widen to 7 m near the ETF pond.

Results of Prototype Calculations

A number of disposal area dn NST rheological configurations were run in the model, in order to evaluate the relative effects of important control parameters.

Model results show that the chief controls on fines capture in the settings explored by this study are:

1. **The quality, or strength, of the input NST.** Figure 10 shows results from a series of model runs with both weak (SFR = 4; FOFW = 20) and strong (SFR = 5; FOFW = 25) NST with each of the discharge configurations. Note that the total fines capture in all sub-environments [beach above water (BAW), beach below water (BBW), and pond bottom] is at least two-fold higher for strong NST scenarios.

2. **The disposal area configuration.** Figure 11 shows that In Pit deposition scenarios result in significantly higher fines capture than ETF scenarios, using the same deposition method (OE). This result is primarily due to fact that NST discharged at the bottom of an In Pit will be subject to significantly less shear than NST that flows a significant distance across an ETF beach.

These results are informative, but are clearly related to the present configuration of the model. That is, the present modeling scheme assumes pit-bottom discharge in the In Pit depositional scenarios, as well as relatively sheet-like flow from the TD – two phenomena that are conducive to fines capture. There are multiple factors that could affect deposition in the In Pit disposal configuration, such as channelization, that are not accounted for in the present version of the model. These additional factors are the subject of ongoing and future research.

CONCLUSIONS

1. The combination of rheological characterization, shear cell testing, pilot flume testing, and analytical modeling has led to a new approach for evaluating NST behaviour and predicting fines capture.

2. The segregation model presented here, while subject to a number of assumptions that will be addressed in future model implementations, is suitable for application over a wide range of discharge and disposal area configurations. As such, it is a valuable tool for future tailings planning, and comparison of deposition strategies.

3. Initial model results indicate that NST quality and disposal area configuration are first-order controls on successful fines-capture strategy. Deposition methods that induce backpressure/backwater conditions, such as bottom-up filling or other strategies that minimize shear of flowing tailings, are critical for optimal fines capture.

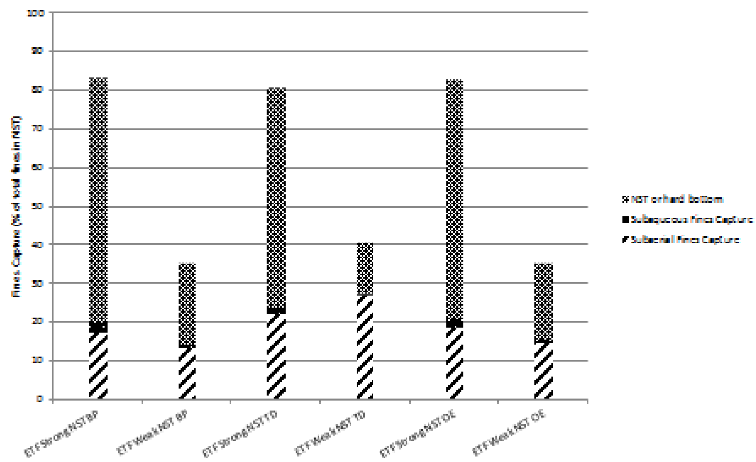

Figure 10. Predicted fines capture by deposition method – ETF configuration.

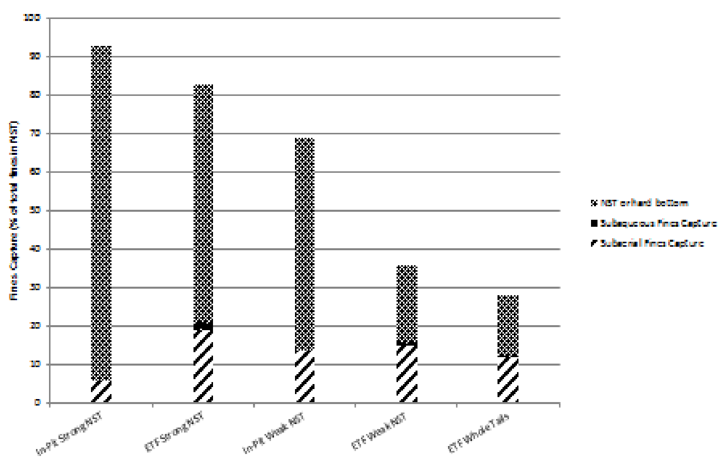

Figure 11. Predicted fines capture by basin configuration – OE deposition method.

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Session 2

Tailings Management 1

OIL SANDS TAILINGS: A LIQUIDS PROBLEM OR A SOLIDS PROBLEM?

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ABSTRACT

Considerable research on the characterization, treatment and management of oil sands tailings has been conducted over many years. Researchers have approached the problem of tailings treatment in one of two ways – as a liquid that requires removal of solids or as a solid that requires removal of liquid. This difference in approaches is important because the way that we frame and approach a research problem will dictate the way that we design the study, collect and analyze the data, and present the results.

Therefore it is not surprising to see numerous journal and conference papers written from the solids-problem perspective that provide little or no information on the amount or quality of release water. Similarly, there are numerous journal and conference papers written from the liquids-problem perspective that provide little or no information on the amount or quality of solids produced.

From an environmental management perspective it is critical to understand the characteristics and fate of both components following treatment so that industry and regulators can choose an optimal solution and avoid unintended consequences. This paper encourages research funders and researchers to consider integrated studies to provide a comprehensive understanding of the implications of various tailings treatment solutions.

INTRODUCTION

Considerable research on the characterization, treatment and management of oil sands tailings has been conducted over many years (e.g., BGC Engineering Inc. 2010, Fine Tailings Fundamentals Consortium 1995, Fuhr et al. 1993, Kasperski 1992, Long et al. 2010, Sobkowicz 2011).

Significant volumes of tailings are produced during the oil sands extraction process. Broadly speaking the tailings are comprised of two major components – liquids and solids – each of which has numerous constituents that determine the behaviour of the individual components and the whole tailings. This complex mixture presents a number of challenges for process and environmental managers.

How we approach these challenges depends to a large degree on what our objective is – for example, recycling water for reuse in the processing plant or utilities plant, treating the water for release to the environment, or creating a solid surface for reclamation.

PERSPECTIVES ON OIL SANDS TAILINGS TREATMENT

A survey of over 65 people involved in the management of oil sands tailings was undertaken to determine how people view the treatment of tailings.

The survey simply asked, when you think of oil sands tailings remediation, do you think of it as:

1. A solids problem requiring liquids removal OR
2. A liquids problem requiring solids removal

Survey respondents were also offered the opportunity to provide additional comments on the subject.

Forty-one responses were received with many of the respondents providing additional information. There was an almost even split between people who view treatment as a solids problem and those who see it as a liquids problem (Figure 1). Twenty per cent of the respondents indicated that they see it from both perspectives.

A couple of respondents took exception to the use of the survey term *remediation* when used in conjunction with oil sands tailings, indicating instead that the correct term is *treatment*. This reaction points out the importance of terminology (and all the baggage that accompanies various terms) in communicating research needs, proposals and results.

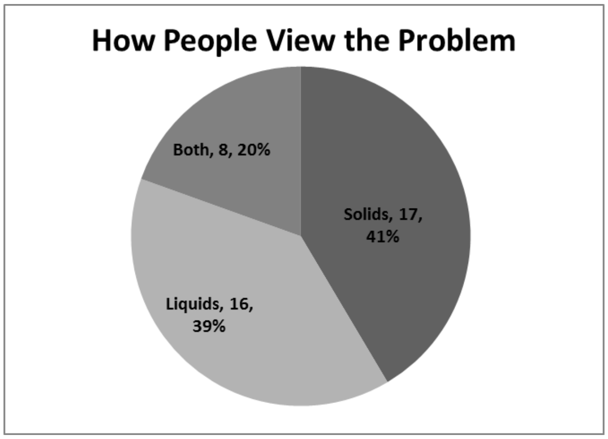

Figure 1. Survey Responses.

The Liquids Perspective

Tailings management is
water management!
There isn't really a tailings problem,
it's a water problem.

The majority of people who viewed tailings treatment as a liquids problem noted that the primary issue is water management because it is a water-based extraction process. The water in tailings is an asset whose value must be maximized by recycling for use in the processing plant or utilities plant.

Others noted that the water will need to be treated for eventual release to the environment, and in particular indicated that removal of the solids is an important step in water treatment. One suggested it is the physical properties of the solids that are an issue, not any chemical contributions they may impart to the water problem.

Another indicated that preventing and treating seepage water is an area of interest.

The Solids Perspective

We start with a solid (oil sands) and add
water – one of the by-products is tailings, a
solids/liquids mixture.
To get back to the core of what we started
with, we need to remove water!!

The majority of those who view tailings treatment as a solids problem focused on the need to remove water to create a solid, trafficable, and reclaimable surface.

Others noted the negative effects of water in the solids matrix on geotechnical stability, thus the emphasis on removing water. One noted that one intent of the Energy Resources Conservation Board's 5 KPa requirement in Directive 074 (Energy Resources Conservation Board 2009) was to ensure that pore water pressures were dissipating actively.

The Solids and Liquids Perspective

Over the years I've come to recognize it as
both!

A number of reasons were provided by those who viewed tailings management as both a solids and liquids problem:

- *It is a sequential issue* – treatment of fluid fine tailings as first a "liquids problem" (i.e., dewatering and treatment of the process-affected water) and then a "solids problem" (i.e., reclamation of the dedicated disposal areas).
- The *answer changes depending on where we are in the process*. At the outset, the tailings are clearly a liquid "contaminated" with solids. At some point in the process of removal of the water, the nature of the problem shifts to becoming a solid "contaminated" with liquid.
- *Focus on the components, not the whole* – tailings is a mixture of solids and liquids, each of which had some baseline characteristics; the goal of treatment is to get the two components back as close as possible to their baseline conditions.

Other Factors Affecting Perspective

Some respondents confessed to their answer being biased by their educational and professional pursuits:

- I think of it as a multi-media problem (i.e., water AND solid) but, I have to admit, I deal with it more like a water problem. My first reaction was "of course, it is both". However, I realized that although in concept I think both media, in experimental design I treat it more like a water problem.

- Our tailings-related research has primarily been in how tailings ponds seepage could potentially affect groundwater and surface water so we've usually thought of it as a liquid problem.
- I fall into the 'solids problem' camp because of the bias in my research programme: de-watering of mature fine tailings.
- As a geotechnical person interested in consolidation, I guess I really think of it as a solids problem requiring water removal. Water treatment folks might think of it the other way around.

One noted that values (personal or societal) affect how we view the problem:

- It is all about competing values. If the dominant value is progressive reclamation, then oil sands reclamation is all about a solids problem requiring liquids removal. If other values are respected (overall environmental impact, resource utilization, cost, etc.) the emphasis changes and could become a liquids problem.

Finally, it is worth noting that changing regulatory requirements can affect how the problem is viewed. For example, the emphasis may have shifted somewhat from liquids to solids with the introduction of Directive 074 (Energy Resources Conservation Board 2009).

PERSPECTIVE AFFECTS RESEARCH AND MANAGEMENT

This difference in approaches is important because the way that we frame and approach a research problem will dictate the way that we design the study, collect and analyze the data, present the results, and form conclusions.

Therefore it is not surprising to see numerous journal and conference papers written from the solids-problem perspective that provide little or no information on the amount or quality of release water. Similarly, there are numerous journal and conference papers written from the liquids-problem perspective that provide little or no information on the amount or quality of solids produced.

Since environmental management goals and objectives established by industry and government are often informed by research they may be indirectly affected by biases introduced by the researcher's perspective on the problem.

Reasons for Perspectives-Based Research

There are a number of reasons why research, and the subsequent conclusions and recommendations, is constrained to one perspective or the other. These can include:

- *Researcher expertise* – individual researchers (and, in the case of academic researchers, their students) are likely to focus efforts on areas where their expertise and equipment lie.
- *Student-based research* – since much academic research is done by grad students, the questions must be focused rather than broad to fit within the normal time to complete a degree, and designed to ensure the success of the individual rather than a team (admittedly the Principal Investigator may have an overall program that the student work fits into).
- *Cost* – research costs can be better controlled when the question and number of researchers is constrained. One might also believe that research proposals may be better received when the budget is low (i.e., focused on a project) than when it is much higher (i.e., focused on an integrated problem and suite of solutions).
- *Current priorities* – when there are limited resources available the tendency is to focus on the most immediate needs. These are often specific problems rather than broad issue areas.
- *Projects vs. programs* – although this is changing, there is a tendency to request research on a project-basis rather than an overall program-basis.

A SOLUTION

From an environmental management perspective it is critical to understand the characteristics and fate of both components following treatment so that industry and regulators can choose an optimal solution and avoid unintended consequences.

Wells (2011) describes the technical disciplines involved and the need for a consolidated approach:

Two primary areas of science and applied science have dominated the work thus far: the science pertaining to slurries, and the science pertaining to earth materials. The science pertaining to slurries is predominated by mineral process engineering while those in the science of earth materials by geotechnical engineering. These two pursuits, while describing very different aspects of tailings behaviour, are both required considerations if one is to develop field scale, successful technologies.

The best path forward then would be for research providers and those requesting research services to build on the strengths of perspective- and project-focused researchers but think in terms of broader, integrated programs. In this context, a research program is a coordinated set of projects undertaking related research.

Alberta has the skills and knowledge to be able to accomplish this goal. Indeed, the province has a rich history of establishing programmatic research efforts – e.g., Alberta Oil Sands Environmental Research Program (AOSERP), Alberta Oil Sands Technology and Research Authority (AOSTRA), the Fine Tailings Fundamentals Consortium (FTFC), and the Northern Rivers Basin Study (NRBS).

Academic institutions and non-academic research organizations (e.g., Alberta Innovates – Technology Futures, CANMET) are well positioned to offer up integrated teams to address multi-disciplinary research programs.

Those soliciting research, whether government departments, research management organizations (e.g., Alberta Innovates – Energy and Environment Solutions) or industry (e.g., Canada's Oil Sands Innovation Alliance), are also well positioned to think in terms of program-based research rather than individual projects. An example of a detailed and integrated list of government tailings research requirements can be found in the 2011 *Environmental Protection and Enhancement Act* approval issued for Total E&P Canada Ltd.'s Joslyn North mine (Alberta Environment 2011; pp. 65-66, section 6.4.10).

One key to successfully implementing program-based research is the designation of an individual or team dedicated to ensuring integration, collaboration and knowledge transfer amongst the researchers and clients. Their primary role is to provide the linkages and feedback loops between what are essentially individual projects that work together to achieve an overarching goal.

In this collaborative, program-based world a researcher may still be focused on the geotechnical properties of the solids produced by a novel dewatering technique, but behind the scenes there is a coordinator linking other researchers who are:

- investigating how the solids will behave as reclamation substrates;
- characterizing the liquids produced and their environmental impacts;
- looking at methods to treat the produced liquids;
- assessing the environmental implications of the solids produced from the liquid waste treatment;
- determining the incremental production of greenhouse gases resulting from the treatment technology and associated reclamation; and
- evaluating the costs of each of the techniques in the chain.

As a result, we arrive at an end-to-end assessment of the benefits and implications of a suite of inter-related solutions to the tailings management problem. This is in contrast to the existing project-based research which provides important answers to only a portion of the overall question.

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INSIGHTS INTO OIL SANDS TAILINGS DEVELOPMENT PROJECTS

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ABSTRACT

The nature of the Oil Sands industry places somewhat unique demands on the design, engineering and management of tailings development projects, designs and field trials.

Among the many challenges faced by tailings development projects are: scale of operations and scale-up; public, environmental and regulatory requirements; cross-functional demands and co-ordination; seasonality, schedule and timelines; managing continuous project changes and change of scope, schedule and priority level.

Faced with increasing demands to deliver tailings solutions, the implementation of tailings technology development projects has become more and more challenging, as a myriad of competing demands require answers simultaneously, against a backdrop of multiple variables.

In this paper the authors describe some of the unique challenges to project managers in tailings development, and offer some tools, insights and approaches into successfully addressing these demands, including: the development of project objectives, potential remedies in considerations of scale-up; proactive anticipation and nimble response to client requirements; understanding, experience, track record and continuity; mobilization and efficient use of multidisciplinary resources; decision-making, risk assessment and technical review; cost tracking; timely and effective communication; careful record-keeping and document tracking; sustained team motivation and attention to detail.

INTRODUCTION

Tailings technology development projects in the Oil Sands over the past two decades have met with considerable difficulties in achieving the objectives which were originally set. Why is this so? What are the pitfalls facing Oil Sands tailings technology development, and how could they be better handled?

The business, public and regulatory environment is constantly changing for Oil Sands tailings. Objectives which were entirely valid and appeared most reasonable to even the most experienced eye at the outset of the project, seem to become dated and flawed in quite short time frames - in a matter of months rather than years.

Faced with these and many other constraints to delivery over the past five years, the authors have developed a number of techniques and key learnings which hold promise in enabling the timely completion of tailings development projects within budget.

THE TOP NINE CHALLENGES

1. Clear, Consistent Objectives

Perhaps we could introduce this section by reading from Alice in Wonderland:

“The Cat only grinned when it saw Alice. It looked good- natured, she thought: still it had VERY long claws and a great many teeth, so she felt that it ought to be treated with respect.

`Cheshire Puss,' she began, rather timidly, as she did not at all know whether it would like the name: however, it only grinned a little wider. `Come, it's pleased so far,' thought Alice, and she went on. `Would you tell me, please, which way I ought to go from here?'

`That depends a good deal on where you want to get to,' said the Cat.
`I don't much care where--' said Alice.
`Then it doesn't matter which way you go,' said the Cat.
`--so long as I get SOMEWHERE,' Alice added as an explanation.
`Oh, you're sure to do that,' said the Cat, `if you only walk long enough.' “

Perhaps Lewis Carroll knew a thing or two about the Oil Sands.

Some tailings development projects fail simply because their purpose or objectives were never defined, understood and agreed between the key players at the outset. The risk of this all-too-common occurrence can be greater if the project:

- has multiple sponsors,
- is of long duration (or conversely is being done in a rush),
- is highly technical,
- is controversial or sensitive,
- attempts to modify objectives "on the fly", or
- attaches success to the development of the technology, rather than to the quality of the research.

Clearly the key is to define the purpose of the project, set objectives and sub-objectives. Prioritized objectives may be useful if trade-offs are later required in order to make progress.

2. Scale Up

One of the most daunting challenges encountered in the development of Oil Sands tailings technology, is that of scale up. Many simply avoid the issue and the technology development often fails, as a result.

Complexity

It is not simply because Oil Sands projects are very large (copper and other tailings facilities are of an equivalent size). Oil Sands tailings projects have become very complex in recent years, requiring specialist inputs in fields as diverse as polymer chemistry, computational fluid dynamics, unsaturated soil mechanics, rheology, two-phase flow, environmental science and patent law.

Literature Scan

In a brief review of published papers, and in discussion with two tailings experts, the authors found that little exists in literature as a guide. Those limited references which were found, are listed and described in brief, below:

Laminar And Turbulent Slurry Flow Regimes

Pirouz et al (2005) describe thickened tailings as a viscous fluid with non-Newtonian behaviour in which scale effects can be significant and are a serious issue in the study of the phenomenon. They conclude for tailings beach flow and deposition:

- "Laminar sheet flow of slurry is always associated with deposition.
- Laminar flow cannot persist over long distances since zone settling within the flowing layer will lead to deposition after a finite period of time.
- Turbulence can overcome zone settling. Thus for long distance travel the tailings forms itself into a concentrated turbulent channel.
- It is postulated that the slope of the channel will be steep enough for the generated turbulence to just overcome zone settling.
- The overall beach slope of the stack is therefore determined by the limiting equilibrium channel slope which is required for a particular tailings, to create the required steady state turbulent total transport flow condition in the self-formed channels.
- Based on full-scale, flow-through flume testing, the following observations have been made:
    - The average velocity, the Froude No., the velocity profile with depth, and the density profile within the flow are essentially constant for a given slurry and do not vary with flow rate.
    - The critical channel slope for full transport of the tailings, and possibly the critical Reynolds number, vary with flow rate."

Beach Profile

Morris and Williams (1996) state:
"The relationships between the profiles of full-size and laboratory-model deltas are analysed using the principles of kinematic similarity and a recent theory of delta formation based on engineering hydraulics. Measured profiles of full-size and model coal-mine tailings deltas are compared and the prediction of full-size profiles is discussed. It is shown that the data derived from laboratory-model deltas are insufficient to enable accurate prediction of full-size delta profiles. Initial slope data or other, equivalent data for the full-size delta are also required."

Open Channel Flow Hydraulics

Researchers in water engineering have long employed models to investigate fluvial, estuarine and marine flow behaviour to predict sedimentation, scour and other phenomena, and as tools in hydraulic engineering design.

Two dimensionless parameters often used in open channel formulas, including those applied in the prediction of tailings beach profiles, are the Froude number:

$$(Fr = v / \sqrt{g \cdot h})$$

and Reynolds number:

$$(Re = v \cdot h / \mu)$$

numbers, where v is the mean flow velocity, h is the depth of flow, μ is the kinematic viscosity, and g is the gravitational acceleration.

Flume Tests, Lubrication Theory, Surface Tension And Momentum

Of the references cited in this memo, the work of Fourie and Gawu (2010) and Morris and coworkers best accounts for scalability issues in flume tests.

Fourie (2012) adds: "At the flume scale we have not resolved the potential interference of factors such as surface tension or interference and merging flow from multiple flumes or channels".

In fluid dynamics, lubrication theory describes the flow of fluids (liquids or gases) in a geometry in which one dimension is significantly smaller than the others. Surface tension may then be significant, or even dominant. Issues of wetting and dewetting then arise. (Wikipedia, 2012)

Fourie adds further: "Likewise, I am unaware of any papers dealing specifically with this issue. The issue of beach slope development must be affected by scale up, especially when considering the massively larger volumes (and therefore momentum) involved".

Simms et al (2009) record:

"The flume test visualization studies showed that lubrication theory based equations described the measured flows quite well. This theory may partially explain the discrepancy between overall deposition angles reported in the field and those in the laboratory, as it clearly shows that this angle is scale dependant. Testing at larger scales and with more complex flow geometries is still required to characterize the usefulness of this approach to planning field deposition.

The use of lubrication theory based equations to model the evolving geometry of paste tailings was shown to be effective at the small scale. Tests at a larger scale are required to examine its applicability to the field. Many of the assumptions of lubrication theory may not hold: certainly, at some mines the velocity of the tailings as they exit the pipe may be quite fast and the flow may be turbulent, though the authors believe the flow becomes laminar at some distance from the pipe, and therefore the geometry will be largely controlled by laminar flow. Another assumption that may not hold true is the time independence of rheological properties - it is possible that some tailings may undergo significant settling and dewatering as they flow. This fact may turn out to be advantageous to mine operators, as it may be possible to rely on settling enhanced by desiccation through evaporation and the wicking action of the underlying layer, to increase the yield stress to that required for deposition. Though possible, this kind of deposition would be complex to control."

Pumping And Pipeline Systems

Cooke (2012) indicates: there are challenges with scaling up pump and pipeline transport systems, but I am not aware of any papers that have dealt with this issue. It has been on my mind to write a paper on the limitations of slurry pipeline transport but I have not yet managed to find the time.

Scale Up Challenges In Oil Sands Technology Development

In describing the potential for Retort-Based Extraction (a possible non-water based extraction method in which heat is used to vaporize any water and generate pyrolysis of the bitumen, the waste coke can be used as a fuel source for future extraction and the remaining tailings are free of bitumen and other hydrocarbons) as an alternative tailings technology, the Alberta Innovates Oil Sands Tailings Roadmap report (2012) laments that:

"Scale-up. This technology is still in the development phase. To date, the largest pilot can only process 1/30th of the amount of ore that is normally produced in one train using current extraction techniques.”

And in reference to chemical alternative tailings treatments:

“In pursuing the development of chemical amendments to Oil Sands tailings, the following challenges should be considered:

- High cost to evaluate chemical amendments at pilot scale.
- Uncertainty surrounding technical aspects of the chemical amendments and a lack of understanding of how they work.
- Performance can be highly variable due to changes in input chemistry which could render results very lease specific.
- Scale up - Quantity and cost of chemical required for commercial scale operations can be considerable.”

The report does offer some solutions:

“Develop a series of standard trials and evaluation tools to evaluate the potential of chemical amendments at first the lab scale then the pilot scale. The evaluation criteria should be both short and long term focused.”

Empirical Similitude Method In Other Engineering Applications

In a recent Ph.D. thesis, Tadepalli (2009) describes the difficulties encountered in many new fields of engineering with modeling, scale-up, or similitude. In describing the development of an Empirical Similitude Method (ESM), Tadepalli records:
“Analyzing the behavior of a system with a network of components requires multifaceted approximations to reasonably simplify the modeling effort. More often than not, a dominant phenomenon is isolated that captures a major segment of the system response without compromising the integrity of the governing dynamics of the associated physical process. Such a simplification cannot be achieved with considerable ease and requires numerous approximations and assumptions.”

Although describing the difficulties of modeling engineering components in a typical manufacturing setting, Tadepalli goes on to list a number of factors which would give rise to the need to use a more abstract model to model non-linear behaviour (in a variety of scientific research fields), including:

1. Product/Model has distortions
2. Product has material non-linear responses
3. Product has multiple materials
4. Product has variable inputs and, initial and boundary conditions
5. Product has space and time varying properties
6. Product to model scaling is not uniform or constant
7. Product behavior has no or complex governing equation
8. Product has no realizable simple model;
9. Parameters have no realizable experiment
10. Experimentation has unrealistic time scale

Computational Fluid Dynamics (CFD)

One of newer and more promising fields of tailings research is that of Computational Fluid Dynamics, or CFD. Even in this field and perhaps starkly so, the issue of scale-up is critical. Strategic dependence on the use of Camp number (itself dependent on pipe dimensions) is reflective of this realization.

Potential Remedies for the Challenge of Scale Up

How should these many challenges of scale up be addressed? (Most practitioners appear to be agreed that this is becoming a critical issue). However, the solution is not simple and answers are not readily to hand. The authors suggest the following approach(es) as a starting point:

1. Review published literature for evidence of scale dependence for the parameter(s) under consideration.
2. Recognize which parameters are scale dependent, and which ones are not (for example, flume test width, momentum and surface tension effects appear to be key in considerations of scale).
3. Isolate the key parameters which are most dependent on scale, researching their influence on project parameters one at a time.
4. Attempt to describe in fundamental scientific terms or at the very least in empirical terms, the nature of the influence of each key scale dependent parameter.
5. Establish the minimum scale of pilot test required to provide meaningful results, by

a combination of research, fundamental analysis, modeling and lab scale testing.

6. Explore the potential for increased understanding and characterization, by successive modeling at gradually increasing scales, while tracking the change between stages from:
   a. Lab (characterization) testing.
   b. Lab (batch and comparative) testing.
   c. Individual pilot testing.
   d. Batches of pilot tests (in sequence or in parallel).
   e. Prototype testing.
7. Calculate, consider and compare the relative costs, risks and benefits of field testing at increasingly large scales, in order to establish the most promising scale of test, for the parameters under investigation.

In implementing the steps above, the following aspects should also be borne in mind:

Step 3 (above) may imply an uncoupled approach. In reality, many parameters are not completely independent of one another and isolation of parameters from one another may be difficult. It may then be useful to reconsider the choice of parameters by selecting only those which are truly independent from one another. In addition, some parameter selection occurs as early as Step 1, and may be influenced by the choice of approach or test method.

The modelling of open channel flow has suggested that in situations where multiple scale dependant variables are present, there is no simple alternative but to resort to larger prototypical models.

The project manager should resist the temptation to over-accelerate development, thereby missing development steps (by leaving out essential steps in the sequence – such as researching the fundamentals, lab testing, pilot testing, etc.) or by attempting to perform steps in parallel, rather than sequentially.

Technical subject matter experts should remain involved in all successive development stages, in order to ensure that scale up effects are fully considered and addressed during the project.

The use of an analogue fluid or material may be considered, (such as is already used in the modelling of fluid flow and mixing in pipelines, and in association with CFD and elsewhere).

3. Skilled Leadership

Responsibility for tailings related decision-making is now vested at most senior levels, as the implications for Oil Sands operators are substantial. This has placed a premium on executives to be familiar with tailings matters and to be well advised by specialist tailings engineers at first hand.

Tailings projects also demand the focused attention and commitment of senior engineering leadership from the consultant, with rigorous scrutiny at the Oil Sands Geotechnical Review Board level.

Close teamwork and a relationship of trust between client and consultant are fundamental. This comes into sharp focus while pro-actively responding to and managing the inevitable changes which occur in projects of this nature, sometimes arising out of a change in the big picture. This is amplified in section 5 below.

4. Continuity, Skills And Teamwork

Working in a young and evolving industry such as the Oil Sands is both exciting and challenging at the same time. It offers tremendous opportunities for learning; however it has its own unique ongoing challenges that require input from experienced specialists with a thorough working knowledge of Oil Sands tailings and geotechnical engineering and who know what will work and what will not, without wasting time and resources on repeat work and trials.

The challenge in the Oil Sands is the limited number of those specialists who are available to promptly respond to the demands of daily mining operations. Also, it is commonplace to change the responsibilities and roles of those specialists to accommodate the needs of different areas of operation. These challenges can be overwhelming, allowing no time for creative thinking and innovative solutions to be developed, and which disrupt communications and project continuity.

These challenges can be lessened through the efficient use of available resources, by assembling teams of professionals with the necessary skill sets to assist in responding to the tailings project demands, and who can call in specialist expertise when needed in a timely and effective manner. The specialist needs to provide strategic guidance to team members, to brief them in detail, to measure their progress and to work closely with them to ensure timely completion of agreed deliverables within budget.

This will provide the Oil Sand industry with much needed and well-trained teams of dedicated professionals who have in-depth experience of tailings projects, who promote continuity between projects and who maintain effective two-way communication within each project team, as well as with outside specialists.

5. Big Picture Thinking

The unique challenges posed in Oil Sands tailings in recent times have demanded with increasing clamour the need for big picture thinking. The turning point was perhaps the publication by the ERCB on February 3, 2009 of Directive 074, and more recent promising initiatives by industry and government to develop common objectives for the management and reclamation of Oil Sands tailings.

There is continued need to consider big picture aspects during tailings engineering and development projects, on account of ongoing media, public and regulatory scrutiny and interest. This was addressed in a paper two years ago at this forum, Boswell (2010):

This paper suggests the view that the Big Picture is managing the interface between:

- Tailings
- The Environment
- The Public

Big Picture Thinking encompasses five components:

- Technical
- Operational
- Environmental
- Legal
- Public

This subject is expanded upon in the paper referenced above, as well as in the references cited in the paper at the time.

6. Pro-Active Independent Research

There is an abundance of learning opportunities which are yet to be captured, in order to enhance the practice of Oil Sands tailings management. Adopting a pro-active research philosophy will allow the Oil Sands industry to shape its own future and engineer its own opportunities, instead of simply reacting on the spur of the moment to challenges as they arise.

To promote healthy growth in Oil Sands tailings management, it is insufficient to keep using the same techniques and practices without recognizing special opportunities when they come along. It is important to then make conscious decisions based on a sound understanding of the engineering principles of tailings to create opportunities, and in so doing encourage step-change advances in the industry.

Pro-active research requires a clear understanding of the industry operational, financial and regulatory drivers. It should ask the questions: Where is the industry going? Where should it go next? How will it get there? Who will take it there, and in what steps? The research clearly requires input and feedback from the operators, regulators and industry experts.

Creativity and the ability to generate alternative responses to future challenges that will face industry are key elements of pro-active research. These elements require the input of independent forward-thinking individuals who can mentally generate and evaluate different options, and have the freedom to explore unique solutions without being restricted or forced to conform to corporate boundaries or prevailing social norms and public expectations. Academic and industry research bodies need to be engaged in the process to examine and verify the scientific principles of the proposed solutions.

7. Scientific Critique And Rigorous Review

A fundamental attribute of high quality research is the capacity to invite and address the scrutiny of scientific critique and peer review. This applies in the Oil Sands no less.

At the same time however, the hard-nosed Oil Sands tailings technology development champion is aware that in order for continued progress to be made, a little deafness to critics may be required for project success.

The challenge is to be able to strike a balance and in so doing to:

- Differentiate valuable technical critique from unfounded criticism.
- Identify fatal flaws in technology development at the earliest possible juncture, and to address them as a priority.
- Establish a project reputation which maintains a readiness to receive critique.

A number of project structures and mechanisms may be useful in allowing a project to receive critique and thereby improve engineering robustness and the chances of success:

- Risk awareness, and a constant vigilance for potential fatal flaws.
- Anticipation and planning in advance for contingencies and risks.
- Regular use of risk assessment techniques, such as a Probable Failure Modes Analysis (PFMA), Value Engineering, Quality Circles or SHERQ – there are many examples; a project structure and schedule which allows for these inputs in a timely manner.
- All work produced by the team to be subject to a rigorous internal review, before being released.
- A close working understanding between consulting Team and Senior Geotechnical Reviewers, and knowledge of the requirements and function of the Review team.
- A "publish-or-perish" commitment to sharing of research findings in journals of high standing, specifically inviting comment and critique.

8. Intellectual Property And Contracts

Recent technology development in the Oil Sands has struggled at times in dealing with the demands of intellectual property rights, and delays occasioned by the requirements of contract negotiations.

There are remedies however:

- Allow sufficient lead time at the outset of the project for the development, negotiation and agreement of contracts dealing with necessary clauses for intellectual property, patents, copyright, non-disclosure and contractual remedies.
- Be flexible (regarding information transfer, deliverables, schedule, resourcing) in addressing difficulties with intellectual property.
- Make use of sufficiently knowledgeable specialists who are able to draw from wide experience.
- Build understanding and knowledge of the particular demands of Oil Sands technology development, when drafting contracts; recognize the need for skills in contract negotiation; bridge the gap between engineers, attorneys and accountants, in setting up project structures and communication lines.
- Recognize the circumstances under which protection of intellectual property is required and those instances where it is not; avoid unnecessarily draconian clauses (for example, indefinite protection of rights in perpetuity).
- Maximize the opportunities to improve co-operation within the Oil Sands industry presented by the formation of the Oil Sands Tailings Consortium (OSTC), and more recently Canada's Oil Sands Innovation Alliance (COSIA). COSIA is an alliance of Oil Sands producers focused on accelerating the pace of improvement in environmental performance in Canada's Oil Sands through collaborative action and innovation.
- Proactive measures to encourage and involve vendors and suppliers especially those from outside of the Oil Sands.

9. Planning, Control And Discipline

There are a number of conventional project management skills and disciplines, which when systematically applied, are capable of delivering even difficult projects on time and within budget. They include:

1. Technical accuracy. Attention to detail and technical accuracy in analysis and design; use of skilled engineers experienced in

state-of-the-art analysis techniques and software.
2. Control. Disciplined project management and control of resources, schedule and budget in a spirit of no surprises.
3. Documentation: Careful record keeping and documentation of the design process and results as they become available; a philosophy of recording key events and decisions.
4. Reporting. An ongoing discipline of summarizing and updating key geotechnical information, results and recommendations into a comprehensive set of Design Memos, long before the Final Report deadline.
5. Communication and record-keeping. Keeping the whole team aware of progress, and the changing needs of the project; tracking the status of documents, ensuring that the most current information is continuously available.

CONCLUSIONS

The authors have presented nine challenges encountered in the project management of Oil Sands tailings technology development projects. For each challenge they offer a few insights and solutions towards addressing these challenges:

1. Clear, consistent objectives.
2. Scale up.
3. Skilled leadership.
4. Continuity, skills and teamwork.
5. Big Picture Thinking.
6. Pro-active, independent research.
7. Scientific critique and rigorous review.
8. Intellectual property and contracts.
9. Planning, control and discipline.

In addressing the particularly vexing challenge of scale up (in Section 2), the authors have proposed the following seven step solution:

1. Review published literature for evidence of scale dependence for the parameter(s) under consideration.
2. Recognize which parameters are scale dependent, and which ones are not
3. Isolate the key parameters which are most dependent on scale, researching their influence on project parameters one at a time.
4. Attempt to describe in fundamental scientific terms or at the very least in empirical terms, the nature of the influence of each key scale dependent parameter.
5. Establish the minimum scale of pilot test required to provide meaningful results, by a combination of research, fundamental analysis, modeling and lab scale testing.
6. Explore the potential for increased understanding and characterization, by successive modelling at gradually increasing scales, while tracking the change between stages
7. Calculate, consider and compare the relative costs, risks and benefits of field testing at increasingly large scales, in order to establish the most promising scale of test, for the parameters under investigation.

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SIMULATION OF TAILINGS MANAGEMENT OPTIONS

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ABSTRACT

Oil sand tailings management practices result in continual accumulation of fine tailings or mature fine tailings (MFT). This has prompted the Energy Resource Conservation Board (ERCB) to regulate fluid fine tailings through performance criterion (Directive 074). The aim of the directive is to reduce fluid tailings accumulation and create trafficable surfaces for early progressive reclamation. To meet the new Directive 074, operators are looking to alternative tailings management options and technologies to reduce their inventory of fluid fine tailings and expedite the reclamation process.

This research aims to assist in the assessment of tailings management options through the development of a dynamic simulation model. The model will simulate the tailings system behaviour and complex relationships from production to the onset of reclamation. The initial version of the model investigated a simple metal mine tailings management scenario. The complexity and scale of the proposed tailings management plan was deemed suitable for the development and calibration stages of the simulation model structure and components. The model was then expanded from the current simple metal mine scenario to reflect the management scheme and material flows at oil sands operations. Extensions for modeling an oil sands tailing operation are discussed.

INTRODUCTION

In Alberta's Oil Sands industry, tailings management practices has resulted in continual accumulation of fine tailings. Refer to Sobkowicz and Morgenstern (2009), Beier at al. (2009a), and Hyndman and Sobkowicz (2010) for a summary of historic and current tailings management strategies that have contributed to this accumulation. Research and development over the past 3 decades have generated technical advances in mining, material handling and bitumen extraction. However, finding practical methods to control and reduce the fluid fine tailings build up has been an ongoing industry challenge.

The fine tailings accumulation prompted the Energy Resource Conservation Board (ERCB) to regulate fluid fine tailings through performance criterion. In February 2009, the ERCB issued Directive 074: Tailings Performance Criteria and Requirements for Oil Sands Mining Schemes. The aim of the directive is to reduce fluid tailings accumulation and create trafficable surfaces for progressive reclamation (Houlihan and Mian, 2008). To meet the new Directive 074, operators are looking to alternative tailings management options and technologies to reduce their inventory of fluid fine tailings and expedite the reclamation process.

Objective

This research aims to assist in the assessment of tailings management options through the development of a dynamic simulation model. The model will allow the user to simulate the tailings system over time, demonstrate various outcomes by alternating management practices, and conduct sensitivity analyses in an effort to find the best course of action (Beier at al. 2008).

The model will simulate the tailings system behaviour and mass transfers from production to the onset of reclamation. This will allow the simulation model to be used as a "what-if" tool to experiment with various operating strategies or design alternatives to support technology assessment, scenario-analysis, fore-sighting and mine planning. Sensitivity/uncertainty analyses can also be used to strategically guide further research and resource expenditure. It could be used to generate a short list of options that would be forwarded for further detailed design and research. There currently are no publically accessible simulation models of this type available for evaluating the management options quickly and efficiently.

This paper will outline the model development and validation with a metal mine tailings management scenario. The complexity and scale of this tailings management plan is suitable for the development of the simulation model structure and components. Fine tuning of the initial model was required to reflect the management scheme and material flows at oil sands operations. The paper will also discuss these extensions for modeling an oil sands tailing operation.

DEVELOPMENT AND MODEL STRUCTURE

A dynamic systems model was developed for the simulation and evaluation of tailings management systems (TMS). An object orientated, systems dynamic modeling software called GOLDSIM was used as the "simulation engine". The systems model essentially tracks the stocks and flows of mass (solids [mineral including both fine and coarse], water, and chemicals) throughout the TMS. A suite of sub-models were used to represent individual components such as the extraction plant, tailings treatment, impoundment and the environment. Critical processes (such as consolidation, dewatering, seepage, etc) within each component will dictate mass transfer between components. (Beier et al 2009b).

The Goldsim modeling platform is essentially a "visual spreadsheet" (Figure 1). The user can visually and explicitly create and manipulate data, equations and relationships (Kossik and Miller, 2004). The simulation model can be constructed from process-based, empirical or even qualitative formulations based on a tentative relationship between two parameters. Therefore, it allows flexible inputs, outputs, time stepping, and coupling of processes (Wickham et al. 2004).

A spreadsheet will be used as the data entry/interface for all model inputs such as site properties, tailings properties, mining and extraction rates, environmental data and pertinent management decision variables (i.e. constraints on the system). The user will have the option to utilize built in functions and sub-models or implement their own models. Implementation of user specific models/data would be completed either in the data input spreadsheet or the Goldsim model directly (Beier et al., 2009b).

Performance Measures

The intent of the simulation modeling is not to mimic or predict the exact behaviour but rather to identify the properties and processes (i.e. consolidation, solids content, treatment options) that are most significant. These significant processes would have the greatest impact on the overall success of the tailings management system and would be the target of further research, or more detailed design (Beier et al. 2009b).

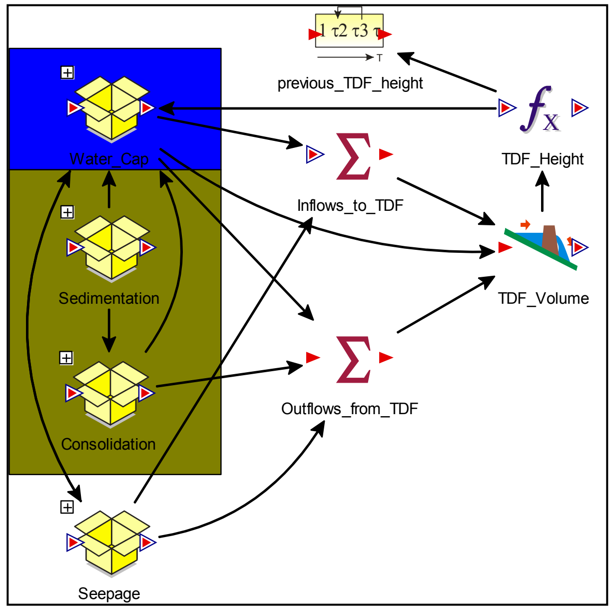

Figure 1. Screen shot of the GoldSim Model.

Performance measures have been defined to assess the management strategies in the simulation model and are outlined below (Figure 2):

- What is the available storage volume (in-pit, impoundment, construction material)?
- What is the required impoundment storage volume (for both solids and water)?
- What is the available recycle water volume and quality?
- Strength gain trajectories within the deposit.
- What is the seepage rate to the environment and its quality?
- Sensitivity/flexibility of disposal option?
- Impacts on extraction?
- Interim model results such as flow rates, solids/fines content, chemistry can be used as input for transport analysis (pipeline/pump)

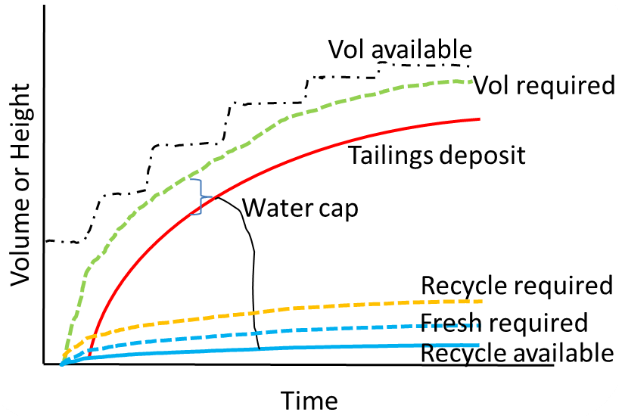

Figure 2. Performance measures obtained from the model results.

TAILINGS MANAGEMENT OPTIONS

There are several options available to the mine operator for management of the various oil sand tailings streams that may lead to a reclaimable ("trafficable") deposit as illustrated in Figure 3 (Beier et al., 2009a). This Illustration represents a majority of the potential flow paths of the dry and wet waste streams for a generic mine scenario. Site specific conditions (geology, ore quality, topography, etc.) may preclude certain technologies from being applied at a particular mine site.

Segregation of the extraction tailings either by cyclones or during deposition onto beaches has contributed to the build-up of clay-dominant fluid fine tailings (MFT). The segregated fine tailings and coarse sand may be managed separately or re-combined. There are three general methods that can be utilized to incorporate fine tailings into a closure landscape (Hyndman and Sobkowicz, 2010).

- The fines can be sequestered into the coarse tailings matrix (consolidated/composite tailings [CT] and non segregating tailings [NST]),
- placed under a water cap (in-pit lake) or
- dewatered separately creating a cohesive, silty-clay deposit using various chemical, physical and environmental dewatering techniques.

Simulation Model Options

The first stage of modeling was used to work out the bugs of the simulation model. Data from a metal mine tailings plan was utilized and provides and analogue for an external oil sand tailings facility with deposition of MFT or TT (30-40 % wt) under a water cap with dyke construction using coarse tailings and/or OB. This represents one of the fine tailings management options and will be discussed in the following section.

The model is currently being expanded to cover the other fine tailings management scenarios. A base case for an oil sands operation was developed from the Syncrude Aurora North tailings management plan to provide an additional calibration step. The data utilized in the model was extracted from publically available D074 submissions and related public domain sources. The extensions will be described in the following sections and reflect the management scheme and material flows at oil sands operations. They include sub-models for large strain consolidation and a dewatering treatment model.

Due to the inherent site specific issues among the mine sites, a generic oil sands mine plan based on data from all the D074 plans was generated. Since the model is intended to be a comparative model i.e simulate and compare "what if" scenarios a generic mine plan was deemed sufficient.

To demonstrate the models ability to "virtually test" a new dewatering technology, a filtration process consisting of dewatering whole tailings (cross flow filtration) will be simulated (Beier et al. 2008). The model will be used to identify sensitivities of performance measures to design parameters as well as variability (uncertainty) in operating parameters.

METAL MINE SCENARIO

For the initial calibration of the simulation model, a metal mine tailings management scenario was modeled. The data used in the calibration was taken from an engineering feasibility study. The complexity and scale of the proposed tailings management plan was deemed suitable for the development and calibration stages of the simulation model structure and components.

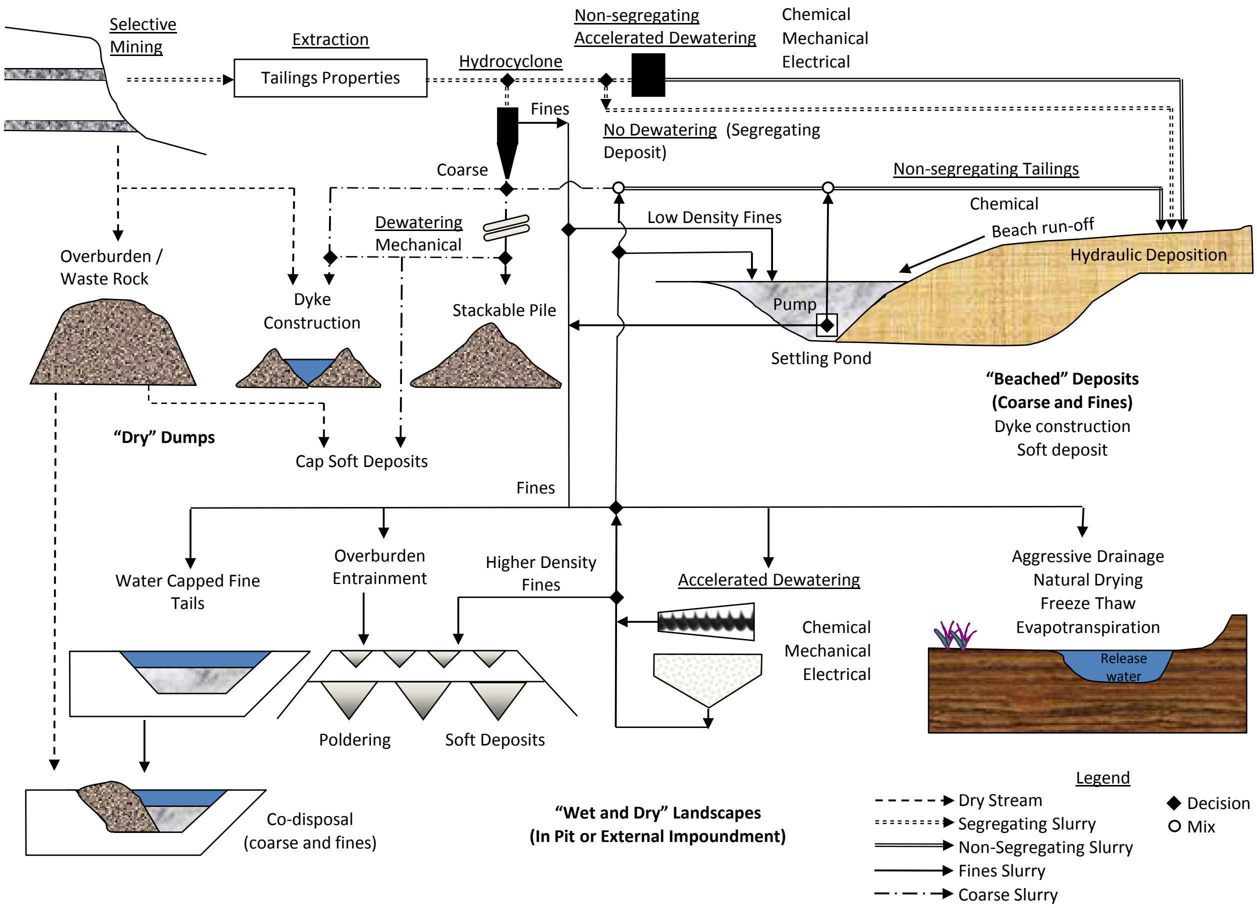

Figure 3. Tailings management options (Modified from Beier et al. 2009a).

The tailings plan at the metal mine operation entailed rendering the tailings non-segregating with mechanical thickeners prior to deposition. The tailings were then deposited subaqueously and stored under a water cap. Overburden was utilized to construct the tailings impoundment structure.

Mining operations will process 35 million tonnes over the 29 year life of the mine. Extraction tailings consist of non-plastic sandy silt with 65 % geotechnical fines content (< 75 micron). These tailings will be discharged into the impoundment structure at 28% by weight where they will rapidly dewater thus negating the need for large strain consolidation modeling. A minimum 3 m water cap will be maintained above the tailings and will be used for continuous recycle to the extraction process.

The Goldsim model was run utilizing the metal mine operational data parameters and then compared to the actual mine data set. The performance measures of interest can be found in Figures 4 and 5. The solid lines represent the Goldsim model data and the dashed line represent the mine site data sets. As can be seen from the figure, at the end of mine life, the mass balance for the model reflects the mine site data set sufficiently. The modeled total impoundment volume (TDF) deviated from the mine data set by up to +/- 5 % over the life of the mine. Rainfall and runoff collected was lower than the mine data set. These differences can be attributed to the method of how the values were calculated. The stage curves used in the Goldsim model were extracted from a curve and not a detailed dataset. This will impact the calculated volume, height, and associated surface area of the impoundment which in turn will impact several other performance measures.

Additionally, GoldSim computes instantaneous rates at each time step (e.g., an instantaneous flow rate). The data used for comparison was rate). The data used for comparison was generated from a spreadsheet model in the feasibility study. Spreadsheets do not actually deal in rates; they compute changes in a quantity (e.g., a volume) over an interval (i.e., the time step). A change in a volume divided by a time interval does not represent an instantaneous rate; it represents an average rate over the interval. Comparing an instantaneous rate at a given time (computed by GoldSim) to an average rate over the time period between two points in time (computed by a spreadsheet) is likely to yield different results. The ultimate accounting will be the same, however instantaneous comparisons may be different.

EXTENSIONS TO OIL SANDS

Due to the clay dominated nature of the oil sands fine tailings, estimating the time of dewatering is a critical parameter. Therefore, addition of self-weight consolidation of the fine tailings to the simulation model is required. The GoldSim modeling platform has the ability to interact with third party software, therefore the large strain consolidation program FSConsol was utilized rather than developing new code. The flow sheet in Figure 6 outlines the process by which GoldSim was linked with FSConsol.

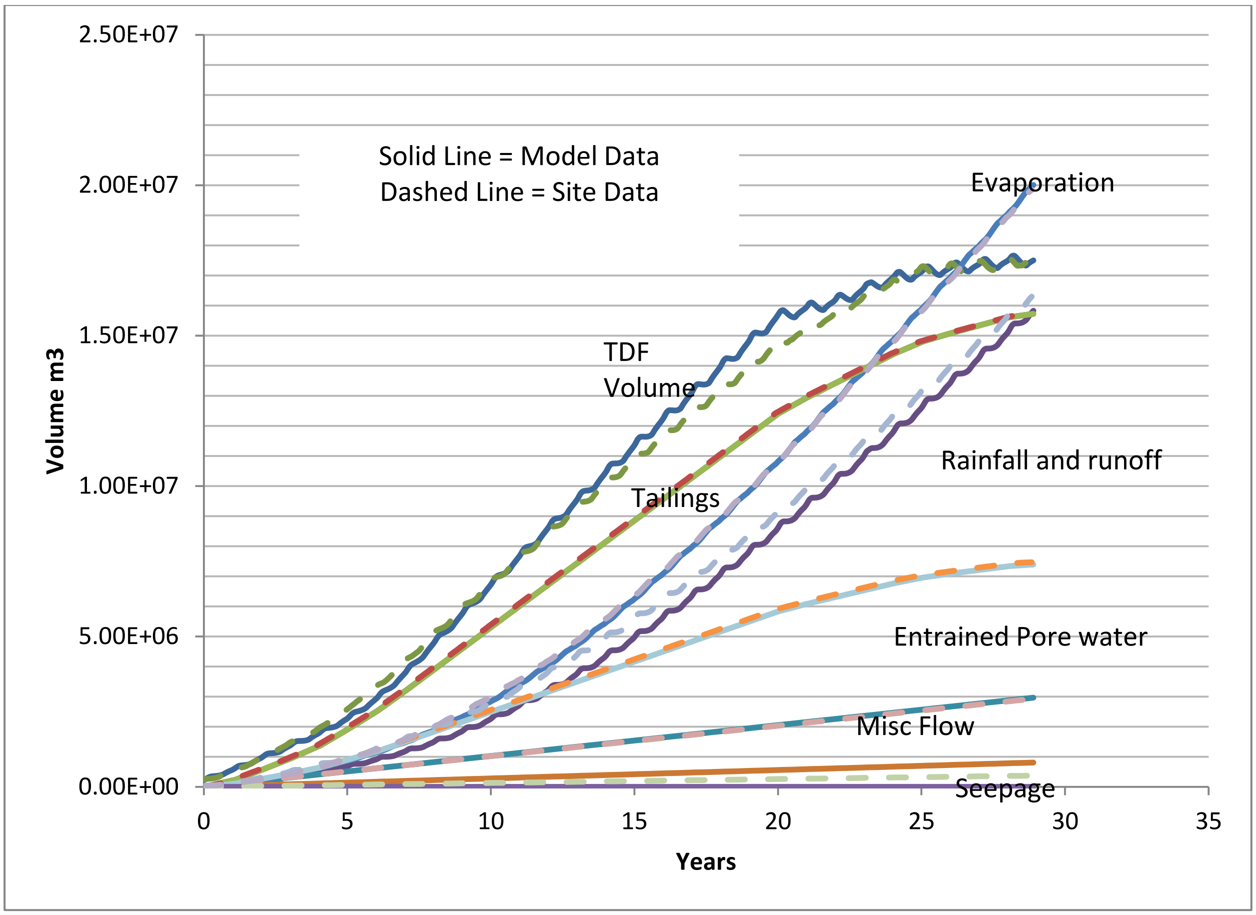

Figure 4. Comparison of simulation model data with the mine site data.

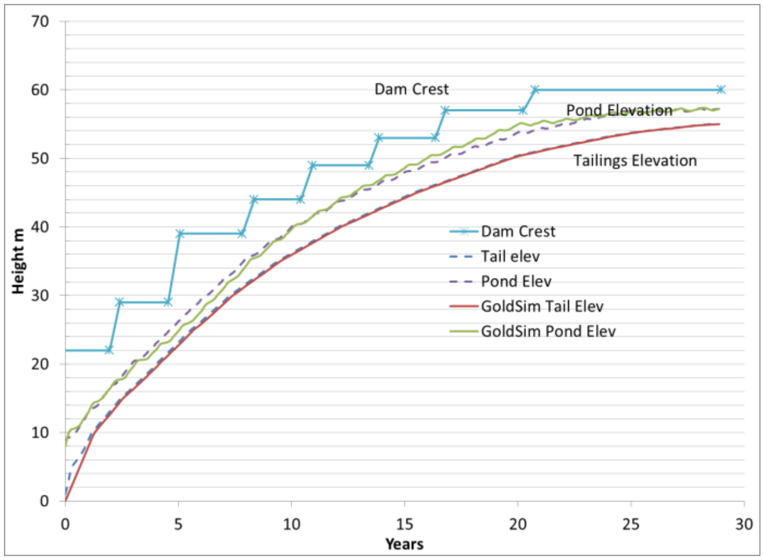

Figure 5. The dyke elevation calibration data.

Initially, GoldSim calculates the tailings production rate based on extraction efficiency which can be influenced by feedback on the system. Material properties can also be influenced by feedback in the system therefore they can be updated at each time step if required. The GoldSim model then logically performs its required calculations until the height of the deposit and/or void ratio of deposit is required for the current time step. A macro is triggered in the data spreadsheet which creates the necessary input based on user input, GoldSim data, and previous deposit information. FSConsol runs a simulation for one GoldSim time step and then the spreadsheet macro returns the necessary data back to GoldSim. This process is repeated until the simulation is complete. This approached allows feedback in the system to modify the tailings properties based on extraction feedback, or other changes in the system.

Validation of FSConsol Linkage

To ensure errors were not introduced each time the FSConsol model was run (i.e. once per time step) a validation run was completed (Figure 7). Tailings material properties and typical loading rates from the metal mine scenario as well as an oil sand thickened tailings stream were utilized to conduct the validation run. An FSConsol simulation was run for 5 years at 30 day time steps for each deposit type. Then a simulation was run using GoldSim linked with FSConsol using the same input parameters. This linkage resulted in 60 instances of FSConsol runs. In both deposits, the height of the deposit was nearly identical (<5 mm difference) for both simulation methods. The void ratios of the deposits were also nearly identical in the deposit (<0.02 difference in the upper 2 m). Running several instances of FSConsol through the GoldSim linkage did not introduce significant errors to the overall simulation.

Dewatering process

The simulation model must also be able to "virtually test" a new dewatering technology. Therefore a sub-model representing a dewatering step must be coded into the simulation model. A dewatering process sub-model will be implemented between the extraction process and deposition, with a similar function used after potential dredging operations prior to re-deposition.

The sub model will be developed using a Conditional user defined function (UDF) in GoldSim. This type of sub-model is extremely flexible and will allow for a simple or a complex numerical model of the dewatering process. The user can code the model directly into GoldSim or utilize the data input spreadsheet.

The GoldSim sub-model is depicted in Figure 8. The user will indicated on the input spreadsheet where dewatering will occur (after extraction or after dredging) and what dewatering numerical model to use. This will trigger the conditional UDF in GoldSim to be included during the simulation. Operational data can be defined by the user and calculated with GoldSim.

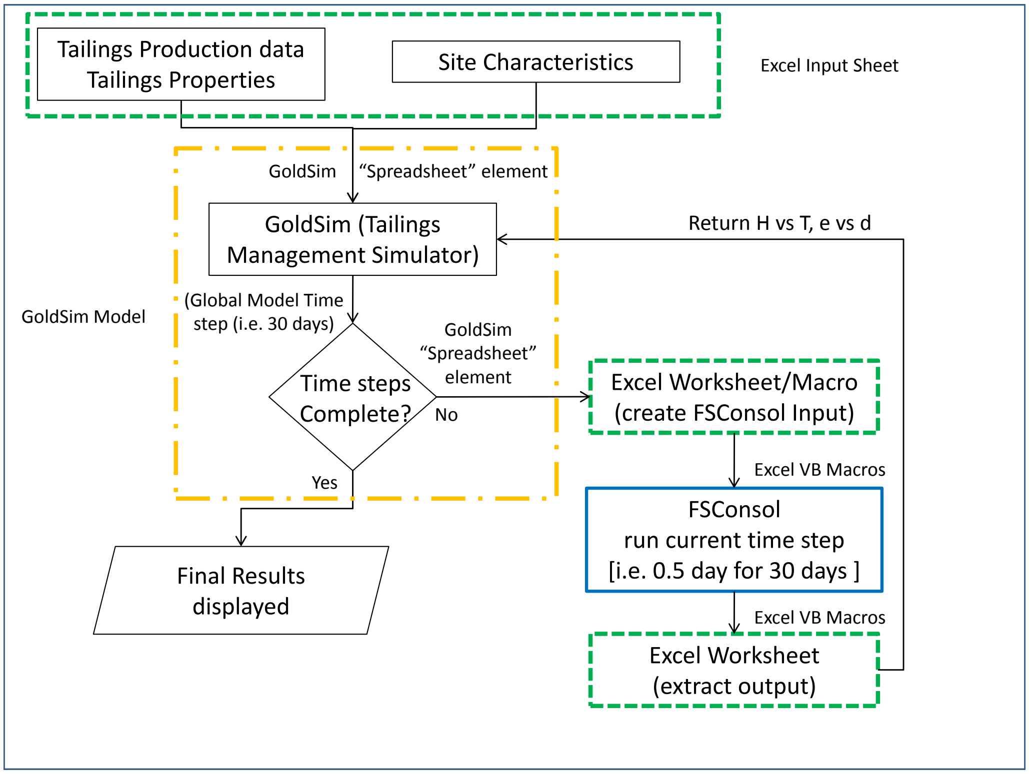

Figure 6. Flow chart of FSConsol linkage with Goldsim logic.

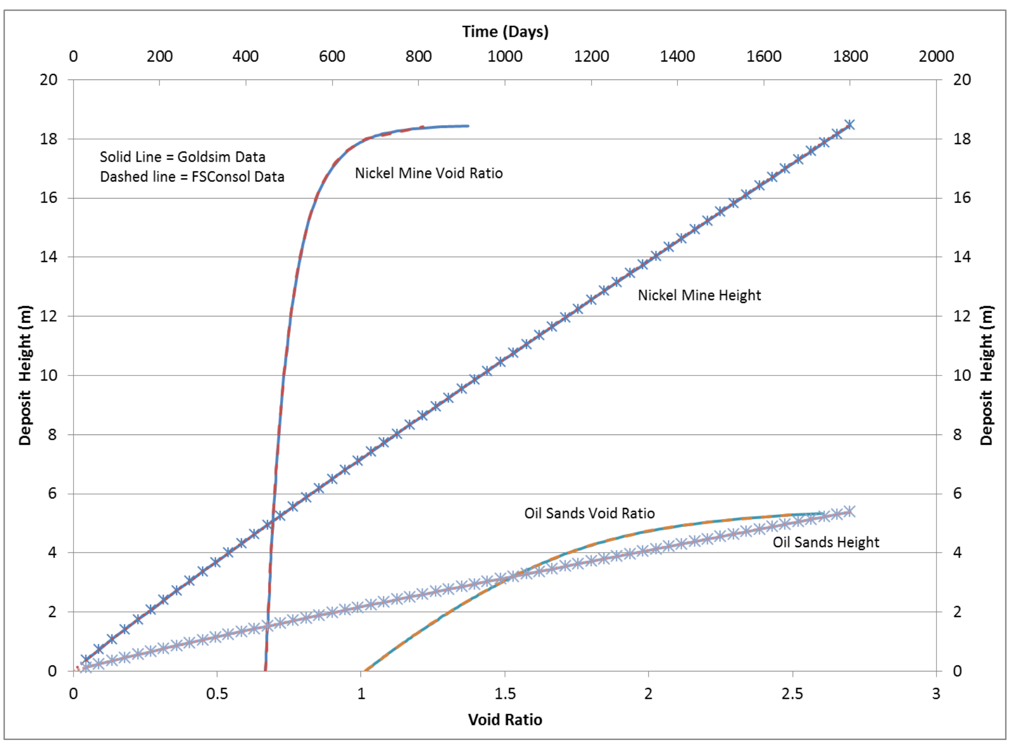

Figure 7. Validation of FSConsol linkage with Goldsim.

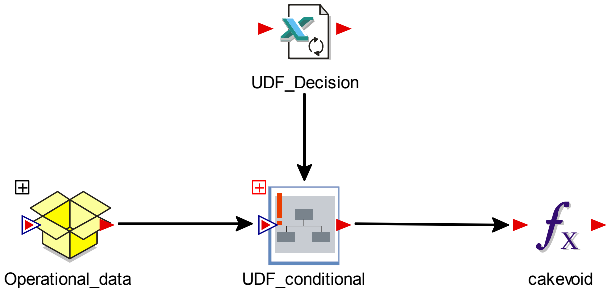

Figure 8. Dewatering sub-model process.

To demonstrate the dewatering sub-model an example simulation using published data on filtration of oil sands tailings was run and shown in Figure 9 (Xu et al., 2008). This data was only used to demonstrate the application of the dewatering sub-model and not to promote the filtration process identified in Xu et al. (2008). The sub-model returns the necessary floc dosage required to maintain a consistent filtration cake with varying feed properties. The model could also return the cake mass fraction based on constant floc dosage.

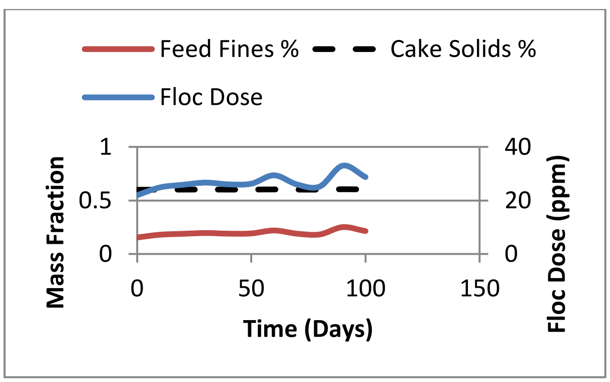

Figure 9. Output from dewatering sub-model using Xu et al. (2008) data.

CONCLUSIONS

Tailings management is a complex issue, especially for large operations such as the oil sands. A dynamic simulation model was developed by combining elements of the tailings life cycle into a single simulation to assist with assessing various tailings management strategies. The model was validated against a metal mine tailings scenario. The complexity and scale of this tailings management plan was suitable for the development of the simulation model structure and components. Extensions to the metal mine simulation were required to reflect the management scheme and material flows at oil sands operations. One extension, a large strain consolidation sub model, was also validated. A sub-model representing a dewatering step was also coded into the simulation model. The sub model was developed using a conditional user defined function. This type of sub-model is extremely flexible and allows the user to simulate simple to complex numerical models of the dewatering process.

ACKNOWLEDGEMENTS

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COMPARISON OF PORE WATER STORAGE REQUIREMENTS OF SEVERAL TAILINGS MANAGEMENT SYSTEMS

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ABSTRACT

Directive 074 of the Energy Resources Conservation Board (ERCB) has intensified the oil sand industry's attention on improved tailings technologies that provide for trafficable surfaces over tailings storage areas. The amount of water being used by the oil sands industry has also resulted in restrictive regulations on water withdrawals from the Athabasca River. These two environmental issues are related since tailings pore water represents the largest consumptive use of imported water in the oil sands industry. Most of the water that is imported by the oil sands industry reports to tailings pore water that is stored in tailings deposits. Interestingly, the quantity of tailings pore water volumes differs among the various tailings technologies being considered by the oil sands industry. Permanent water consumption by tailings pore water is therefore a relevant criterion for comparing tailings technologies. This paper provides a comparison of pore water volumes associated with a selection of leading tailings technologies being considered.

INTRODUCTION

This paper will show that consumptive water use by the oil sands industry is strongly affected by tailings management and tailings deposition systems. Efficiency of water usage by the oil sands industry depends significantly on water used for tailings storage. A low density tailings deposit requires more water than a high density tailings deposit because without special measures to promote drainage, voids of the tailings deposit are almost entirely filled with water in a water-based ore extraction process that is used by all existing and planned mine operators in the oil sands industry.

Water is extremely important to the oil sands industry. Water is used to extract the bitumen from the oil sands ore, convey the ore by pipeline, convey tailings by slurry pipeline and operate cooling towers. The large majority of water for oil sands mine operations is obtained from the Athabasca River with large pump stations ranging in peak capacity from 2 to 5 m^3/s.

Water consumption by the oil sands industry, is also important to regulators and stakeholders because of potential environmental effects associated with large withdrawals of water from the Athabasca River. This concern by government regulators is reflected by recent imposition of water withdrawal constraints to preserve flow rates in the Athabasca River. The overall water withdrawals from the Athabasca River currently form a very small portion of the total river flow, however, regulators and some other stakeholders are resolved to keep it that way.

Directive 074 by the ERCB was enacted in 2009 to provide for appropriate reclamation and future land use of tailings disposal areas. It has resulted in the current focus on adopting tailings management systems that offer measurable strength gain during mine operation to give regulators and stakeholders' confidence that the tailings deposition areas will become trafficable and ultimately enable land reclamation and planting of forests. The resulting tailings management systems being considered by the oil sands industry are associated with deposited tailings material densities that are much higher than the tailings materials resulting from historic tailings deposition practices in the oil sands industry. Therefore, the new tailings deposition technologies are expected to enable a significant reduction in permanent water utilization for pore water storage. Considering the value of water and the importance of minimizing water consumption, it seems appropriate that the range of tailings management schemes be compared in terms of water consumption.

WATER BALANCE OF A TYPICAL OIL SANDS MINE

This section presents some key information to help understand the overall use of water by the oil sands industry and the prominence of pore water in the mine water balance. Much of the assessment in this section on mine site water balance was conducted during various projects commissioned by individual oil sands firms as well as a major oil sands industry water use study that was commissioned by the Oil Sands Developers Group (OSDG) and the Canadian Association of Petroleum Producers (CAPP) resulting in a report titled Water Conservation, Efficiency and Productivity Plan – Upstream Oil and Gas Sector (CAPP and OSDG, March 2, 2009). Relevant information was also provided by Alberta Environment. Additional details of this topic are given in another paper entitled Oil Sands Mining Water Use and Management (Bender, Long and Fitch, 2010).

The four existing operating companies with bitumen mining operations in northern Alberta utilize the Athabasca River as the primary water source for makeup water. The required quantity of makeup water depends on factors such as climate conditions, ore characteristics, mine operations, stage of mining and tailings management systems. The rates of water diversions from the Athabasca River by individual mine operators vary greatly over time depending mainly on the quantity of water needed for pond storage, ore processing rate, size of mine footprint that is related to the quantity of natural surface runoff and the relative quantity of denser (mature) tailings deposits. Water diversions are relatively high during the early years of mining when the mine footprint is relatively small (relatively small surface water inflows), the required water inventories have not been developed, and the average density of deposited tailings is relatively low. Water diversions from the river typically reduce as the mine matures, due to filling of required water inventories, the expanded mine footprint that results in the collection of greater water volumes from on-site drainage of surface water, and the consolidation of tailings materials. Eventually, the proportion of water volume derived from on-site drainage collection may equal the water diversions from the river. The overall trend for river diversions for mine water supply is shown on Figure 1 for two tailings management systems.

River diversions for oil sands mine operators are subject to various regulations by the Alberta Government and the Federal Government. Such regulations include water licences issued under the Alberta Water Act that limit oil sands industry water withdrawals in accordance to the Athabasca River Water Management Framework (AENV and DFO 2007). Licences by the Alberta Government govern the peak rate of water diversion from the Athabasca River, the annual volumes of water diversions from all sources and the release of water from oil sands mines to the natural receiving water. Oil sands mines are currently required to operate closed circuit water management systems. Exceptions include non-contact water such as surface runoff and ground water that has not been in contact with oil sands ore, and certain types of process water that have not been in contact with bitumen (only one oil sands mine operator).

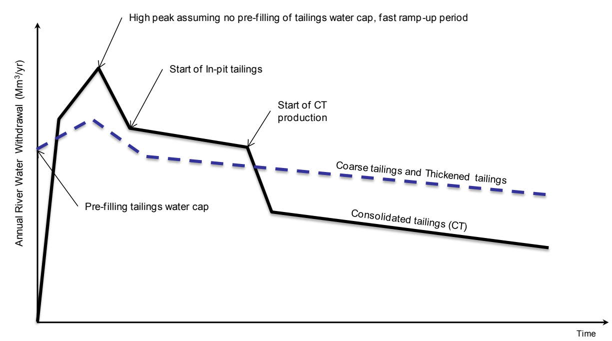

Figure 1. Typical oil sands mine life makeup water.

The makeup water requirement for oil sands mines is predominately due to the tailings pore water. The proportion of water consumption for tailings pore water storage varies among the existing oil sands mines, but typically, about 80% of the average net water demand over the life of the mine can be attributed to water storage in tailings pores. Tailings pore water accumulates over time, even though each mine recycles the release water from tailings ponds. Such recycle water derives from excess tailings slurry transport water as well as consolidation of tailings deposits (mainly fine tailings) over time. The requirement for pore water storage drives the requirement for makeup water, as shown on Figure 2.

Figure 2 illustrates that water is recycled effectively within the various processes of the bitumen extraction plant. Typically, every 1 m^3 of makeup water withdrawn from the Athabasca River is utilized alongside about 4 m^3 of recycle water from tailings slurry and collection of on-site drainage water. This ratio differs depending on the tailings management system that is used at each mine, and on-site drainage conditions.

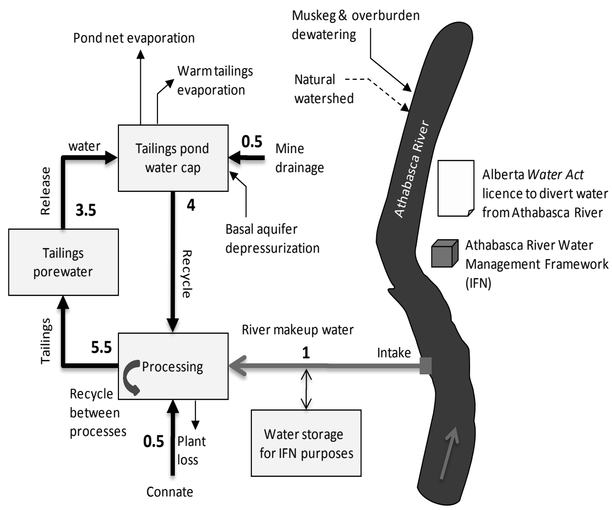

Figure 2. Typical oil sands mine site water balance.

Each oil sands mine has several licences for sourcing water, including water collection from the mine footprint (surface water runoff), collection of any groundwater that enters the mine pit and water diversions from the Athabasca River or tributaries of the Athabasca River. The majority of water supply is from the Athabasca River, particularly during the early stages of mining.

The potential maximum water diversion rate from the Athabasca River (peak pumping capacity) typically ranges from 3.5 to 6 times the rate (volume) of bitumen production. The potential maximum rate is far higher than the typical average annual river diversion rate because of variations in water demand, temporary water requirements for storage, variable natural inflows from surface and ground water, and variations in ore processing rates.

Average water withdrawals by oil sands mines are also much smaller than the annual licenced water diversions from the Athabasca River. Licenced diversions are elevated to accommodate fluctuations in water demand and uncertainty in estimates of water usage. Historical water use in the oil sands industry is presented on Figure 3, including 2008 during startup of the newest mine, the Canadian Natural Resources Limited Horizon Project. The ratio of actual average Athabasca River makeup water to bitumen production volumes is about 2.3 (water:bitumen). The 118 Mm^3 Athabasca River withdrawal in 2008 is about 50% of the 256 Mm^3 licensed allocation of Athabasca River water for oil sands mining.

Athabasca River water diversions represent about 75% of the total water supply required by the oil sands mines.

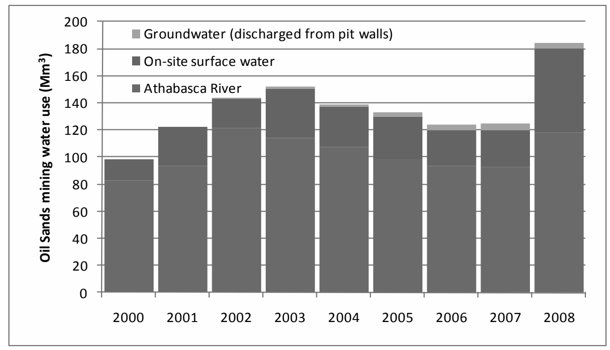

Figure 3. Historical oil sands mining water use.

SIGNIFICANCE OF WATER DIVERSIONS ON ATHABASCA RIVER FLOWS

The average river withdrawal rate in 2008 was about 3.8 m^3/s equivalent to 0.5% of the 745 m^3/s long-term annual average water yield for the Athabasca River basin (Golder 2008). During the winter low flow period, the 2008 withdrawal would have been about 4% of the 88 m^3/s historical minimum river flow recorded in December 2001 during river freeze-up (weekly average flow in the Athabasca River, as measured by Environment Canada near Ft. McMurray).

Oil sands mining production is currently projected to increase from about 730 kbpd in 2010 to about 1,800 kbpd by 2025 (CAPP 2010). If the current water usage continues (discounting the trends to reduced water usage with time as the mines mature), the oil sands mining industry will need to divert Athabasca River water at an average rate of about 7.6 m^3/s. This is equal to about 9% of the historical minimum recorded flow in the river. The more likely water use rate would be lower because the enlarged mine footprint (increased surface runoff), reduced withdrawals for initial tailings water storage, higher density tailings material due to consolidation and implementation of new tailings management systems that consume less water and produce higher density tailings material.

Although river diversions at other major rivers in Alberta are far more significant relative to average and low flows of the river, Alberta Government regulators have intervened to control the diversion of water from the Athabasca River. The Athabasca River Water Management Framework (AENV & DFO 2007) currently limits the total instantaneous withdrawal of Athabasca River water, with the most restrictive limits during winter low flows. As a result of this framework, future mines may require additional water storage to help offset the loss of water availability during winter low flow periods.

Reducing peak and average water diversions form the Athabasca River is an important goal for the oil sands industry considering the significance of river water withdrawals relative to Athabasca River flows as perceived by regulators and many stakeholders. Reducing the major consumptive use of water, that is water stored in pores of tailings material, is therefore a worthy objective for the oil sands industry.

ESTIMATION OF WATER STORED IN TAILINGS MATERIAL

Estimation of water volumes that are stored in tailings material as pore water is highly variable depending on varying ore material properties, ore/tailings processing technologies, and deposition conditions. The variability of consumptive water usage for tailings pore water storage makes it nearly impossible to develop a definitive comparison of tailings management systems based on consumptive water use for pore water storage. This explains why comparison of tailings management systems based on water consumption has been given relatively little attention in the literature and in the Tailings Roadmap Project that was initiated by Alberta Innovates and the OSTC. However, there is merit in attempting a comparison of water consumption for pore water between the various tailings management systems to give a high level understanding of the processes affecting water consumption, and a rough understanding of the magnitude of water that is needed to satisfy water demand for pore water storage. This paper provides such a high level understanding aimed at a knowledgeable stakeholder.

A detailed comparison of the variable water demands associated with an array of optional tailings management systems is not presented in this paper because of variable site conditions, mine processes and tailings management plans. However, such a detailed comparison should be conducted at the detailed planning level for each mine development or mine expansion, when it is possible to determine definitive pore water requirements based on site-specific ore properties, known ore/tailings processing technologies and specific tailing disposal schemes that need to be compared. Such a comparison at the detailed planning level is necessary because of the great importance of minimizing water usage, specifically at each oil sands mine and collectively in the oil sands industry.

Tailings material should ideally be stored in a dry state with all water used for ore extraction and tailings conveyance reused. Such an ideal cannot be achieved in a water based extraction and tailings processing/conveyance system because it is impossible for all water to be effectively removed from the tailings material and reused. Water ponds can be readily removed from the tailings area with the aid of pump barges but the remaining tailings are almost entirely saturated. Sand tailings can theoretically be drained by pumped wells or piped under-drains. However, even with complicated gravity drainage and pressurized or vacuum water removal systems, much of the water remains is permanently retained as pore water. With sand tailings, it might be possible to install gravity drains that remove about 50% of the pore water relative to the initial saturated condition. With fine tailings including saturated Mature Fine Tailings (MFT) and saturated thickened Fine Tailings, most of the water is permanently captured in the pore spaces of the tailings material even with passive gravity drainage or active pressure or vacuum filtering systems. This explains why a significant quantity of water cannot be recovered in water-based ore extraction and tailings processing/conveyance oil sands mine operations.

Estimating the quantity of water lost to pore spaces of tailings material is difficult because there are many variables affecting such a calculation. The pore water loss to saturated tailings material depends mainly on the solids content that can be expressed in terms of % solids by weight of the tailings material and can be roughly defined by the following formula.

$$V_w = \left(\frac{W_s}{S} - W_s \right) \frac{1}{\rho_w}$$

Where: V_w = volume of pore water lost to saturated tailings material

ρ_w= density of fluid (water)
S = solids content by weight (weight of solids divided total weight of fluid and solids) of the tailings material = % solids by weight/100

A detailed calculation of pore water volumes and bulk tailings material volumes specific to an actual mine operation will involve a modified formula to take account of the small amount of pore water in the ore (connate water), the residual bitumen content of the tailings material and the occurrence of two or more types of solids in a composite tailings material each with a different Specific Gravity.

Each of the parameters of the above simplified formula and modified formula for actual conditions (not shown) can be estimated with relatively little margin of error, except for the solids content. The solids content is highly variable as discussed below.

Historic tailings deposition in the oil sands region involved mainly beach disposal of whole tailings that resulted in a relatively short exposed beach composed mainly of sand deposited on a beach above the pond at the time of deposition (beach above water – BAW) and finer tailings material deposited in the pond at the end of the exposed beach (beach below water – BBW). The latter was composed of sand mixed with fines as well as fine tailings further down slope within the pond composed mainly of silt and clay. The fine tailings located in the pond further from shore is commonly known as Mature Fine Tailings (MFT) following settling and several years of consolidation. The water pond of the historic tailings deposition systems occupied (occupies) relatively large areas at each operating mine, covering the majority of the tailings deposition areas.

The historic tailings deposition resulted in a great deal of segregation of the tailings material with densities and material size compositions that varied greatly with distance from the point of tailings discharge and with densities that varied with time due to consolidation of fines.

Many of the new tailings management schemes being currently considered, planned and operated, involve mainly exposed tailings deposition with a water pond that is much smaller than the exposed tailings surface. The new tailings management systems will therefore involve much less variation in solids content along the tailings deposition profile. Therefore, for purposes of a high level understanding of the water consumed by pore water storage of various new tailings management systems, the estimation of solids content of the deposited tailings material is based on thickened, non-segregating tailings deposited on a long exposed beach involving a relatively small water pond.

The actual solids content will vary depending on a host of parameters including:

- Fines content of the beach sand
- Sand content in the TFT
- Processing specifications prior to deposition
- Chemistry and flocculation
- Slope of tailings deposit
- Weather
- Tailings discharge rate
- Length of tailings surface and degree of inundation by water at toe of the slope
- Lift thickness
- Size of tailings deposition area
- Depth of overall tailings deposit
- Degree of segregation resulting in variable solids content of deposited tailings material that is strongly affected degree of TFT thickening, amount of water in the sand tailings slurry, distance from the point of tailings discharge on a beach deposit and slope of the beach deposit.

An accurate estimate of the pore water storage of highly segregated tailings (characteristic of historic tailings disposal), sand disposal by beaching and dilute thickened tailings disposal is subject to error because these deposits are composed of tailings materials with varying solids contents and fines contents depending on the distance from the point of tailings discharge. Accurate estimation would require numerical modelling based on a comprehensive field monitoring program. Accordingly, the analysis based on only two tailings material size classifications (beach sand and processed fines) represents a simplification that is necessary for purposes of this high level comparison.

There are contrasting estimates of tailings material solids content in the oil sands industry depending on the specific experience of tailings management specialists. Table 1 presents estimates of this parameter based on a broad review of data in the literature, the experience of oil sands mines as reported in public domain submittals to Governments and the opinions of a number of tailings management specialists at Golder and elsewhere. The estimates in Table 1 tare considered to be 'middle of the road' and are not purported to be definitive and should not be used for ranking specific schemes at a mine site since a detailed analysis of known conditions should be used for a more precise comparison. Nevertheless, the estimates in Table 1 are provided for 'directional' planning and decision making.

The estimates of % solids by weight in Table 1 are based on several simplifying assumptions as follows:

- Tailings are deposited on a long exposed surface with minimal segregation
- Minimal drying time
- Sand Fines Ratio (SFR) for Thin Fine Tailings (TFT) = 1
- Fines content of beach sand or coarse sand tailings (CST) = 7%
- A nominal 30 m depth of tailings deposit

The estimates of solids content in Table 1 represent normal operations accounting for minimal off-spec tailings processing. Actual solids content would be far lower when tailings processing systems malfunction as experienced by the oil sands industry in some previous attempts to produce denser tailings material.
The estimation of solids contents in Table 1, based on experience and expert opinion, deserves some explanations as follows:

- Minimal drying time was assumed because of the extreme variation in solids content due to the effects of drying that depend on disposal and weather conditions.
- The initial deposition solids content is much smaller than the Year 1 solids content because of dewatering that occurs during deposition and Year 1 consolidation.
- Thickened Tailings by thickener (involving different initial disposal densities) do not consolidate to equal solids contents due to imperfections in deposition conditions of tailings material that is deposited at lower solids content (ponding, layering, differences in required polymer treatment and segregation along the beach profile).

COMPARISON OF PORE WATER VOLUMES FOR A SELECTION OF TAILINGS MANAGEMENT SYSTEMS

The solids content parameters discussed above were used to calculate pore water volumes for a selection of tailings management systems. The comparison begins with an estimate for the historic method of tailings deposition referred to as 'Whole Tailings Disposal'. A range of new tailings management systems is also considered below ranging from Composite Tailings (CT; otherwise known as Consolidated Tailings and Non-Segregating Tailings), to centrifuge tailings, in-line thickening and thickened Thin Fine Tailings (TFT) by thickener.

All tailings management systems itemized below, are based on permanent storage of saturated tailings material because this is most common in the oil sands industry. There are some exceptions including surface materials at the top or sides of tailings disposal areas and also some external tailings sand disposal areas where sand is deposited over permeable sand deposits that enable more rapid drainage. Some oil sands mines are considering installing gravity drainage systems within the beach sand deposit for improved drainage and partial recovery of pore water. This has the potential of reducing the water content in beach sand tailings from about 30% to about 15% water volume basis. This reduction in beach sand pore water would have the potential of reducing the estimated pore water volumes by 50%.

Except for whole tailings disposal, the new tailings management systems itemized below, incorporate large areas of exposed beached sand, large areas of exposed fine tailings and a relatively small pond area at the end of the exposed fine tailings profiles. Disposal of tailings material, especially fine tailings, on a long beach with relatively small pond (if any) is critical to the effectiveness of the tailings management system since such a configuration facilitates disposal with minimal segregation, enables drainage and drying, and reduces re-saturation. The presence of a large pond as in most of the historic tailings deposits, results in a great deal of segregation that results in a relatively low density tailings deposit (larger volumes of tailings to be stored) and soft fine tailings that cannot be easily reclaimed. The presence of a large pond also prevents drainage and surface drying of a large portion of the tailings deposit. Accordingly, reduction of pond volumes and pond areas is an essential feature of the improved tailings management schemes.

Pore water volumes were estimated for the following selection of tailings management systems:

- **Whole Tailings with Beach Sand & MFT.** This is the historic system of tailings disposal that results in significant segregation of tailings on a deposition profile ranging from beach sand to fine tailings farthest from the point of whole tailings discharge. The fine tailings become MFT following several years of consolidation. The deposition involves a short sand beach and large pond.
- **Whole Tailings with Beach Sand followed by Centrifuge MFT after several years of fines accumulation.** This system continues the historic tailings disposal system but the resulting MFT is removed by dredging after 5 to 10 years of consolidation and processed by centrifuge. The initial deposition involves a short sand beach and large pond but deposition of centrifuge MFT several years later would involve a large area of exposed centrifuge MFT and a relatively small pond (if any).
- **Beach Sand and Centrifuge Thin Fine Tailings (TFT) with SFR=1.** This system disposes tailings sand by traditional beaching methods but processes TFT by centrifuge. The deposition of centrifuge TFT involves a large area of exposed centrifuge TFT and a small pond (if any).
- **Composite Tailings based on MFT.** This system combines sand and MFT to produce a composite material that drains and gains strength more rapidly than fine tailings that are processed by other techniques (centrifuge and thickening). The deposition of CT should comprise a large area of exposed CT and a relatively small pond. Actual practice at some operating mines has involved a small area of exposed CT and a large pond resulting in excessive segregation.
- **Beach Sand and In-Line Thickening of TFT with SFR=1.** This system disposes sand by traditional beaching methods but processes the TFT by in-line thickening techniques. The deposition of thickened TFT would comprise a large area of exposed thickened TFT and a relatively small pond (if any).
- **Beach Sand and In-Line Thickening of MFT.** This system disposes sand and fines by the traditional whole tailings method but then processes the resulting MFT by in-line thickening techniques. The deposition of thickened MFT would comprise a large area of exposed thickened MFT and a relatively small pond (if any).
- **Beach Sand and Thickened TFT by Thickener deposited at 55% solids.** This system disposes sand by traditional beaching methods but processes the TFT by thickener with a relatively high density of 55% solids by weight. The deposition of thickened TFT would comprise a large area of exposed thickened TFT and a relatively small pond.
- **Beach Sand and Thickened TFT by Thickener deposited at 50% solids.** This system disposes sand by traditional beaching methods but processes the TFT by thickener with a moderate density of 50% solids by weight. The deposition of thickened TFT would comprise a large area of exposed thickened TFT and a relatively small pond.
- **Beach Sand and Thickened TFT by Thickener deposited at 45% solids.** This system disposes sand by traditional beaching methods but processes the TFT by thickener with a relatively low density of 45% solids by weight. The deposition of thickened TFT would comprise a large area of exposed thickened TFT and a relatively small pond.

Atmospheric drying with thin lift cycling is an important new tailings management system but it was not selected for comparison of pore water storage requirements because of the extreme variation in solids content depending on disposal and drying conditions.

The results of analysis and simplifying assumptions are given in Table 2 for three points in time, end of Year 1 following deposition, end of Year 10 following deposition and end of mine life assumed to be 40 years after deposition. The analysis is based on a deposited material amounting to the weight of dry tailings material of 2.22 tonnes of sand with 7% fines and 0.56 tonnes of fine tailings having a Sand to Fines Ratio (SFR) of 1.0. These volumes are equivalent to the disposal rates of 8000 tph beach sand and 2000 tph treated fine tailings reduced to a common period of 1 second so that the resulting volumes can easily be compared to an equivalent water demand rate in m^3/s without conversion.

Key assumptions used in this analysis of pore water volumes are listed below:

- SG Sand = 2.65
- SG silt/clay = 2.45
- Centrifuge Thin Fine Tailings and Thickened Tailings have an SFR = 1
- Sand is deposited with a fines content of 7% on average; actual fines content becomes much larger further downslope especially for beach below water (BBW)
- All tailings are saturated (draining beach sand would reduce beach sand pore water from about 30% to 15% by volume)
- Sand tailings (with 7% fines) is produced at a rate of 8000 tph; pore water storage volume is based on 2.22 tonnes of tailings sand (deposited in one second)
- Fine tailings (treated TFT and MFT) is produced at a rate of 2000 tph; pore water storage volumes are based on 0.56 tonnes of fine tailings (deposited in one second); assuming TFT and MFT produced at equal rates is a simplifying assumption that is balanced by the assumed 7% fines content of beach sand.

The results on Table 2 suggest that the historic tailings management system results in the highest pore water volumes. All of the new tailings management systems offer a sharp reduction in the water demand for pore water storage, relative to the historic tailings management system (Whole Tailings Disposal). The lowest pore water volumes may be achieved by CT.

CONCLUSIONS

Water supply is critical to the current operation of oil sands mines and ore processing facilities in the oil sands region of Northern Alberta. Avoiding excessive withdrawals from the Athabasca River is an important goal of many stakeholders including regulators and local residents. Though not very significant relative to the average flows of the Athabasca River, cumulative water withdrawals by the oil sands industry are of growing concern. New tailings management systems offer the potential for significantly reduced consumptive water uses particularly during mine operations. The high level analysis suggests that the new tailings management systems (selected for this analysis) offer a 40 to 60% reduction in water demand for tailings pore water storage after at the end of Year 1, and a 27 to 43% reduction in long term water use for pore water storage (after Year 40). Of the new tailings management systems that were selected for this analysis, it appears that the greatest reduction in long term consumptive water use for tailings pore water is associated with CT (commonly call Composite Tailings, Consolidated Tailings and Non-Segregating Tailings).

This assessment presents a high level understanding of the factors affecting water storage in pores of selected tailings materials and a rough estimate of quantities required for pore water storage. A site specific analysis based on local ore conditions, tailings processing/conveyance systems and specific tailings disposal plans, is recommended for a more precise comparison of tailings management systems and their ability to reduce water diversions from the Athabasca River.

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Table 1. Assumed Average Solids Contents for Various Tailings Materials.

Tailings Materials	**% Solids – 1 year**	**% Solids – 10 Years**	**% Solids – Post Closure**
Sand	80	80	80
Mature Fine Tailings (MFT)	30	40	45
Composite Tailings (CT) based on MFT*	75	80	80
Centrifuge MFT	60	65	65
Centrifuge Thin Fine Tailings (TFT) with	65	65	65
In-line thickening of MFT	70	70	70
In-line thickening of TFT with SFR=1	75	75	75
Thickened tailings by thickener deposited	60	65	70
Thickened tailings by thickener deposited	55	60	65
Thickened tailings by thickener deposited	50	55	60

*Personal Communication – Gord McKenna

Table 2. Comparison of Pore Water Volumes for Various Tailings Deposition Technologies.

Tailings Technology	Pore Water Volume[1] – 1 Year After Deposition (m^3)				Pore Water Volume[1] – 10 Years After Deposition (m^3)				Pore Water Volume[1] - Post Closure 40 Years After Deposition (m^3)			
	Combined	Beach Sand	Processed Fines	Total	Combined	Beach Sand	Process ed Fines	Total	Combined	Beach Sand	Processed Fines	Total
Whole Tailings with Beach Sand & MFT		0.56	1.31	1.86		0.56	0.84	1.40		0.56	0.68	1.24
Whole Tailings followed by Beach Sand & Centrifuge MFT		0.56	0.37	0.93		0.56	0.30	0.86		0.56	0.30	0.86
Beach Sand and Centrifuge Thin Fine Tailings (TFT) with SFR=1		0.56	0.30	0.86		0.56	0.30	0.86		0.56	0.30	0.86
Composite Tailings based on MFT (CT with SFR of 4 to 5)	0.93			0.93	0.7			0.7	0.7			0.7
Beach Sand and In-Line Thickening of TFT with SFR=1		0.56	0.19	0.74		0.56	0.19	0.74		0.56	0.19	0.74
Beach Sand and In-Line Thickening of MFT		0.56	0.24	0.80		0.56	0.24	0.80		0.56	0.24	0.80
Beach Sand and Thickened TFT by Thickener deposited at 55% solids		0.56	0.37	0.93		0.56	0.30	0.86		0.56	0.24	0.80
Beach Sand and Thickened TFT by Thickener deposited at 50% solids		0.56	0.46	1.01		0.56	0.37	0.93		0.56	0.30	0.86
Beach Sand and Thickened TFT by Thickener deposited at 45% solids		0.56	0.56	1.12		0.56	0.46	1.01		0.56	0.37	0.93

Notes:
1. The total pore water volume represents an equivalent water demand in units of m^3/s
2. Beach Sand includes 7% fines; production rate is 8000 tph
3. Processed Fines (MFT) contains no sand sizes; production rate is 2000 tph
4. Processed Fines (Centrifuge TFT, In-Line Thickened TFT and Thickened TFT by Thickener) contains 50% sand (SFR=1); production rate is 2000 tph
5. CT contains sand and fines with a production rate of 10,000 tph (combination of 8000 tph beach sand and 2000 tph processed fines)

AN INTEGRATION OF LONG-TERM MINE PLANNING, TAILINGS AND RECLAMATION PLANS

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ABSTRACT

Bitumen extraction in oil sands surface mining generates huge volumes of slurry known as tailings, which is sent to tailings ponds. Oil sands operators must reclaim tailings ponds before leaving the mine sites. Typically, tailings and reclamation models are not included in mine planning models. In this research, an integrated optimization framework is developed that optimizes the long-term mine production plan with respect to tailings management considerations. The proposed mixed integer linear programming (MILP) model maximizes the net present value and at the same time, minimizes reclamation material handling cost. The integrated model is constrained by reclamation material requirements, the capacity of tailings facility and the volume of tailings components. The volume of tailings is calculated through tailings model, based on Clark hot water extraction method. The proposed model is coded in Matlab, run with CPLEX and verified by testing on a real-case oil sands data set.

INTRODUCTION

In oil sands surface mining, the material is extracted from an open-pit mine and being sent to processing plant if the bitumen content is above the cut-off, or to the waste dump if it is low grade material. There are environmental consequences associated with both cases (Singh, 2008; Woynillowicz et al., 2005; Rodriguez, 2007). High-grade material processing results in huge volumes of slurry, known as tailings, that is kept in tailings ponds for further treatments. Remaining bitumen in tailings and leaking of tailings to nearby fresh water resources are examples of environmental issues around tailings ponds, while disturbing the landscape is an example of environmental issues in waste material stream. Oil sands operators are now obliged to have long-term reclamation plans to ensure that the environmental consequences of operations are under control. Almost all of reclamation material comes from mining operations (top soil) or oil sands processing (tailings coarse sand).

The volume of tailings is also a key factor in mining production, because the available area for holding the tailings is limited to the lease areas. In addition to the total volume of tailings, having control on tailings material such as fine material and water is important too. Therefore, it is reasonable to consider the capacity of tailings facilities and limitations for the tailings components in the mine planning.

Sustainable mining concept is developed to include environmental aspects of the operations in mine planning and mine design. In the case of oil sands surface mining, the most important environmental concern is dealing with tailings ponds and their reclamation. In mine design phase, the overall specifications of the mine, such as pit limits and ramps are designed. In this phase, tailings management and reclamation requirements are involved by defining a new cost term of "environmental costs". Some qualitative measures such as audit reports are used to investigate the compatibility of a mining project with environmental requirements. The works done by Sinding (1999) and Manteiga and Sunyer (2000) are examples of qualitative approach, mostly useful for mine design phase.

In mine planning, the objective is to find the optimal production schedule to maximize the net present value (NPV), with respect to a set of constraints, such as block precedence and capacity constraints for mining and processing. In this case, environmental concerns must be quantified so to be included in mine planning optimization. Examples of quantitative approaches in sustainable mining are done by Odell (2001) and Shepard (2005).

Typical oil sands mine planning frameworks do not include tailings management and reclamation planning. Instead, tailings and reclamation plans are developed separately from mine planning. The missing part in mine planning literature is the merger between conventional long term mine planning, tailings and reclamation plans. The objective of this research is to develop an integrated long-term optimization framework that maximizes the NPV over the mine life, with respect to tailings capacity constraints and material requirement for reclamation. The objective function includes a new term to minimize reclamation material handling cost.

PROBLEM DEFINITION

Different environmental costs are considered to find the optimal pit limit in mine design phase in recent years (Odell, 2004; Rodriguez, 2007). Moreover, there have been many works addressing the maximization of NPV in mine planning (Askari-Nasab et al., 2010; Askari-Nasab et al., 2011). In addition to pure mine planning and mine design, tailings plan is also included in some models (Ben-Awuah and Askari-Nasab, 2011). However, the critical aspect of mine planning is a merger between all these areas: profit maximization with respect to tailings plan and reclamation costs.

A good example of reclamation plan is what Shell Canada proposes in fulfillment of Directive 074 (McFadyen, 2008). Shell Canada considers dedicated disposal areas (DDA) for JackPine Mine (JPM) site at Athabasca river region in Alberta. Each tailings facility is consisted of multiple cells adjacent to each other. Thickened tailings (TT) is discharged into the cells consecutively, meaning that the cells receive TT in the order of their location, e.g. west to east. The cell that receives the discharge earlier is considered to be the first DDA and after a certain period of time, it changes into a dried and reclaimed landscape. Then, reclamation starts in the next cell. The drainage system is designed in such a way that any flow of surface water from DDA is discharged to the next cell. Figure 1 illustrates the layout of JPM (Shell Canada, 2011).

There are three phases involved in site decommissioning; construction, operations and closure. Mine planning has effects on all three phases. For instance, waste material that is used later for construction of starter dyke and external dyke walls in reclamation are generated in mining operations phase. Coarse sand tailings (CST) and thickened tailings (TT) that are used in filling and capping in operation phase are produced in oil sand processing which is directly influenced by mine planning. The example of Shell's DDAs shows that mine planning and reclamation plans are completely related to each other. In other words, any change in mine planning results in changes to the amount of produced tailings and hence, the material requirement for site reclamation.

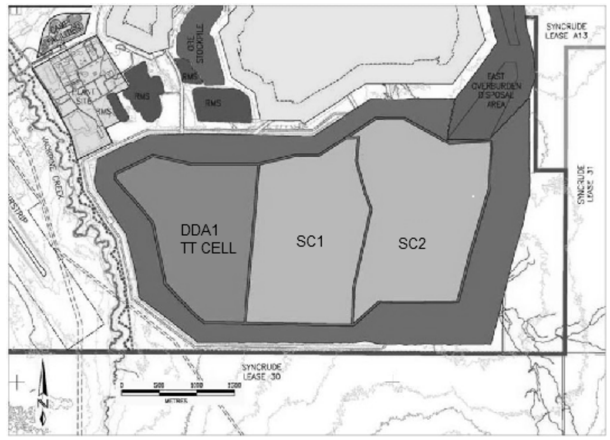

Figure 1. Layout of DDA1 within the JPM ETF.

Considering that the required material for reclamation of tailings facility is produced in mining and processing operations, the objective is to develop an integrated long-term mine plan that includes the capacity of tailings facility and reclamation material requirement. In other words, the optimal production plan should not only maximize the NPV of the extracted oil sands, but it must satisfy the capacity constraints for holding the tailings and guarantee to provide the required reclamation materials for capping. Therefore, the integrated model should be capable to manage the stream of materials for reclamation.

A typical MILP model that maximizes the NPV with respect to mining and processing constraints is organized as follows:

Maximize (NPV)
Subject to:

- *Processing plant constraints*
- *Mining capacity constraints*
- *Extraction precedence constraints*

In an integrated mine planning, in addition to the typical constraints, the costs and constraints for reclamation material requirements are included in the model. Decision variables are revised in such a way that sending portions of extracted material to different destinations becomes possible. Limited capacity for tailings facilities is also taken into account in the integrated model. Therefore, new objective function terms and also some constraints are added to the previous version of MILP model to include tailings capacity, reclamation material requirement and reclamation cost. The organization of the revised MILP is as follows:

Maximize (NPV – reclamation costs)
Subject to:

- *Processing plant constraints*
- *Mining capacity constraints*
- *Extraction precedence constraints*
- *Tailings capacity constraints*
- *Reclamation material requirement constraints*

A schematic overview of integrated mine planning model is illustrated in Figure 2.

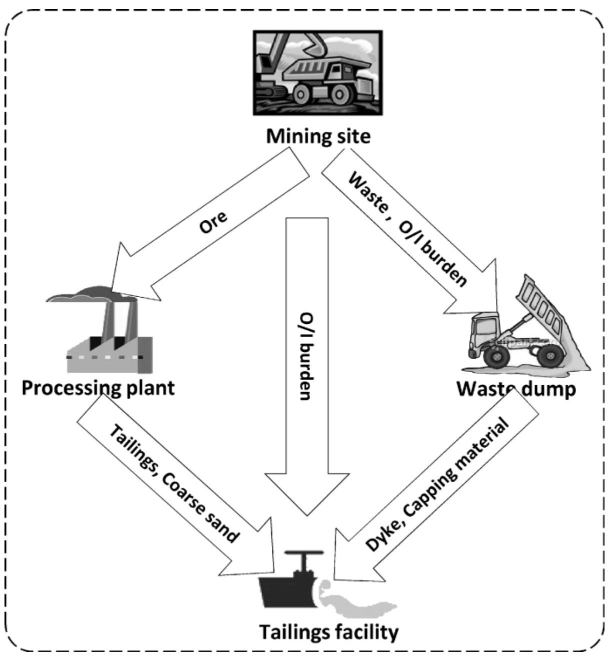

Figure 2. An integrated mine planning, modified from Badiozamani and Askari-Nasab (2012).

THEORETICAL FRAMEWORK

Two concepts that are investigated and developed in this research are; (1) advancement in precedence definition (directional mining) and (2), tailings calculations. Therefore, prior to proposition of mathematical model, it is essential to discuss about directional mining and tailings model.

Directional Mining

Vertical precedence relation between blocks is well introduced and implemented in mine planning models (Askari-Nasab et al., 2011). Except for the very first level of blocks on top of the pit (first bench), it is assumed that there are nine blocks on top of each block in a regular grid that all must be extracted prior to get access to the block under. A schematic configuration of vertical precedence is illustrated in Figure 3(a).

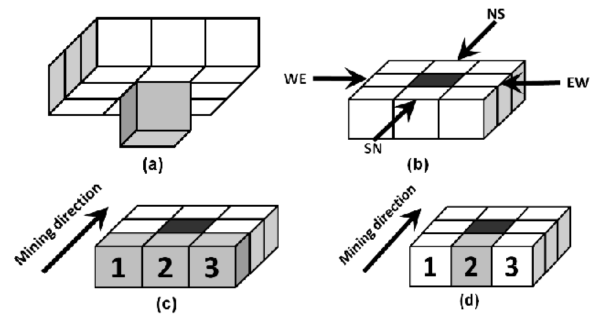

Figure 3. Block precedence in vertical (a) and horizontal (b, c and d) directions.

In many applications such as in oil sands mining, defining vertical precedence does not guarantee a feasible practical solution from mining point of view. In addition to the vertical precedence, horizontal precedence also must be defined to have a practically feasible optimal solution. In some case of oil sands surface mining, it is essential to clear the pit completely from one side and push the mining face forward in a specific direction so that after few periods, tailings can be pumped into the excavated empty pit (in-pit tailings facility). Directional mining includes group of mining problems with a specific horizontal direction. Many different horizontal directions can be defined for any problem. However, there are eight main directions, including four in east-west and north-south directions and opposites, and four in north_east-south_west and north_west-south_east directions and their opposites. Four of directions are illustrated in Figure 3(b).

Figure 3(c) shows an example of horizontal precedence relation in west-east direction. Block numbers 1, 2 and 3 must be extracted completely to access the shaded block in centre (block number 5). In other words, blocks 1, 2 and 3 are horizontal predecessors for block 5. However in a simplified version of horizontal precedence, block 2 is assumed to be the only horizontal predecessors for block 5, as illustrated in Figure 3(d). This assumption decreases the number of precedence relations and hence, reduces the number of precedence constraints which means a decrease in the problem size.

For the sake of practicality, blocks are aggregated to mining cuts as the input into the MILP model (Tabesh and Askari-Nasab, 2011). This means that any precedence must be defined between mining cuts, not blocks. Knowing that each block belongs to which mining cut, it is easily possible to map block precedence to cut precedence. Since all mining cuts do not have regular shapes, they are not in a regular grid. However, if a horizontal or vertical precedence can be defined between any of blocks within two cuts, then a precedence relation is defined for cuts.

There is one more step in precedence definition; pushback precedence. Any mining pit is partitioned in a number of pushbacks. Currently, pushbacks are defined based on some increments in the revenue factor associated with selling price of the ore. In practice, these smaller pit subsets are used in mine planning. A good example that shows necessity of pushback precedence is in-pit tailings facility. In order to guarantee that the pit is completely cleared from one side and the surface is pushed in a specific horizontal direction, it is needed to define precedence between pushbacks. Figure 4 shows the way that pushback precedence is defined in this paper.

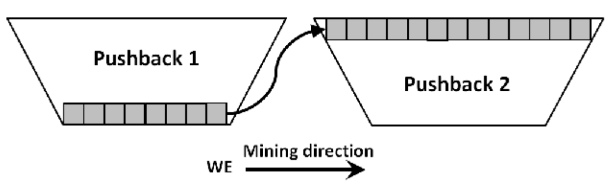

Figure 4. Pushback precedence.

Supposing that pushback 1 is predecessor of pushback 2 in this specific mining direction, all the mining cuts at the bottom of pushback 1 are defined as predecessor for all the mining cuts at the very top bench in pushback 2. This set of new precedence is called pushback precedence and is added to the precedence list.

Tailings Model

An oil sand processing plant receives the ore feed, and then the material is sent to crusher reduce particles size. Then, the slurry is prepared by adding hot water to the feed. The slurry goes to the separation cell and separates into two streams, bitumen froth and waste material, through flotation process. The bitumen froth is processed to recover the bitumen and separate the water from the froth that is added in former steps. The waste stream is sent to the cyclone for further separation of fine and coarse material, using a centrifuge. The cyclone has two output streams, (1) the overflow slurry with more fine material and water and (2) the underflow slurry that contains mostly coarse material. This is the basis for Clark hot water extraction method. Figure 5 shows a schematic view of the process flow sheet that Suncor energy uses for bitumen recovery (Suncor, 2009).

To have the volume of total produced tailings and also its components (water, fine material and coarse sand) to be used in mine planning model, Suncor's flow sheet and its mass balance relations are used in this paper.

The Mixed Integer Linear Programming (MILP) Model

The mixed integer linear programming (MILP) is used to formulate the long-term mine production scheduling problem. The MILP formulation framework is developed based on mining-cuts (an aggregation of mining blocks) as the units of scheduling. The optimization model has two objective functions, one for NPV maximization and one for reclamation material handling cost minimization that are combined in a single function. The assumption of ore parcels within mining cuts that provides selective mining is the basis for the concept of economic mining-cut value. The overall profit from mining a mining-cut is proportional to the value of the cut and the total costs associated with mining, processing and material handling cost for reclamation at a specified destination. The discounted profit from mining-cut k is the difference of the discounted revenue obtained by selling the final product contained in mining-cut k and the discounted cost involved in mining mining-cut k as waste (Askari-Nasab et al., 2011). In this model, the extra discounted cost of mining over/inter burden (OI) and tailings sand (TS) material for reclamation are added as new terms in calculation of economic mining cut value.

The model includes different sets of constraints, including mining and processing capacities, material requirements for reclamation, tailings capacity, the quality of ore feed to the processing plant, mass balance relation between different parcels of mining cuts and precedence constraints. Directional mining is one of the newly added constraints to the previous mine planning models such as Ben-Awuah, and Askari-Nasab (2011), that suits reservoirs such as oil sands.

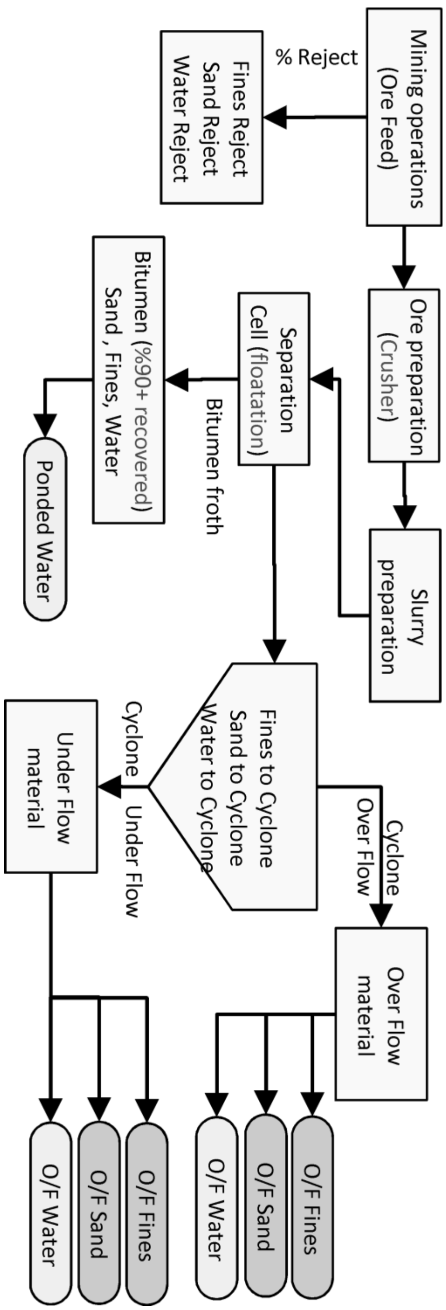

Figure 5. Part of Suncor flow diagram, modified from Badiozamani and Askari-Nasab (2012).

To include tailings management and reclamation plan in the optimization framework, decision variables and cost coefficients are defined in such a way that enables the optimizer to send different portions from each block to different destinations. The model follows the concept of dynamic cut-off, meaning that the optimizer determines the destination of each parcel in such a way to maximize the NPV, rather than having a fixed cut-off that predetermines material destination based on ore content of the parcels.

A complete formulation of the mathematical model including notation for sets, indices, parameters, decision variables and the MILP model can be found in Badiozamani and Askari-Nasab (2012).

CASE STUDY

A real oil sands data set is used to check the results of running the MILP and tailings models. The data set includes 45,648 blocks of 50 by 50 by 15 meters, aggregated into 980 mining cuts. There are two pushbacks in the case, separated by a river. Processing plant and the waste dump at tailings pond are two possible destinations for extracted material. The problem is solved for four horizontal mining directions and ten periods. Two sample plan views of resulted schedule for east-west and south-north directions are illustrated in Figures 6 and 7, respectively.

Different periods are represented by period numbers in plan views. Figures 6 and 7 show that the schedule follows pushback precedence and mining direction in each pushback.

A MATLAB program (MathWorksInc., 2011) is developed to run the model for the presented case. The code calls the TOMLAB/CPLEX (Holmström et al., 2009) to solve the MILP model. The code is executed on an Octa-core Dell Precision T7500 computer at 2.8 GHz, with 24GB of RAM. The results for four mining directions are reported in Table 1. Comparing the results shows that mining direction with the highest value of objective function is east-west, with the value of \$26256 M.

Total tonnage of mined material, including ore, waste and over/inter burden (dark gray bars), together with total tonnage of material that is sent to processing plant (light gray bars) for east-west and south-north directions are illustrated in

Figures 8 and 9, respectively. Horizontal lines represent mining and processing capacities.

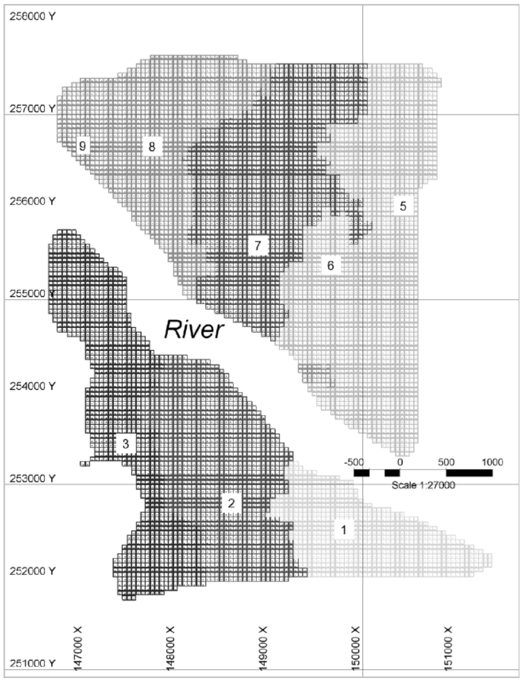

Figure 6. Sample plan view for east-west mining direction.

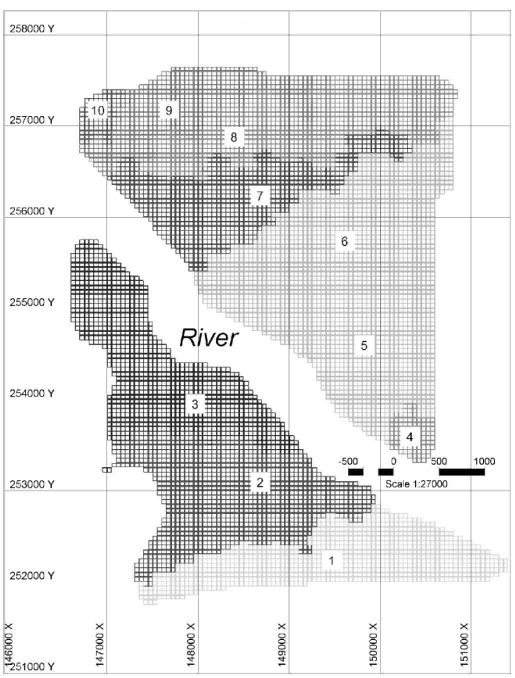

Figure 7. Sample plan view for south-north mining direction.

Figures 8 and 9 shows that mining production is almost uniform that is practically a good solution. However, ore feed to the processing plant is not as uniform as mining production. This is because of the fact that in east- west direction, the ore (bitumen) is not accessible with a smooth rate, while mining in south-north direction results in a more smooth ore feed (Figure 9).

Table 1. Comparing results for different directions.

Direction	Solution gap (%)	Run time (min)	Objective function (M$)
West - East	1.86	194	23109
East – West	1.00	7	26256
South - North	1.99	5	24487
North - South	1.15	18	23998

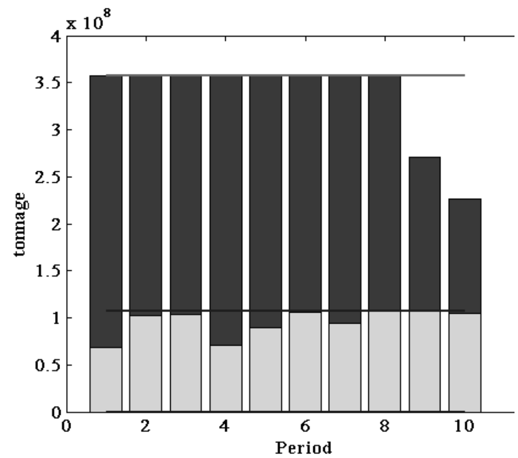

Figure 8. Mining and processing tonnages for east-west direction.

Total volume of tailings in periods for east-west direction is compared in Figure 10. In addition to the total volume of tailings, the quality of tailings is practically important. The results show that all tailings components including water, fine material and tailings coarse sand are within the specified input ranges. Finally, reclamation material that is produced in south-north mining direction is reported in Figure 11. Since over/inter burden material does not have any value added to the NPV, the model keeps it in its minimum possible level, only to provide required material for reclamation.

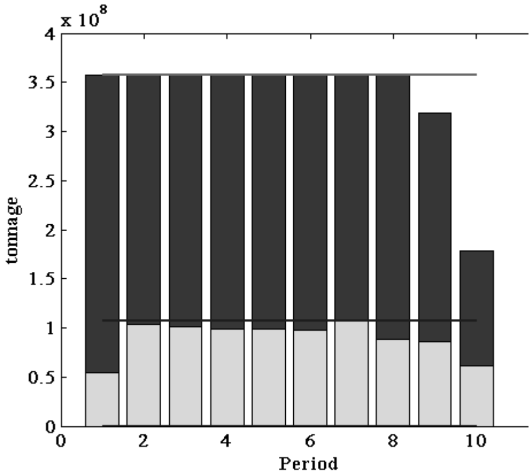

Figure 9. Mining and processing tonnages for south-north direction.

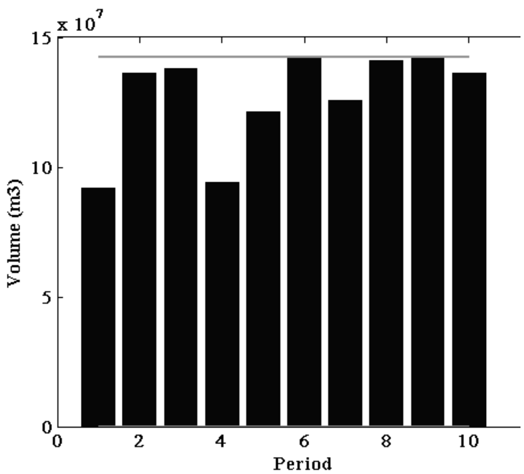

Figure 10. Tailings volume for south-north direction.

CONCLUSIONS

Processing of oil sands produces a huge volume of tailings, which is pumped into tailings ponds and kept there for a long period of time. Oil sand operators are responsible for monitoring their tailings facility environmental situation and decommissioning of mining site and tailings facility prior to leave the mine site. For decades in oil sands industry, tailings and reclamation plans have been developed separately from mine production plan. In this paper, a new MILP model is developed that maximizes the NPV and at the same time considers material handling costs associated with reclamation operations as the new term in its objective function. Furthermore, the proposed MILP result ensures that the required material for site reclamation is available. Suncor's bitumen extraction process flow sheet is used to capture the mass balance relation in bitumen extraction process. The formulation that is used to calculate the tailings volume is verified by testing the formulation on real data from an oil sands surface mining case. Performance of the proposed MILP model is tested on real case oil sands data sets. The results show that the optimal production schedule meets material requirements for reclamation and also the produced tailings volume is within tailings capacity range in all periods. Moreover, the mining pattern follows certain horizontal direction. The integer optimal solution is found with 1.21% from problem's optimal solution. The next step in this research is to investigate some methods for problem size reduction. One way for reducing problem size is using of larger mining units, by aggregating mining cuts to mining panels. Although the block model resolution is lost with the new mining unit, but the solution closes more to what happens in real mining operations. Reducing the number of decision variables by eliminating irrelevant periods for extraction of near surface or deep mining blocks is the other method for problem size reduction. Another potential improvement in solving the MILP model is to use more efficient solution methods, such as Lagrangian relaxation algorithm, to find the solution in a reasonable time for large-scale problems.

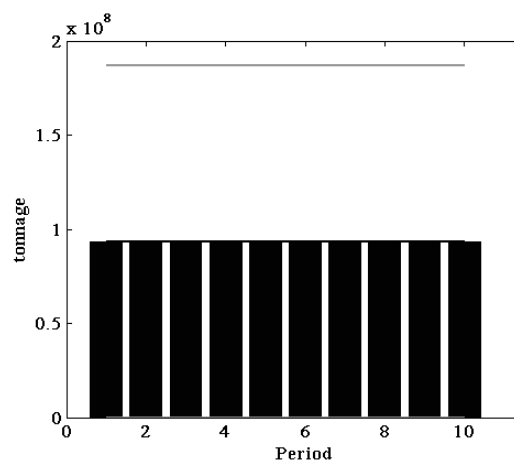

Figure 11. Over/Inter burden material for reclamation in south-north direction.

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Session 3

Chemical Additives

DEVELOPMENT OF FLOCCULANTS FOR OIL SANDS TAILINGS USING HIGH-THROUGHPUT TECHNIQUES

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ABSTRACT

Recent government regulations and growing environmental concerns are driving the need to remediate large amounts of tailings produced during bitumen extraction from Canadian oil sands. However, remediation of these tailings is particularly challenging due to the multi-component nature of the tailings, the variation in their mineralogical and bitumen composition from site to site, and the inherent colloidal stability of clay particles which resist rapid settling.

One approach to remediate the tailings is to use additives to flocculate the fine particles. However, to design an effective flocculant requires knowledge of a large number of variables. These include flocculation efficacy as a function of molecular structure (chemical composition, molecular architecture, molecular weight), dosage, effectiveness in a wide range of tailings sources with different compositions, insensitivity to ionic content of recycle water used to prepare flocculant solutions, and robustness to shear conditions before, during and after flocculation.

This paper describes the use of high-throughput methods to accelerate the development of improved flocculants for remediating oil sands tailings. The robotic preparation of samples enables reproducible and precise control of flocculant solution injection and mixing with the tailings. Using automated digital image analysis, particle settling behavior can be rapidly and quantitatively compared for flocculants added under a wide range of pH, ionic strength and dosage levels.

INTRODUCTION

The ever increasing demand for energy has fueled the development of the oil sands industry. However, one of the byproducts of the hot caustic bitumen extraction process is a tailings waste stream containing fine clay particles that is particularly challenging to remediate. Ideally the warm water could be reclaimed from this waste stream and recycled for use in the extraction process, and the resulting dewatered solids could be used to reclaim the land. There are many methods under development for reclaiming the tailings, ranging from thickening tailings, addition of inorganic coagulants, centrifugation and others (Sobkowicz et al. 2009, Mikula et al. 2009, Wells et al. 2011). Most of the methods under development rely on the addition of chemical amendments such as high molecular weight polymers to flocculate the clay particles. The performance requirements for these flocculants are quite high. A flocculant must capture a sufficient amount of clays and silts, while rapidly dewatering to produce a high solids content deposit with high yield stress. It should also be stable to shear flows during transportation and mixing, perform well over a range of tailings compositions but yet have no effect on the extraction process stream after released water recycle. In addition, potential flocculants should be easy to handle and mix into tailings streams, and ideally be inexpensive. This combination of properties is challenging for chemical suppliers to provide. Due to the wide range of possible chemistries under consideration and the specific requirements of each operation, it is critical to have rapid performance screens to guide chemical structure/property relationships and accelerate product development.

High-throughput or combinatorial approaches are particularly useful when a large number of variables must be investigated during the product development process. For new flocculants, this might include molecular composition and architecture, molecular weight, dosage level, mixing or shear conditions, and efficacy in tailings having different compositions. The ability to rapidly define the effect of multiple variables on performance has proven to be very useful in multiple industries, including new catalyst development, new multi-component personal and household care items, architectural coatings, food additives and biocidal formulations (Peil et al. 2004; Tucker et al. 2008, Johnson et al. 2009, Mohler et al. 2009, Kalantar et al. 2007). Recently, high-throughput solubility and rheology workflows have been used in the development of new rheology modifiers for use in oil and gas drilling (Mohler et al. 2011).

For this study, a new high-throughput workflow has been developed to follow the settling rate and floc appearance of tailings after mixing with a wide variety of potential flocculants. The workflow described here encompasses formulation preparation using robotic liquid handlers, as well as formulation characterization based on automated analysis of digital images.

EXPERIMENTAL

High-Throughput Formulation Preparation

The software application Library Studio (FreeSlate Inc., Sunnyvale, CA) was used to design the high-throughput sample preparation and settling tests, and convert the design into a template suitable for use by the robotic liquid handler. Experiments were typically done in small scale (~5 mL) with samples arranged in a 4x6 plate array. A typical experimental arrangement is shown in Figure 1. In this design, eight samples were prepared with varying amounts of flocculant in each to enable a dose-response curve to be generated. An appropriate amount of tailings and process water were added to maintain a constant solids loading. In this configuration, up to 24 different samples could be prepared and tested in any given experimental run so that variations in ionic content, molecular composition or other variables could be investigated.

Stock solutions of various flocculants (0.5 weight percent) were prepared conventionally by dissolution in water with magnetic mixing at ~100 rpm for 1 hour. Each stock solution was used within 4-6 hours of preparation, and new stock solutions were made each day. Mature fine tailings (MFT) were obtained from various operators in Alberta; for some experiments, the tailings were diluted with water before addition of the flocculant to study the effect of varying the solids content of the tailings. Water used for this dilution process was obtained from the same ponds from which the tailings were sampled. The tailings were shaken for 10 minutes in a horizontal shaker before dispensing into the 10 mL vials.

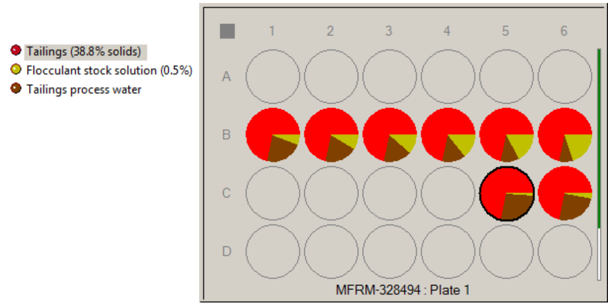


Figure 1. Typical experimental design for high-throughput formulation preparation. The relative amounts of tailings, flocculant and process water for each populated cell in the 4x6 array are represented in a pie-chart format. This example plate shows the compositions for eight samples to be prepared by the robot.

A diagram of the workflow is shown in Figure 2. All sample additions and formulation mixing were accomplished using a liquid handling robot (FreeSlate Inc., Sunnyvale, CA). First, any process water required to dilute the tailings was added to the dispensed tailings with a robotically-controlled 1000 μL disposable positive displacement (PD) pipette. The mixture was stirred at 100 rpm for 5 minutes using a magnetic mini-stir disk (V&P Scientific, CA). The desired amount of stock flocculant solution was then accurately delivered to each vial robotically with a 250 or 1000 μL PD pipette while stirring at 100 rpm, and stirring was allowed to continue for the desired length of time (usually 3 minutes). The total amount of tailings and flocculant was approximately 4-5 g in each vial. Both the speed of dispense as well as the point of delivery in the vial can be controlled by the robot. Unless otherwise noted, the flocculant stock solution was reproducibly added at a range of 1 to 4 mm from the top of the tailings. After completing the dispense and stir steps, the solution was capped and moved onto the imaging portion of the workflow. All information about the sample creation (weights, stirring conditions, dates/times of creation) was automatically stored in a fully searchable database and archived for future retrieval and data analysis.

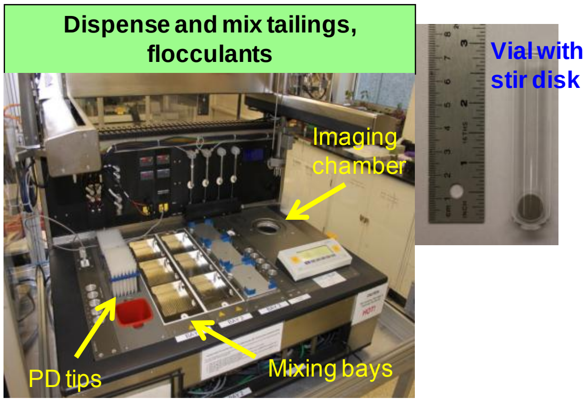


Figure 2. High–throughput workflow for preparing flocculant/tailings formulations and assessing settling behavior. Flocculant solutions are automatically dispensed into 10 mL vials (see insert) with tailings while stirring. Digital images of the samples are collected robotically as a function of time.

High-Throughput Settling Rate Measurements

To determine the settling rate, the individual vials were removed periodically from the plate by one of the robotic arms and placed in the imaging chamber on the robot deck, where a digital image of the sample was collected. Significant improvements to the image chamber lighting and lens systems were made to obtain high quality images of the settling tailings and exuded water, since the tailings themselves are an opaque brown material and challenging to image in detail. Improvements to the robotic protocol allowed images to be collected as quickly as 10-15 seconds after the cessation of mixing, and as rapidly as 5 seconds between images as needed. In this way, very rapid settling could be observed. Settling tests were usually completed over a 20 hour time period. The digital images are automatically stored in the database and are indexed to the library identification number associated with the sample generation information. The digital images were processed with a MATLAB-based algorithm that automatically extracts the mud height for further data analysis.

The current high-throughput settling rate workflow enables the settling rate of many diverse materials to be screened in a relatively short period of time. With the initial robotic protocol, typically 10 samples can be screened for a settling time of 20-24 hours. If fewer data points are desired, or if a single time point comparison is sufficient, over 100 samples could be screened in a similar time frame.

An additional advantage of the image collection is that the relative clarity of the supernatant fluid can be followed with time, as well as any evaluations of macroscopic floc structure and entrained water.

RESULTS AND DISCUSSION

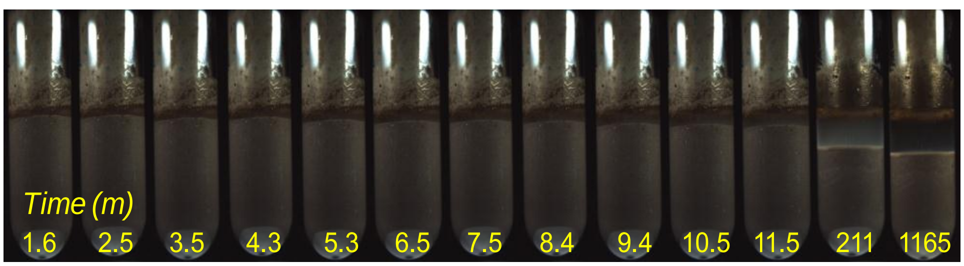
There are several possible flocculation mechanisms, such as bridging between particles, charge neutralization (coagulation), charge patch flocculation (attraction between patches of oppositely charged areas on particle surfaces), and depletion flocculation, where non-adsorbing flocculant molecules force particles to associate by osmotic forces (Gregory 1989, Hogg, 2000). For oil sands tailings waste streams with high solids content, high molecular weight polymeric flocculants have been found to be useful and are generally thought to operate by a bridging mechanism (Kitchener, 1972, Sworska et al. 2000, Gregory et al. 2011). Polymer chains are hypothesized to attach to multiple particles and facilitate agglomeration and subsequent removal of particles from the remaining solution. Flocculant molecular weight is thought to be one of the most important factors in controlling flocculation effectiveness due to its relationship with polymer size and availability of polymer loops to enhance particle attachment. Charge density of polyelectrolyte flocculants can also play a significant role since repulsion between charged segments can lead to a larger effective size of the polymer through coil expansion (Gregory et al. 2011).

The properties of the tailings themselves are also critical in determining how well the flocculant will work. Variations in clay content, clay type, particle size and residual bitumen may all potentially have an effect on flocculation, which emphasizes the importance of developing a flocculant with robust performance. Furthermore, several studies have shown the importance of hydrodynamic conditions such as flocculant addition rate, concentration during addition, and mixing conditions on flocculation efficacy (Sworska et al. 2000, Revington et al. 2010, Demoz et al. 2011). The floc itself may also be sensitive to shear conditions, either densifying or breaking up as it is mixed into the tailings.

One common screen of flocculation effectiveness is a settling rate test, typically done in a large graduated cylinder to follow the dependence of settled material or "mud" height over time. The settling rate test is adaptable to high-throughput methodology and has formed the basis of a screening program for effective flocculants for oil sands tailings described here. The ability to control and vary flocculant composition, dose levels, tailings properties, flocculant addition and mixing conditions allows the effects of these variables to be determined and promising flocculants to be quickly identified. In addition, the smaller scale of these methods requires much less material (whether tailings or flocculant) so experimental materials are more easily tested under a variety of conditions (pH, ionic water content, etc). A smaller set of flocculants with promising characteristics can then be selected for more conventional larger scale testing.

Typical settling images for a flocculant/tailings system as a function of time are shown in Figure 3. The mud line drop with time is clearly visible and for this particular system drops quickly with time initially and then slows appreciably. Using the automated image analysis algorithm, the relationship between mud line height and time can be determined and is also shown in Figure 3. The time interval between images is user-defined, so that both short and longer time processes can be followed. In addition to capturing information about settling time, the quality of the digital images is high enough that macroscopic features of the flocculation process can also be investigated as a function of time. This can give powerful insight in determining whether flocculation has occurred, assessing the onset (perhaps as a function of dose) and whether mixing or other process conditions have a deleterious effect on the flocculation.

To illustrate, images of a polymeric flocculant dosed into a 28 wt % solids tailings formulation are shown as a function of flocculant dose from 200 to 3500 ppm, after 20 hours of settling (Figure 4).



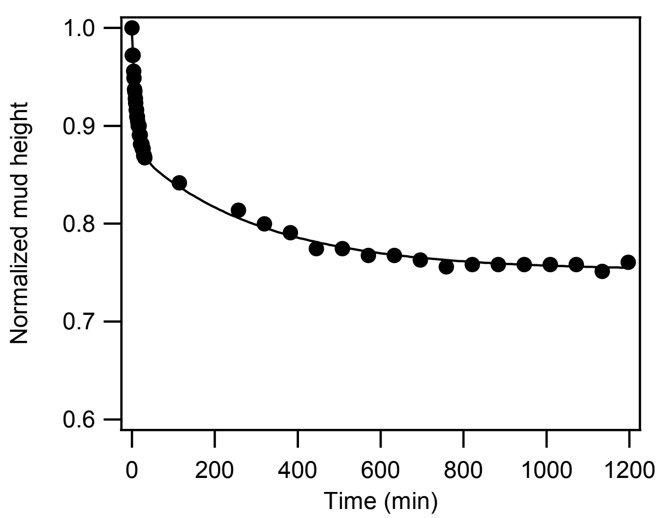


Figure 3. Images collected as a function of time of a select flocculant/tailings system (top). The relationship between mud height extracted from the digital images and time is also shown (bottom)

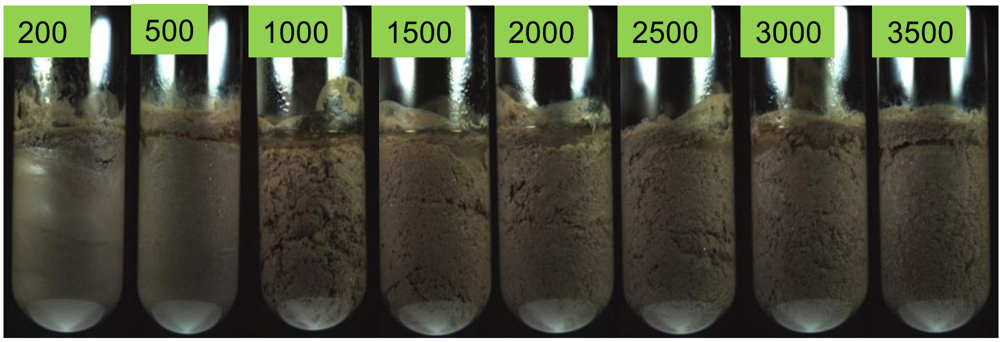


Figure 4. Images of a polymeric flocculant/tailings system (28 wt% solids) collected after 20 hours of settling, for a range of flocculant doses from 200 to 3500 ppm. The optimal flocculant dose is likely near 1000 ppm judging by the onset of dewatering and the formation of large floc particles. The narrow range of flocculant dose over which large floc size is observed indicates a potential lack of robustness in optimal flocculant performance.

For doses up to 500 ppm, the texture of the formulation is smooth, suggesting little if any flocculation has occurred. As the flocculant dose is increased to 1000 ppm, large flocs become visible. These are accompanied by a significant amount of entrained water, and a thin layer of water appears at the top of the mixture. In some respects, the amount of water exuded at 1000 ppm does not appear to be appreciable, but as with all confined settling tests, this is a static experiment and water cannot escape the container as it would in the field. Nevertheless, further increases in the flocculant dose beyond 1000 ppm cause a decrease in apparent floc size, and less exuded water. Coupled with the observation that the surface exhibited a layer of “slime” with somewhat elastic behavior for the samples dosed above 1000 ppm, the optimal dose for this flocculant system is judged to be near 1000 ppm. The relatively narrow region of flocculant concentration over which large floc and good dewatering is observed indicates this particular flocculant may have a relatively small process window with respect to flocculant performance.

The effect of mixing conditions (stirring speed and time) on flocculation effectiveness can readily be investigated using the robotic system, enabling comparisons of the relative stability of various flocculants to shear. For example, the effect of mixing time at a stir speed of 100 rpm on the settling rate of two different flocculant systems (in 28 wt% solids MFT) is shown in Figure 5. For short mixing times (1 minute), both flocculants exhibit similar levels of dewatering. As the mixing time is increased, the flocculants behave differently. The initial dewatering rate of Flocculant B is decreased as mixing time increases from one to three minutes. For Flocculant A, the initial dewatering rate appears to be relatively independent of mixing time up to three minutes, but the formulation achieves more complete dewatering as mixing time is increased and the flocculant more effectively incorporates into the solution. At higher stir speeds (300 rpm), the dewatering capacity of Flocculant B is severely compromised, even for stir times as brief as 1 minute (Figure 6). The dewatering is also not recovered for longer stir times. Visual observations indicate a collapse in floc structure due to more aggressive mixing, exhibited as a smoother texture. However, the dewatering of Flocculant A is unaffected by the increased stir speed until the sample is stirred for the longest time (3 minutes). Even under these conditions, Flocculant A dewaters appreciably more than Flocculant B. Additionally, the higher mixing speed appears to improve the initial dewatering of Flocculant A compared to a slower stir speed (compare Figure 6 and Figure 5). While larger scale mixing studies and rheological tests are needed for confirmation, this data suggests that the shear stability of Flocculant A is superior to Flocculant B.

It is also of interest to understand any deleterious effect of the ionic content of process recycle water used to formulate the flocculant solution on-site.

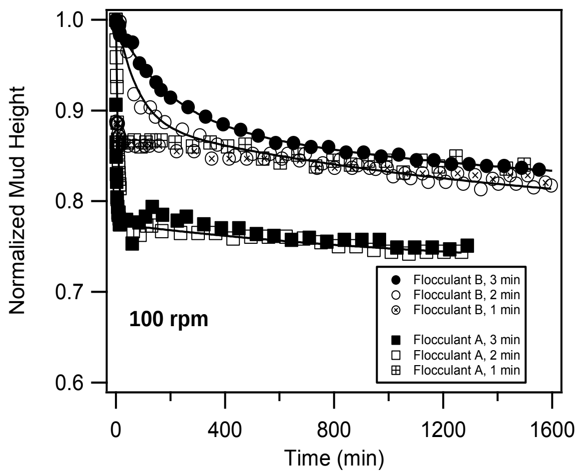


Figure 5. **Effect of mixing time at 100 rpm on settling rate for two different flocculants in 28 wt% solids MFT. Flocculant A shows faster and more complete dewatering than Flocculant B under these conditions.**

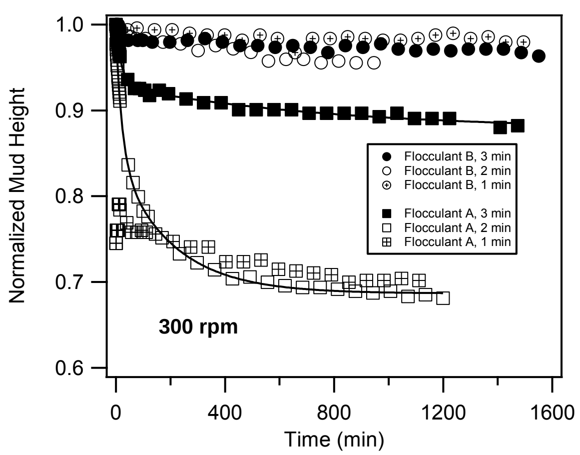


Figure 6. **Effect of mixing time at 300 rpm on settling rate for two different flocculants in 28 wt% solids MFT. The dewatering behavior of Flocculant A is less affected by higher stir speeds than Flocculant B.**

The ionic content of a sample of process recycle water was determined by ICP-MS (Table 1) and is within the ranges found in the field. The settling curves shown in Figure 7 represent the effect of using process water on the performance of two different flocculants. The dewatering rate of one of the flocculants (Flocculant B) is adversely affected by the higher ionic strength of the process water, while that of the second flocculant (Flocculant A) appears to be unaffected. This data suggests that Flocculant A may be more robust to the ionic content of the water under the conditions tested.

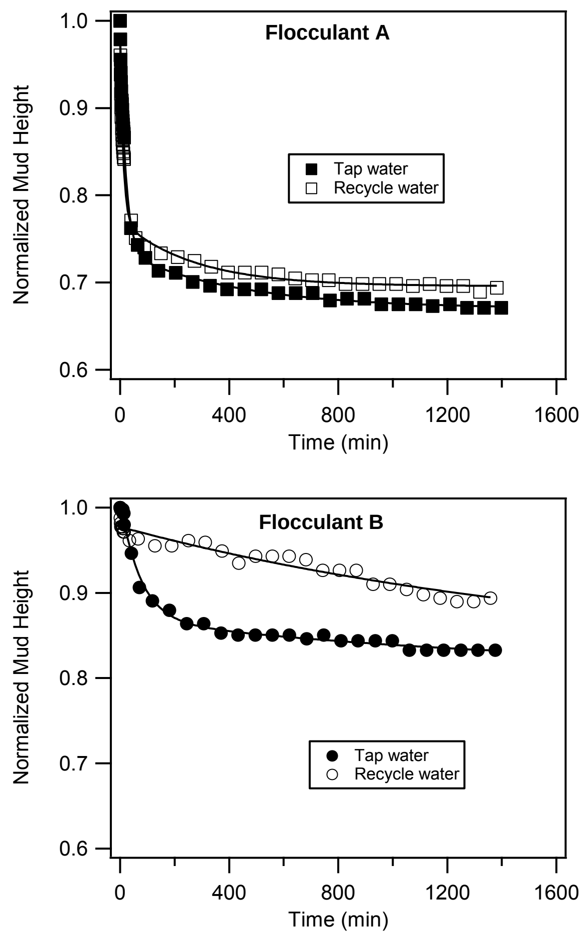


Figure 7. **Effect of recycle process water in flocculant makeup on the settling rate of two different flocculants in 28 wt% solids MFT. The high ionic content in the recycle water decreases the dewatering rate of Flocculant B, while that of Flocculant A is relatively unaffected**.

CONCLUSIONS

A new high-throughput workflow has been developed that is capable of rapidly determining settling rates of flocculant formulations in high solids oil sands tailings. High quality digital images can be collected over a user-controlled period of time, which provides not only settling rate information but gives a unique window on the flocculation process with the ability to quantitatively and visually characterize the flocculated sample. A large number of variables controlling flocculation can be assessed with this workflow such as molecular composition, molecular weight, and flocculant dosage. The ability to control the addition of various components and their mixing conditions not only improves the reproducibility of the flocculation preparation but also allows initial information to be collected on the effect of hydrodynamic conditions on the stability of the floc, a critical issue in the field. The application of this workflow greatly accelerates the identification and development of new, more effective flocculants for oil sands tailings. These methods are shown to be able to distinguish differences in settling performance between potential flocculants, and are currently being applied to screen multiple chemistries. The performance of flocculants which pass this small-scale screening are in the process of being verified in larger scale tests.

Table 1. Ionic content of deionized, tap and recycle process water determined by ICP-MS. (ND <0.1 μg/g).

Element	DI	Tap water	MFT #2 recycle
	μg/g	μg/g	μg/g
Calcium	ND	14.3	23.4
Iron	ND	ND	12.7
Magnesium	ND	8.4	16.5
Sodium	2.6	5.2	755.5
Phosphorus	ND	ND	0.3
Sulfur	ND	6.1	152.3
Potassium	ND	0.8	26.1

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CONSOLIDATION OF OIL SANDS PROCESS TAILINGS BY ENHANCED COAGULATION – FLOCCULATION

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ABSTRACT

One of the major operational and environmental challenges facing oil sands mining is the separation of water from the fine tailings to strengthen the deposits so they can be reclaimed. The directive 74 released in February 2009 urges oil sand producers to accelerate the pace of developing tailings technologies. With no unique and acceptable solution yet in sight, research is now focusing on schemes which utilize more than one technology and combining them into a solution package which is both technically and economically viable. In this work, flocculation of oil sands process tailings with combinations of inorganic coagulants and organic flocculants was investigated.

Process fluid tailings were obtained from hot water extraction of low and high grade ores using a laboratory hydrotransport loop. The effect of the size distributions on stratification after centrifuging with and without the addition of polymeric flocculants were examined, while mechanical properties were determined rheologically. Effects of charge density and molecular weight of organic polymers in the presence of inorganic coagulants were also studied. Results from settling and dewatering of tailings indicate some advantages of dual inorganic-organic treatments over traditional polymer flocculation methods. A synergistic effect due to ionic and electrostatic interactions between inorganic-organic moieties was found to have a marked effect on yielding enhanced coagulation – flocculation. This approach was found to be relatively less-sensitive to water chemistry and the mechanics of solid separations. Based on the results, an overall mechanistic picture of enhanced coagulation – flocculation is proposed.

INTRODUCTION

Background

Solid-liquid separation of finely divided suspensions often requires effective chemical treatments to enhance settling rates and to tailor the morphology of the settled bed to result in adequate dewatering rates and drying. This is especially true under circumstances where the settling and dewatering is driven mainly by gravity, as is the common practice in oil sand tailings management. Other separation technologies such as hydro-cyclone treatment, centrifuging, and filter press require applying high mechanical force to extract fluids from solids. Due to diversity and complexity of oil sands tailings, no physical method is capable to provide adequate separation without adoption of proper chemical aids. Jeeravipoolvarn et al. (2009) studied consolidation of oil sands tailings in 10 m standpipe tests over a long-term since 1982. The settlement of the tailings containing more than 90% silts and clays (< 45 micron) has reached approximately 3.2 m and has settled slowly at a uniform rate during the last 15 years. The average solid content has increased from 30.6% to 41.8% in 25.7 years.

Chemical treatment can include the addition of simple salts, pH modifiers, coagulants, flocculants, or combinations thereof. Such treatments are designed to form aggregates by destabilizing suspended fine solids through either a process of coagulation, flocculation, or both. As always, water chemistry is an important factor in the action of these chemicals by virtue of its affect on speciation of the additives as well as the solid surfaces.

Dewatering of oil sand tailings using pretreatment with inorganic electrolytes (Ca^{2+} and Mg^{2+}) followed by flocculation by an anionic polyacrylamide (PAM) was first described by Rao (1980). It was found that dewatering of a thin layer of the flocculated sludge by drainage on a sand bed or on a wire mesh can result in solid beds with less than 40% moisture. Rao concluded that the role of Ca^{2+} to obtain the best results using an anionic PAM is apparent.

Hydrogen bonding is the main driving force for PAM polymers to adsorb on clay surface. For hydrolyzed PAM, it is therefore not sufficient to enhance their affinity on clay surface by adjusting molecular properties as charge density and molecular weight. New functional groups must be integrated into the PAM molecular structure. These groups should provide a stronger binding with clay surface than hydrogen bonding, thus being able to enhance their adsorption on clay surface. Hybrid inorganic-organic flocculants such as Al-PAM has shown some promise for tailing treatments (Li *et al.*, 2008). Preparation, stability and performance of hybrid polymers remain to be verified at industrial scale.

Our main purpose in the study reported here was to gain some insight into features of settling of oil sand fine fluid tailings prepared using enhanced coagulation – flocculation treatments. A blend of fine solids derived from the extraction process of low grade oil sands ores was used as the substrate in order to ensure a high level of fines including silts and clays (D_{90} of 45 microns and D_{50} of 10 microns). The mixture contained less than 1% (by weight of solids) of toluene soluble organics.

Suspensions of this mixture were treated with a variety of coagulants and flocculants, and allowed to settle in cylinders at room temperature or centrifuged for 10 min at 670 G-force. Samples from various settled beds were excised and analyzed. In addition to using analysis methods commonly employed in the oil sands industry (e.g. solids content and dynamic rheometry), we utilized techniques from other disciplines wherein beds of aggregated solids were also studied. These methods included zeta potential to evaluate particle surface charge, static particle size measurement and steady-state floc size distributions as a function of mixing intensity to evaluate aggregate/floc strength.

EXPERIMENTAL

Materials

Low grade oil sands ore samples were obtained from Alberta Innovates – Technology Futures sample bank in Edmonton, Canada. The sample composition was determined as being: bitumen (6.7%), connate water (8.0%), solids (85.3%) (including 31.3% fines). Hot water extractions were conducted in a laboratory scale hydrotransport loop a under semi-batch conditions (batch water, continuous air) as previously described (Mahmoudkhani, et al., 2012). An inorganic polymer was used as a process aid prior to the addition of ore and pH of the medium was monitored during flotation process. The slurry was pre-conditioned for 5 min before air bubble flotation using an airflow rate of 150 ml/min. Then, froths were collected at time intervals up to 60 min after flotation. Fine fluid tailings (FFT) were separated from coarse solids by wet sieving on a 200 micron Mesh sieve. The pH was adjusted by sparging CO_2 gas to about 7.0.

Solids content is commonly used in the oil sands industry to express the solids-water composition of tailings. For geotechnical purposes, solids content is defined as the mass of tailing solids divided by the total mass of the tailings. In this work, solids content was determined by oven drying samples of fine tailings at 110 °C for 3 hours. Solids content of FFT was measured to be at 9.0 ± 0.2%. Settling tests were performed in 100-mL graduated cylinders (Figure 1). Over a 24 hours period the fines settled uniformly at 40 mL interface, indicating a relatively fast settling rate in the absence of coarse particles.

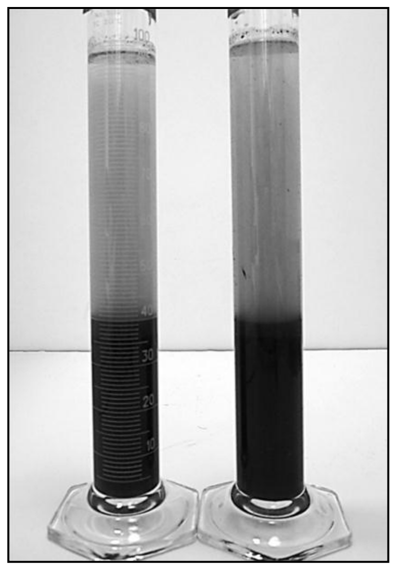

Figure 1. Settling of FFT sample over 24 hours in 100-mL cylinders with no tailing chemical treatment.

Chemical treatments included:

1) Calcium chloride dihydrate (Aldrich; ACS reagent grade)
2) Polyaluminum chloride (Kemira; coagulant grade)
3) Polymer A (Kemira; flocculant grade polyacrylamide, no charge)
4) Polymer B (Kemira; flocculant grade anionic polyacrylamide, low charge)
5) Polymer C (Kemira; flocculant grade anionic polyacrylamide, medium charge)

The flocculant grade water-soluble polymers were of medium to high average molecular weights. Treatment chemicals were mixed into the tailings slurry using a perforated disk mounted on a steel rod. Three transits of the disk (up and down) through the slurry were sufficient for mixing. Solids separation was also performed by centrifuging 50 mL of slurry at G-force 670 for 10 min. After decanting the supernatant, samples of settled bed were analyzed for solid content, PSD and rheology.

Dynamic Particle Size Determination

Particle size distributions (PSD) were determined with a Malvern Mastersizer S, which measures the angular dependence of scattered light (mainly in the forward direction). FFT or paste samples were dispersed in the impeller driven flow loop of the Mastersizer. The flow loop contained Milli-Q water without additional chemical dispersants. Particle size distributions were computed as equivalent-sphere size distributions based on Mie scattering and Fraunhofer diffraction formalisms applied to the scattering data. The optical constants (λ = 432.8 nm) used for these computations were a complex refractive index of 1.5295, 0.100i for the solids and a real refractive index of 1.3300 for the continuous phase.

Measurements were taken during each test at 30, 60, 120, 240, and 360 seconds after addition of the solids to the diluent under continuous agitation at a single impeller speed. This procedure allowed us to assess whether a steady-state size distribution had been achieved. Typically we used the size distributions after 120 sec of mixing for comparisons, since steady state was achieved by this time. By using this procedure at different impeller speeds and comparing steady-state distributions at these various impeller speeds we were able to assess the strength of aggregates.

Zeta Potential and Static Particle Size Measurements

Zeta potentials of the kaolinite particles were measured by a Brookhaven ZetaPALS at 22°C. To measure the particle electrophoretic mobility, this instrument uses a phase analysis light scattering (PALS) based on the shifted frequency spectrum. Square cuvettes of 10 mm (Brookhaven

Instruments Corporation BI-SCP) with a sample holding capacity of 4,5 ml were used to hold the samples. The kaolinite colloidal suspensions were then illuminated by a cross focused laser beam. Zeta potential for each sample was determined by taking the average of 10 runs (with 10 cycles for each run). The standard deviation for the 10 runs was less than ±2 mV. For each zeta potential measurement, five mL stock suspension was diluted with 50 mL 0.005 mol/L NaCl. The suspension was conditioned by a magnetic stirrer for 10 min during which calcium ion was added and pH adjusted as required by HCl or NaOH. About 1.6 mL of the conditioned sample was transferred to the sample cuvette for zeta potential measurement.

Static particle size analysis of kaolinite samples was measured by a dynamic light scattering method based on the particle size option in the ZetaPALS instrument. DLS measures the intensity fluctuations of scattered light arising from Brownian motion of suspended particles and calculate the size. The wavelength of the incident laser beam (λ) was 660 nm. The scattered intensity was set at a scattering angle of 90° and the temperature of 22°C. Time-averaged particle size distributions were collected over an analysis period of at least 10 min at room temperature. Five separate measurements were acquired for each freshly prepared suspension of kaolinite in water. The autocorrelation functions were deconvoluted using the built-in non-negatively constrained least squares-multiple pass (NNLS) algorithm in order to obtain particle size distribution. Lower and upper limit of this DLS method is typically 2 nm and 10,000 nm, respectively. However the upper limit is set by the onset of sedimentation and therefore is sample dependent.

Dynamic Rheometry

Rheological investigations were conducted at 22°C on an Anton-Paar MCR 300 rheometer equipped with a six-bladed Anton Paar ST 22-6V-16/106 spindle. The supernatant is decanted from the 50 mL centrifuge tube, and the 50-mL tube is placed into the rheometer sample holder. The centrifuge tube is clamped in place to make sure it doesn't spin during the testing. The spindle is lowered until it is 1 mm above where the taper starts in the centrifuge tube. The rotational shear stress is linearly ramped from 0 to 250 Pa at a rate of 1 Pa/s. Past 250 Pa, the shear stress is logarithmically ramped at a rate of 1 order of magnitude every 10 seconds. The test ends when the shear stress reaches 25,000 Pa or the shear rate exceeds 80 s^{-1}. As the shear stress is ramped, the strain is measured. The resulting graph looks like a tensile testing machine's stress-strain curve, except the axes are flipped. The yield stress is calculated as the first inflection point along the strain-stress curve.

RESULTS

Aggregation of Kaolinite in the Presence of Calcium Ions

It is well documented that coagulation decreases and neutralizes the surface charge of suspended particles by using inorganic salts. Zeta potentials were measured for Kaolinite powder (Ward's Natural Science, < 45 micron) in sodium chloride solution (5 mM) and upon addition of calcium chloride solution (10 mM) at pH ranging from 3 to 10. Results are summarized in Figure 2.

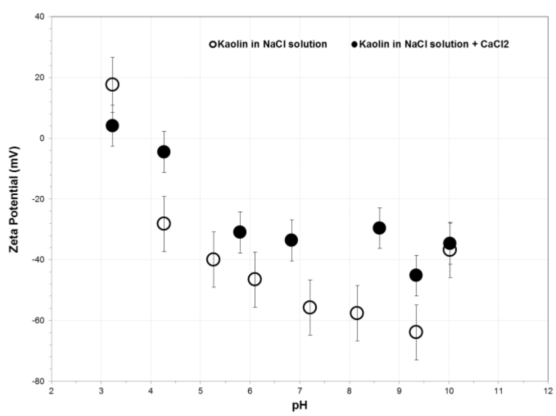

Figure 2. Zeta potential of Kaolinite clay as a function of pH with and without calcium ions.

Particle size distribution of the coagulated samples was also measured on ZetaPALS under static condition. As shown in Figure 3, larger aggregates are formed upon addition of calcium ions and at pH 7 or lower.

Coagulation–Flocculation of Fine Fluid Tailings

Coagulation-flocculation of colloidal suspensions has been an important process in industrial wastewater treatment. Coagulation decreases and neutralizes the surface charge of suspended particles by using inorganic salts such as aluminum salts. Flocculation plays a major role in aquatic environments by bridging the aggregates to form larger agglomerates in the presence of polymers. Particle size distribution of fine fluid tailings used for treatments is shown in Figure 4.

Treatment doses were chosen based on experience with tailings of this nature and some preliminary screening data. These doses were:

1) Calcium chloride dihydrate: 500, 1000, 2000 and 3000 g/dry T
2) Polyaluminum chloride: 500, 1000, 2000 and 3000 g/dry T
3) Polymer A: 1000 g/dry T
4) Polymer B: 1000 g/dry T
5) Polymer C: 1000 g/dry T
6) Dual: mixed inorganic coagulant (calcium or aluminum) @ 500,1000 & 1500 g/dry T and polymer A, B or C @ 1000 g/dry T

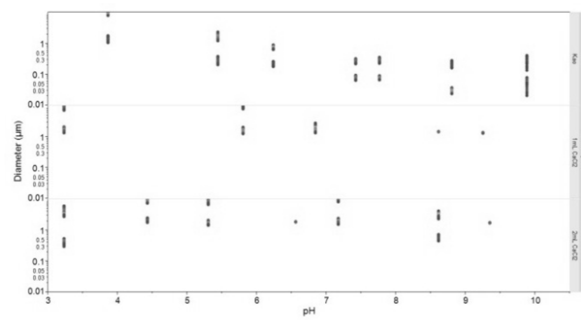

Figure 3. Particle size distribution as measured by ZetaPALS on suspended Kaolinite fines in the presence of calcium ions as a function of pH.

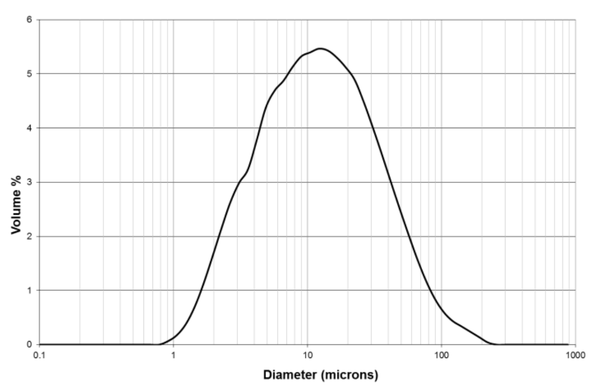

Figure 4. Particle size distribution of fine fluid tailings separated from extraction of low grade oil sand ores using hydrotransport loop.

All doses were given based on the weight of solids in FFT. After mixing the FFT tailings and chemicals the settled bed typically formed from the top down; i.e. the column of settling slurry formed a discernable interface with the clarified volume above it and this interface dropped with time, finally forming a compact bed. Accelerated tailing settling was achieved by centrifuging 50 mL of slurry at G-force 670 for a period of 10 min.

FFT Treatment by Coagulants

Figures 5 and 6 below illustrated tailing settling when the sample was treated by various dosages of calcium or aluminum coagulants. Solid contents in the supernatants were measured.

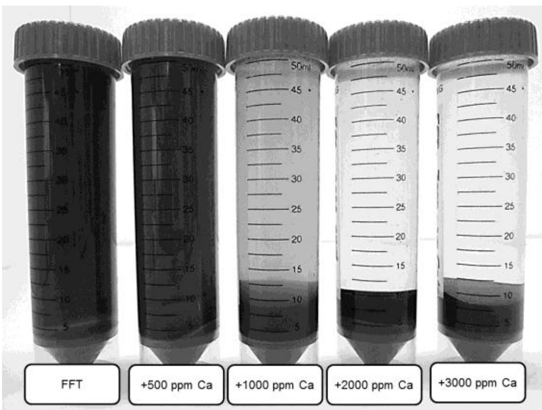

Figure 5. Settling and turbidity of FFT samples treated by calcium coagulant.

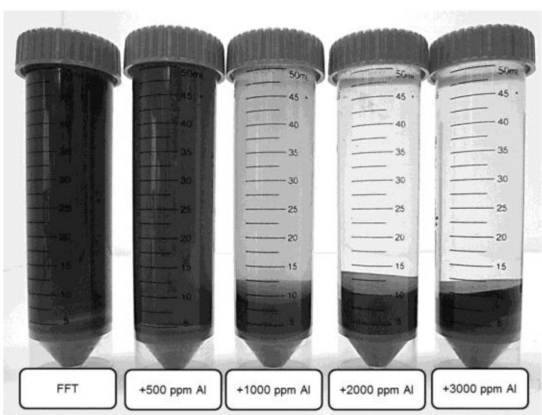

Figure 6. Settling and turbidity of FFT samples treated by aluminum coagulant.

The settled beds produced from tailings treatments with either calcium, or aluminum coagulants had very similar compaction rates and solids content of supernatant. In fact, coagulation resulted in a compaction rate independent of the type of coagulant. However, residual solids in the supernatant decreases (see Figure 7) as concentration of coagulant increases and water clarity can be obtained using treatments higher than 1000 ppm.

The addition of divalent cations is known to increase settling rates, but not bed densities, in kaolinite dispersions. Using cryo-SEM imaging, Zbik et al (2008) have shown that in the presence of Ca(II) cations, the 2–5 μm kaolinite aggregates consist of individual particles aggregated mostly in a EF and EE orientations. This feature is illustrated in Figure 8.

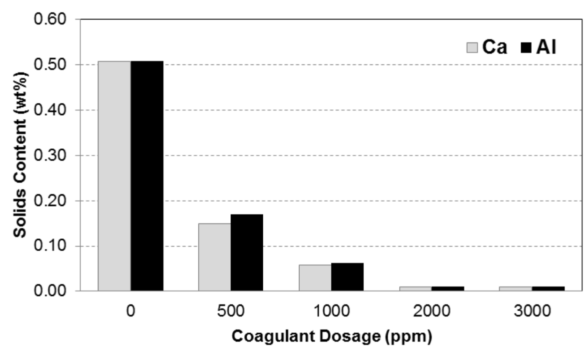

Figure 7. Solid content of supernatant from coagulation of FFT samples.

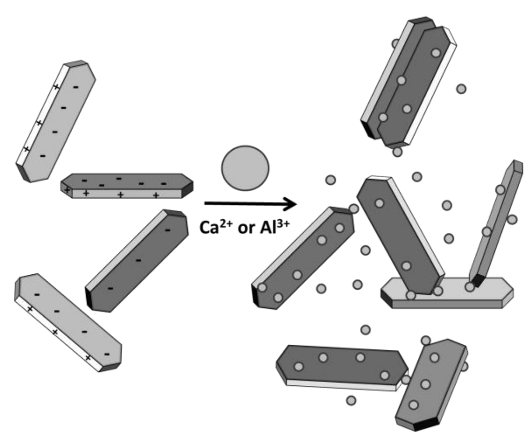

Figure 8. Schematic representation of EE and EF aggregations of kaolinite in the presence of Ca^{2+} or Al^{3+}.

Larger aggregates are also forming in dynamic condition (shear rate 1600 rpm) on a Mastersizer S. Subsequent additions of 10 mM calcium chloride solution to FFT shifts particle size distribution from < 10 micron to particles as large as 80 microns as shown in Figure 9.

FFT Treatment by Flocculants

Figure 10 illustrates tailing settling when sample was treated by various polymer flocculants at 1000 ppm dosage.

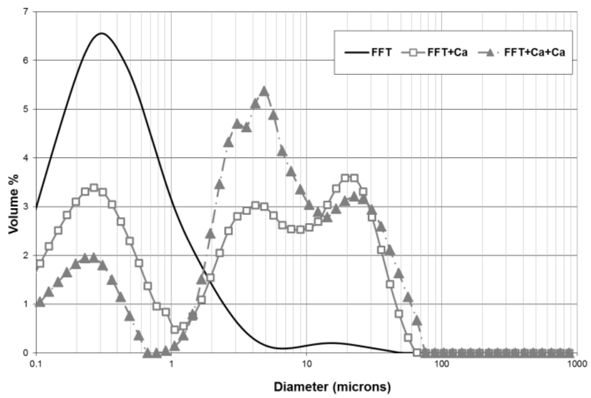

Figure 9. Formation of larger aggregates as a function of calcium ions to FFT.

Figure 10. Settling and turbidity of FFT samples treated by various polymer flocculants.

The settled beds produced from tailings treatments with flocculants A, B & C had very similar compaction rates, but solids content of supernatant (see Figure 11) decreases as polymer charge increases. For treatment with 1000 ppm of polymer C, supernatant clarity is equivalent to the treatment by 500 ppm of either calcium or aluminum coagulants. Polymer A (no charge) has no effect on supernatant clarity.

FFT Treatment by Coagulation-Flocculation

Figure 12 illustrates tailings settling when sample was treated by various polymer flocculants at 1000 ppm dosage after pre-treatment by 500 ppm of inorganic coagulant.

As shown in Figure 13, supernatant clarity has been significantly improved over treatment by flocculants only and solid content in supernatant has been decreased accordingly. Further improvement can be obtained at higher dosages of coagulant and flocculants as shown in Figures 14 & 15.

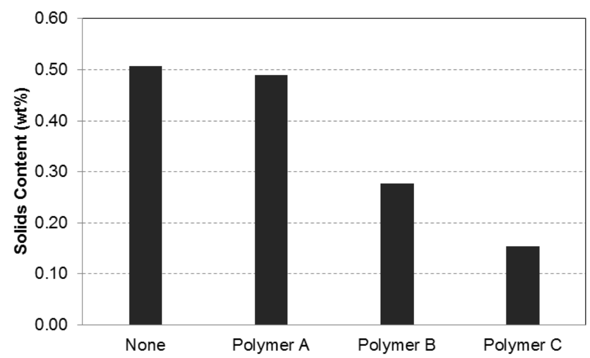

Figure 11. Solid content of supernatant from flocculation of FFT samples.

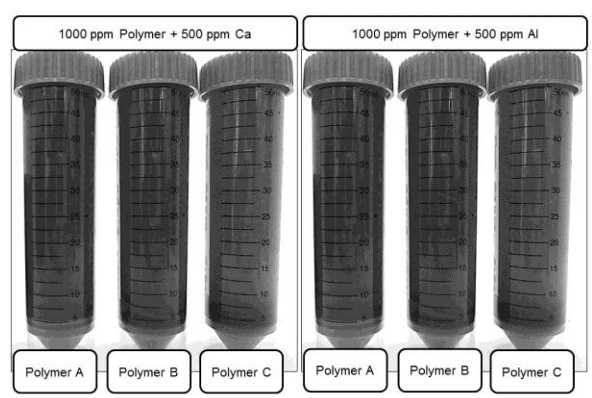

Figure 12. Settling and turbidity of FFT samples treated by coagulation – flocculation (500/1000 ppm).

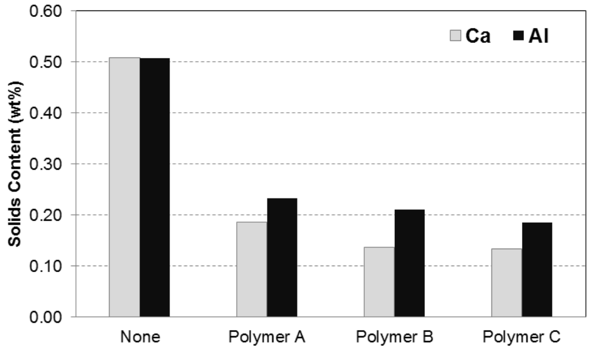

Figure 13. Solid content of supernatant from coagulation (500 ppm) - flocculation (1000 ppm) of FFT samples.

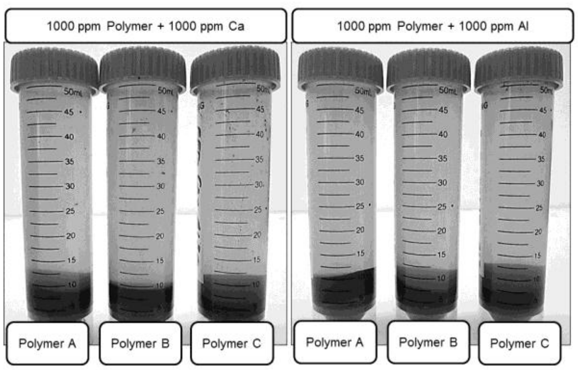

Figure 14. Settling and turbidity of FFT samples treated by coagulation – flocculation (1000/1000 ppm).

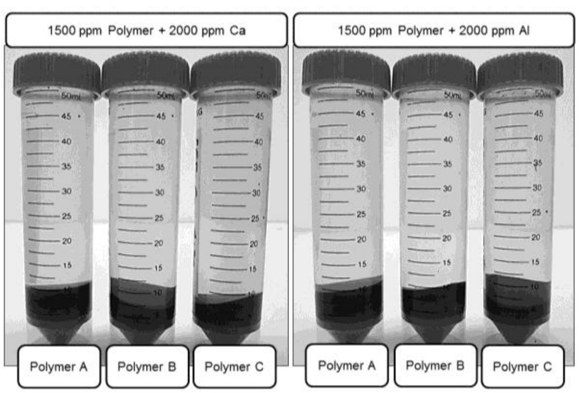

Figure 15. Settling and turbidity of FFT samples treated by coagulation – flocculation (2000/1500 ppm).

Floc Formation and Breakage

Steady-state particle size distributions under fixed mixing conditions have been used to assess the floc formation and breakage pattern of flocculated solids, mainly in wastewater treatment applications (Jarvis, et al, 2005). We have applied this method to flocculated oil sand tailings in the impeller-type mixer that is a component of the Malvern Mastersizer S. A 50 mL sample of FFT was centrifuged for 10 min at G-force 670 and supernatant was separated from settled bed. It was then diluted directly in the Malvern mixer. This pre-fractionation was done in order to have better definition of the size distributions in the sub-10-micron range. Note that the amounts of material added to the Malvern mixer result in very high dilution ratios (i.e. a few tenths of a gram of solids in 100 mL of water). Steady-state particle size distributions as determined at an impeller speed of 1600 rpm upon addition of coagulants and flocculants are shown in the plots below for visual clarity (Figures 16-21).

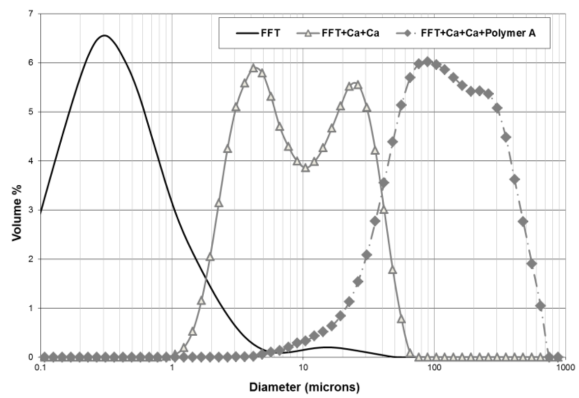

Figure 16. Floc formation by coagulation with Ca followed by flocculation with polymer A.

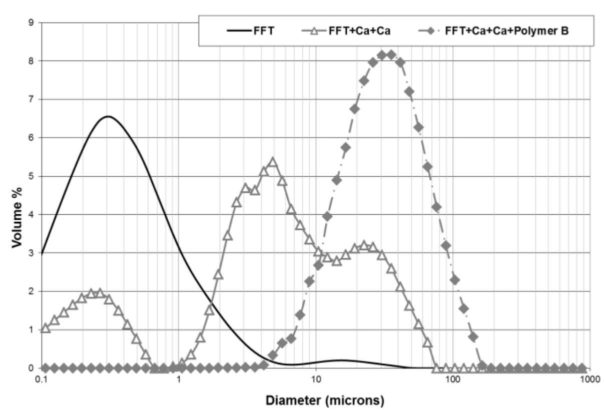

Figure 17. Floc formation by coagulation with Ca followed by flocculation with polymer B.

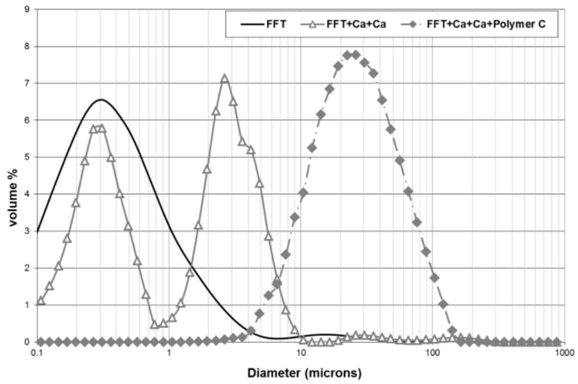

Figure18. Floc formation by coagulation with Ca followed by flocculation with polymer C.

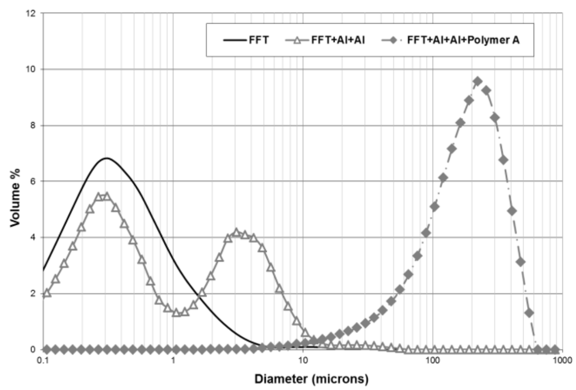

Figure 19. Floc formation by coagulation with Al followed by flocculation with polymer A.

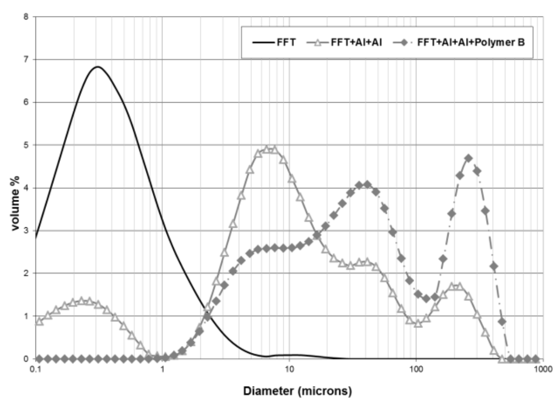

Figure 20. Floc formation by coagulation with Al followed by flocculation with polymer B.

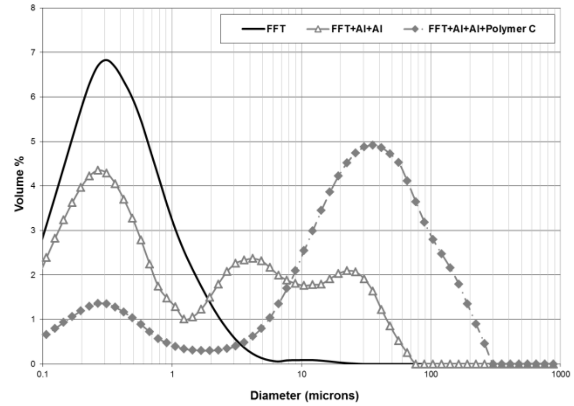

Figure 21. Floc formation by coagulation with Al followed by flocculation with polymer C.

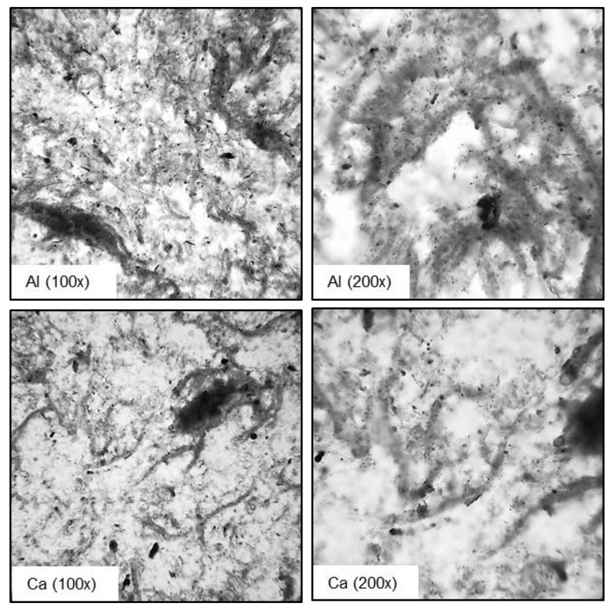

Figure 22. Optical photomicrograph of flocculated tailings fines treated by Ca or Al coagulant and polymer C.

Aggregation and floc formation was most clearly evident in the tailings treated with either inorganic coagulant or the mixed coagulant and flocculant (dual treatment). In dynamic conditions of the experiment where colloid sample flows through the cell for DLS measurement, higher concentrations of coagulant is required for aggregation of fine solids. The shift from median size of 0.1 – 1 microns to larger particles of up to 10 microns can be attributed to formation of EF & EE aggregates in the presence of either Ca or Al cations. Upon addition of flocculant, larger flocs are formed in the range of 10 to 100 microns.

Optical microscopy revealed that flocs for both calcium and aluminum coagulants followed by polymer C have similar structural characteristics. As shown in Figure 22, in the presence of calcium or aluminum ions, the aggregates cross-linked by polymer chains to form larger flocs, corresponding to the shift in size distribution from sub-10 microns to 10 – 400 microns range. The porous structure consists of pores which are related to volumes between micro-aggregates and larger inter-aggregate channels, providing channels for dewatering. The conclusions discussed above may be put in the form of a simple cartoon (see Figure 23) that captures the essentials of our hypotheses regarding the networks formed from dual chemical treated tailings.

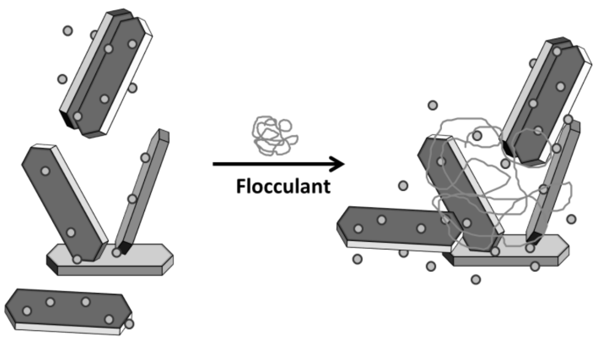

Figure 23. Schematic representation of our hypothesis for floc network.

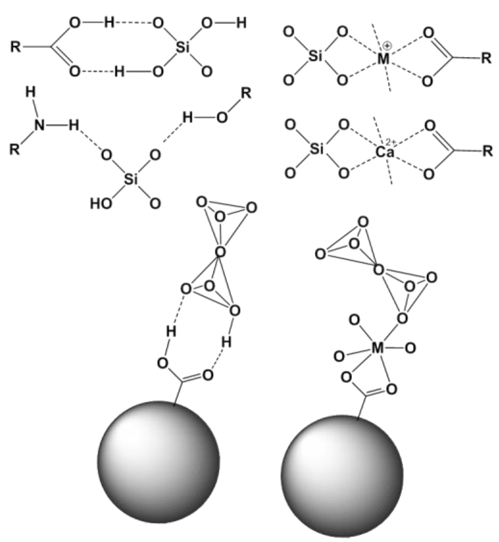

Figure 24. Possible ionic and H-bonding interactions between clay's surface, coagulant ion and polymer functional moieties.

Network stability arises from inter-particle interaction in the forms of hydrogen bonds and bridging due to chelation of calcium or aluminum ions by carboxylate groups or silicates as schematically presented in Figure 24.

We also investigated floc fragmentation effects for dual treatment systems. Treated tailings samples were all sized under steady-state conditions at 1600 rpm impeller speed. The speed was increased to 3500 rpm and PSD was measured at the steady state. Comparison of distribution profile before and after impeller ramp provides an evidence for floc stability. A shift towards smaller particle size can be attributed to floc breakage. As shown in Figures 25 – 27, flocs from Ca + polymer treatments exhibit good stability in the order of polymer C < polymer B < polymer A.

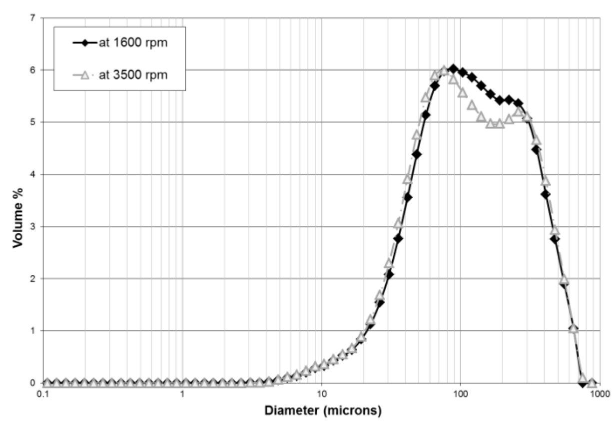

Figure 25. Floc stability and breakage for FFT treated by Ca and polymer A.

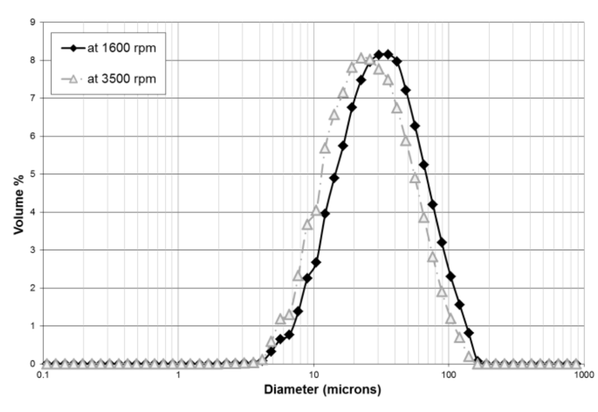

Figure 26. Floc stability and breakage for FFT treated by Ca and polymer B.

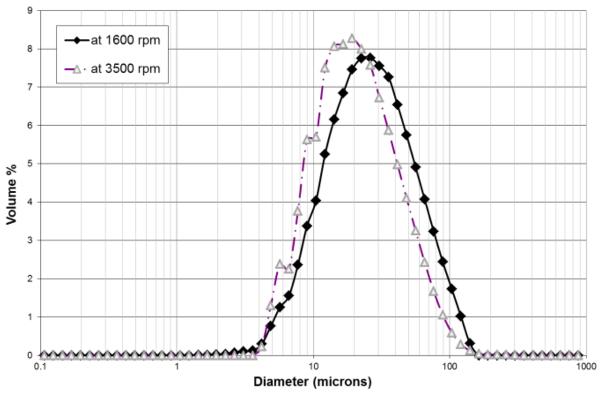

Figure 27. Floc stability and breakage for FFT treated by Ca and polymer C.

Similar studies on Al coagulant + polymer treatments exhibit slightly less stability of floc networks. See Figure 28 for floc breakage of FFT treated by Al and polymer A.

Dynamic Rheometry

Rheological investigations were conducted at 22 °C on an Anton-Paar MCR 300 rheometer. Some insight on the microstructure of networks of flocculated particles may be acquired from their rheological behavior. Solid content of settled beds were also measured. Results are presented in Figure 29, where correlations of solids and yield stress of the settled beds with coagulant's and flocculant's charge and dosage are graphically demonstrated. It really does seem that the coagulant dose is the key factor to obtain high yield stress of settled bed. Flocculant charge may be the second most important factor. The other factors, namely coagulant charge, flocculant charge and molecular weight don't help as much in the supernatant, but come into play with the flocculated bed.

DISCUSSION

Aggregate formation not only affects settling rate and completeness of separation, but it also governs the structure of the settled bed, which in turn affects dewatering and compressibility of the network of particles. These features impact the ultimate material properties of the dewatered solids and the rate at which those final properties are attained. Thus to obtain appropriate mechanical properties of the final paste, the structure of the network of aggregated particles must be properly engineered. Aggregation and settling of fine fluid tailing suspensions (particles less than 10 micron) is a complex and very slow process, mainly due to large surface charge and inter-particle repulsions. Such behavior might reasonably result in stratified properties within the settled bed, which will impact dewatering behavior. It is well known that the fine particles are difficult to dewater using the existing dewatering techniques, such as vacuum, pressure, and centrifugal filtrations.

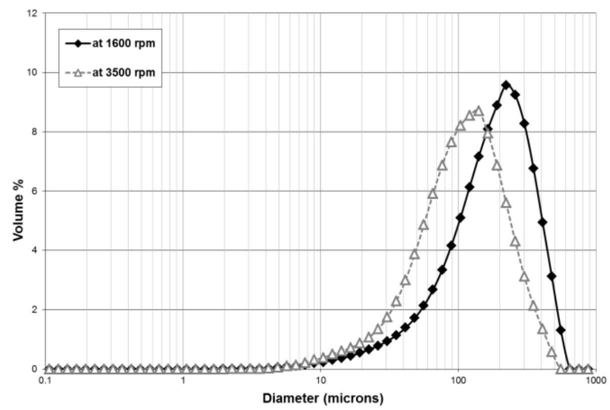

Figure 28. Floc stability and breakage for FFT treated by Al and polymer A.

Reduction of electrical double layer repulsions and surface charges can activate particles aggregation. In the kaolinite fines, we have shown that decreasing pH or addition of di- or multivalent cations greatly enhances inter-particle interactions. Oil sands tailings are often alkaline by nature that is originated from caustic and other process aids used to improve bitumen recoveries. Acidification by injection of CO_2 is a viable and safe option and is currently being practiced in oil sands industry. Adsorption of carbon dioxide by tailings can also reduce GHG emissions. Coagulation decreases and neutralizes the surface charge of suspended particles by adsorption of cations and their complexes.

Enhanced coagulation can be defined here as the addition of excess coagulant for the improved floc stability upon treatment of oil sands tailings by flocculation and enhanced mechanical integrity of settled beds for optimal dewatering.

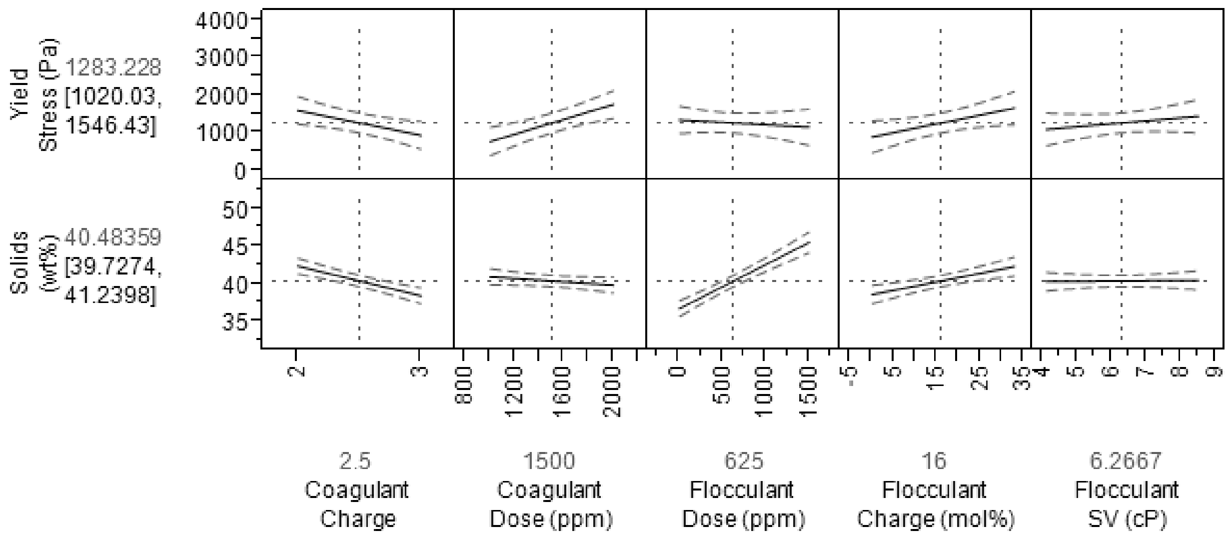

Figure 29. Correlations of solids and yield stress of settled bed with coagulant's and flocculant's charge and dosage.

Tailings treated with only coagulants settled fast into a bed that had a similar compaction profile and low solids in supernatants; however, the dewaterability was only slightly improved. This might be expected from the application of a coagulant that compressed the electrical double layers in the electrostatically stabilized fine particle fraction, thereby creating very small aggregates from the fines. This theory is supported by the PSD data as a function of coagulant dosage as measured on the ZetaPALS instrument.

Treatment of the tailings with enhanced coagulation – flocculation produced substantially different settled beds compared to those made from the coagulant-treated tailings. All polymers tended to result in a same dense settled bed; more so with polymer C than with polymer A or B. Dewatering was also significantly improved in all cases; although more so with polymer C. The coagulated – flocculated tailings tended to be more shear stable than both coagulated and flocculated ones. All of these behaviors are consistent with aggregate formation due to increase inter-particle interactions mediated by the presence of calcium or aluminum ions.

Compaction of solids content and dewaterability in these settled networks was quantified by centrifugation. This allowed us to determine treatments where this compaction was mitigated. Optimum treatments where the settled bed has high solids, high yield stress, good dewaterability and minimal suspended solids in supernatant can and were identified using this system of metrics. We have shown that several simple measures of chemically treated oil sand tailings, some of which have not traditionally been used in oil sands research, can yield insight into the networks formed in settled beds.

ACKNOWLEDGEMENTS

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A STUDY OF ENVIRONMENTALLY FRIENDLY CHEMICAL AIDS FOR CONSOLIDATION OF OIL SAND TAILINGS

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ABSTRACT

A primary challenge in oil sands processing is the consolidation of fine solids that are < 45 µm in diameter and do not settle readily over time. Several strategies are currently being employed for post-treatment of oil sand tailings after they are produced, those mainly consist of chemical flocculation of fine fluid or mature fine tailings. Adding a chemical aid during extraction process that accelerates tailings settling and facilitates consolidation process, would lower operating expenses, and reduces environmental risks and hazards associated with tailings ponds maintenance. An efficient and fast settling process may also reduce the amount of heat lost during the sedimentation phase, allowing for warmer water to be recycled and conserving energy in the oil sands water cycle.

Biodegradable low molecular weight organic process aids can fulfill such a role for both high- and low-grade oil sands. A laboratory-scale hydrotransport (HT) loop (Wallwork et al, 2004) and Denver flotation cell were utilized to evaluate the effects of chemical dosage and operating parameters on processability of Athabasca oil sands samples. Process tailings were examined for properties such as suspended solids, settling rate and ease of tailings treatment. The results demonstrate that adding the organic aids during or at the end of the froth processing improved tailings settling rates and facilitated treatment without hindering bitumen recovery. These chemistries can be easily incorporated into current processing facilities and deliver economic and environmental benefits.

INTRODUCTION

In oil sands surface mining, the conventional process of extracting bitumen involves slurrying the oil sand ore with warm or hot water to release bitumen from sand grains (Masliyah et al, 2004). The bitumen is then free to attach to air bubbles so it can float to the surface and be collected as a froth consisting of bitumen, water, and fine (< 45 µm) solids in a series of recovery steps. The remaining water, fine solids, and coarse solids are transferred to a pond as process tailings. Upon reaching the pond, coarse solids rapidly settle while the fine solids tend to resist settling and dewatering. In the processing of lower-grade oil sands, which have the lowest amount of bitumen and the highest content of fine solids, process aids are commonly added to improve bitumen recovery. The commercially preferred process aid is sodium hydroxide (NaOH). Although NaOH effectively improves bitumen recovery, it negatively affects downstream processes by dispersing the fine solids present in the ore (Miller et al, 2010). This complication necessitates a separate tailings treatment step to re-aggregate the fines so that water can be recycled or discharged.

With the anticipated increase in crude bitumen production from open pit mining over the next 10 years, there is an increasing sense of urgency to process the ore with a lower consumption of water or a more efficient, straightforward recycling process. Since every barrel of bitumen recovered results in two to three barrels of tailings, the industry can significantly benefit from an upfront chemical aid that improves bitumen recovery and/or consolidates the fine clays. The latter can facilitate the process tailings treatment steps resulting in significant cost savings and reduction in environmental and hazards risks over the time.

The main purpose of this work is to demonstrate the use of environmentally friendly chemicals referred to as low molecular weight organics (LMWO) as ore processing aids. The results demonstrate that when LMWO is applied upfront during the bitumen extraction step, no adverse impact on bitumen recovery is observed, while the treatability of the fine solids is significantly improved.

EXPERIMENTAL

Process Aid Assessment

To evaluate the use of LMWO as process aids, a low-grade ore sample was tested using a laboratory-scale Denver flotation cell (Metso Minerals, Danville, PA) under semi-batch conditions (batch water, continuous air). The composition of these ores as determined by Dean-Stark Soxhlet extraction can be found in Table 1.

Table 1. Composition of Low- Grade Oil Sands.

Bitumen	9.03 %
Water	4.10 %
Total Solids	86.87 %
Fines content (<45um)	32.12 %

For each experiment, 300 g of oil sand ore was added to 1.5 L of pre-heated water at 50° C at an impeller speed of 1000 rpm in a 2-L rectangular tank. The flotation tank was kept at 50° C with a hot water circulating bath. Prior to the addition of ore, the process additives were added to the test water. Upon ore addition, the slurry was pre-conditioned for 5 min before aeration began at an airflow rate of 200 mL/min. At 2, 5, 10, 20, and 60 minutes after the start of aeration, the agitation was paused for 30 sec and all froth which had developed was collected in order to simulate both primary and secondary bitumen recoveries. The slurry which remained after froth collection, referred to as the tailings, was transferred to a 2 L graduated cylinder to monitor whether the fine clays settled or remained in suspension. For each froth, the concentrations of bitumen, water, and solids were determined by toluene solvent extraction using standard Dean-Stark Soxhlet extractor units. Triplicate experiments indicated recovery rates being reproducible within ±5%.

Post-Treatment

When processing with LMWO in the Denver cell or HT-loop, the majority of the solids in the slurry were coarse solids, i.e. larger than 45 μm in diameter. In order to more closely observe the interaction of LMWO with fine solids, additional testing was performed by processing a low-grade oil sand in the laboratory scale hydrotransport loop without additional process aids. The slurry remaining after a 60 minute flotation experiment was collected. After allowing the coarsest fraction to settle for approximately 1 minute, the unsettled slurry (supernatant) was collected. These tailings were transferred to 100 or 250mL graduated cylinders to conduct additional settling experiments. LMWO was added to tailings, with the dose based on the amount of ore used in the flotation process, by capping and inverting the graduated cylinder fifteen times, with each inversion cycle lasting approximately one second. The settling was monitored for a minimum of 24 hours and observed for the type of settling and the initial settling rate (ISR). The supernatant and settled beds were sampled and analyzed by additional techniques.

Particle size distributions (PSD) were determined with a Malvern Mastersizer S, which measures the angular dependence of scattered light (mainly in the forward direction). The original and treated tailings were dispersed in the impeller-driven flow loop of the Mastersizer. The flow loop contained deionized water without additional chemical dispersants. Particle size distributions were computed as equivalent-sphere size distributions based on Mie scattering and Fraunhofer diffraction formalisms applied to the scattering data. The optical constants (λ = 432.8 nm) used for these computations were a complex refractive index of 1.5295, 0.100i for the solids and a real refractive index of 1.3300 for the continuous phase. Measurements were taken during each test at 30, 60, and 120 seconds after addition of the solids to the diluent under continuous agitation at a single impeller speed. This procedure allowed us to assess whether a steady-state size distribution had been achieved. Typically the size distributions after 120 sec of mixing were used for comparisons, since steady state was achieved by this time.

Particle zeta potentials were measured using a Brookhaven ZetaPALS equipped with a 659 nm laser. Samples of settled beds were diluted to approximately 0.5 wt% with water from their corresponding supernatants to a volume of 1.5 mL, and then added to 1 cm^2 acrylic cuvettes. Scans were collected every minute for 10 minutes at a temperature of 24.8 °C and were averaged to give the final results.

The ability of flocs formed by the addition of LMWO to reform after being subjected to additional shear was investigated by agitating the flocculated bed once it had initially settled for 24 hours. In the first test, the process of inverting the cylinder was repeated to simulate a low-shear agitation, and the ISR was followed for an additional 24 hours. A second test was then performed by vigorously shaking the sample for 60 seconds to simulate the higher shear conditions which might be encountered during transport in a pipeline to a tailings pond. The settling was then followed for 24 hours.

Combination with Polymer Flocculant

The addition of a high-molecular weight anionic polyacrylamide flocculant was used alone and in combination with LMWO. Others have demonstrated that these types of flocculants can deteriorate the bitumen froth quality (Li et al, 2005), but the addition of LMWO as process aid for ore conditioning, followed by a small amount of flocculant during the transport of the tailings to the settling ponds, could result in improved ISR or final flocculated bed heights. For tests involving both chemicals, LMWO was initially added with the inversion method, immediately followed by the addition of the polymer and a second inversion step.

RESULTS AND DISCUSSION

Bitumen Extraction

The addition of LMWO at a range of doses was applied in the Denver cell tests for ore sample LG1. The results for bitumen recovery compared to a test using the addition of sodium hydroxide to pH 8.5 are shown in Figure 1.

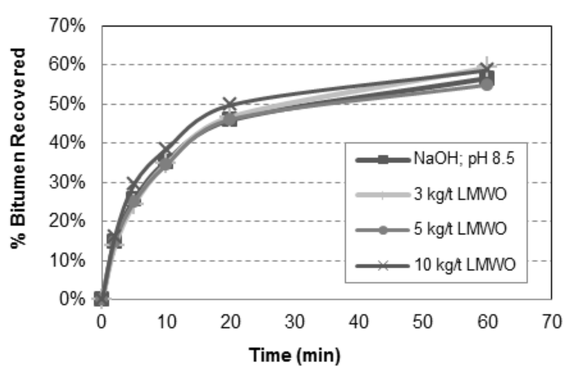

Figure 1. Bitumen recovery vs. time for LG1.

It was observed that the addition of LMWO at each of the concentrations performed similarly to the use of NaOH at a slurry pH of 8.5. This indicated that the upfront addition of LMWO did not interfere with the liberation and aeration of bitumen in a way that would reduce bitumen recovery. The additional impact of the LMWO addition could be seen when observing the tailings after 24 hours, shown in Figure 2.

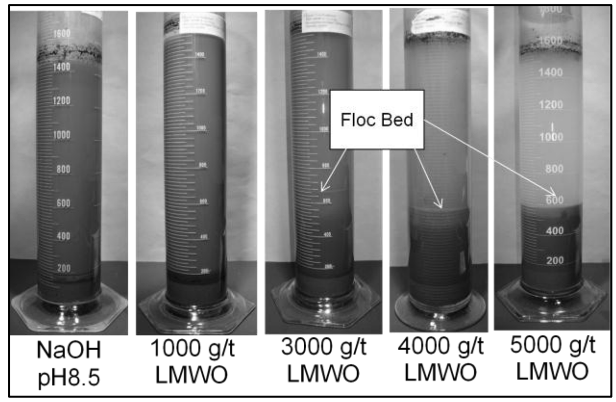

Figure 2, Denver cell tailings settling after 24 hours.

At LMWO concentrations of 3000 g/t and above, the clarity of the supernatant began to improve and a secondary settled bed of fine solids was observed to form on top of the sedimented coarse tailings. At 3000 g/t of LMWO, segregation was also noticed within the fine solids which settled. As the concentration of LMWO increased, the degree of segregation within the fines was reduced and a more uniform settled bed was observed.

Post-Treatment

A more comprehensive range of LMWO concentration was monitored in the post-treatment of process tailings acquired from the HT-loop. The ISR results for these experiments can be found in Figure 3, while the final tailings settling after 24 hours is shown in Figure 4.

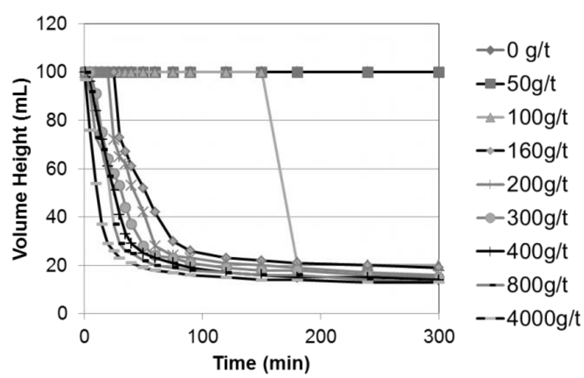

Figure 3. Initial settling rate vs. time.

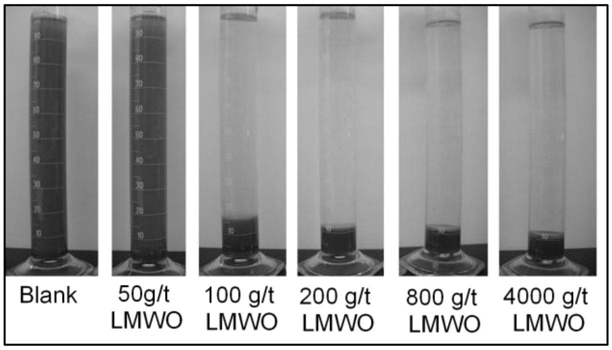

Figure 4. Tailings settling after 24 hours.

At a concentration of 100 g/t, the ISR increased drastically from 150 to 180 min. The increase was due to suspended fines being destabilized and joining the coarser particles in the settling process. This appeared to be a critical concentration of LMWO where the type of sedimentation began to shift. Below 100 g/t of the LMWOs, the sedimentation occurred as a rising settled zone where the solids accumulated on the bottom and began to stack on top of each other. This type of sedimentation was also much slower and was not observed within the initial 3 hours of settling shown in Figure 3, but was noticed and monitored after the cylinders settled overnight. Above this dose, the sedimentation existed as a falling front which collapsed quickly and resulted in a clear supernatant after 24 hours.

The type of settling could also be described by the level of segregation in the settled bed. Observing the images in Figure 5, at the lower concentrations of LMWO where a rising-bed sedimentation dominated, two distinct zones were noticed. Note that even though the coarsest material was removed before this stage of testing, there was still a size fraction which would settle on its own after a few hours, as can be seen in the control experiment. As the concentration of LMWO increased beyond the critical concentration, the stratification was no longer observed and a single, non-segregated bed existed. This was also observed in the Denver cell tailings where higher concentrations of LMWO produced less stratification. Minimizing the amount of segregation can significantly improve the dewatering capability of the settled bed. (Farinato et al. 2010).

For the tests which produced a settled bed after 24 hours, a sample was removed and dried in an oven at 105°C for a minimum of three hours. To prevent the solids which settled naturally from being included in the solids content measurement, the excised sample was taken from the upper portion of the floc bed. The results for the oven solids analysis are shown in Figure 6.

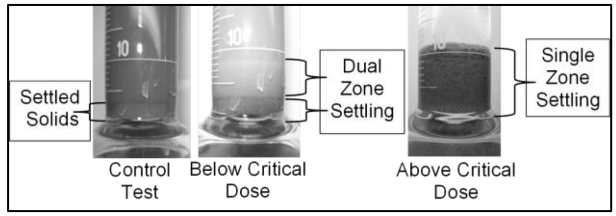

Figure 5. Effect of LMWO concentration on settling type.

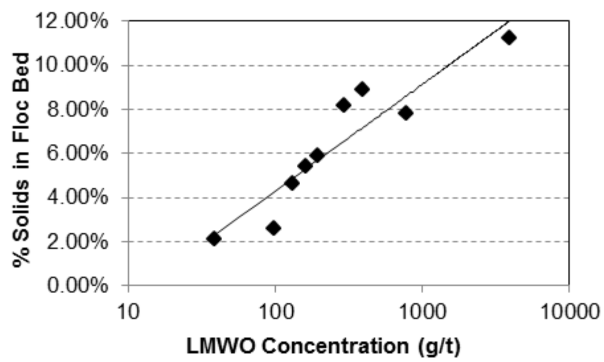

Figure 6. Solid content of flocculated bed as a function of LMWO concentration.

As the amount of LMWO increased, the solids content of the settled bed also increased. At a higher dose, the coarse particles were more evenly distributed throughout the settled bed. This led to more compaction and an aggregate structure which was better able to dewater.

PSDs were determined for the untreated tailings sample as well as several of the settled beds. The results in Figure 7 had shown that the distributions of the settled beds were unchanged from the original tailings.

Aggregates which formed through the addition on LMWO were dispersed by the impeller of the Mastersizer, indicating that they were weakly attracted with a low resistance to shear. A flocculated structure with strong interactions would show resistance to the shear induced by the Mastersizer impeller and demonstrate a shift in distribution to larger particle sizes after treatment (Farinato et al, 2010).

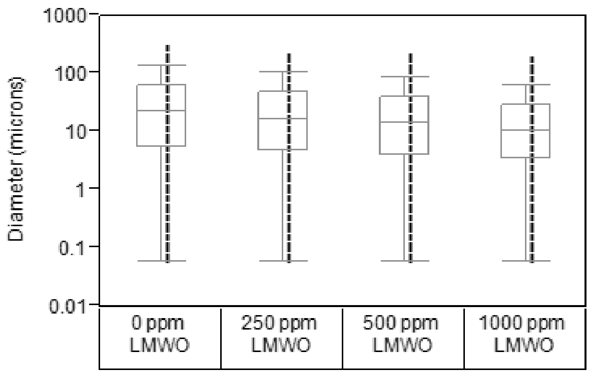

Figure 7. Particle size distribution of tailings treated with LMWO.

The zeta potential results shown in Figure 8 supported the findings of the PSD measurements. The addition of LMWO reduced the intensity of the surface charge. The reduction was most clearly explained by the effect of charge screening, which would lower the electrostatic repulsion between the particles and allow aggregation to occur. As the fine solids began to form aggregates with coarse material, an improvement in supernatant clarity was observed due to the increased settling in accordance with Stokes law.

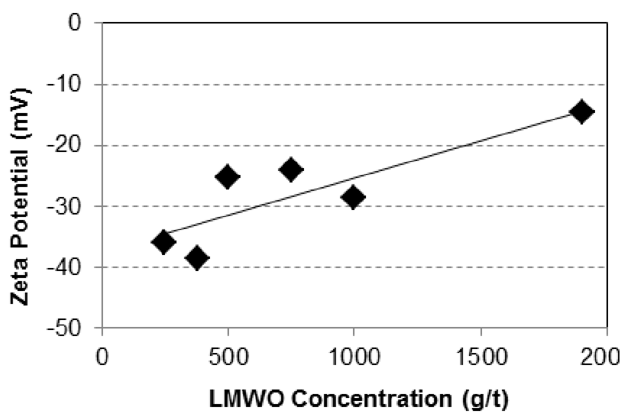

Figure 8. Zeta potential as a function of LMWO concentration.

Effect of Re-agitation

The results of PSD measurements had shown that aggregates formed by LMWO were easily separated when subjected to shear. The treated tailings were, however, able to reform and settle as demonstrated by the results in Figures 9.

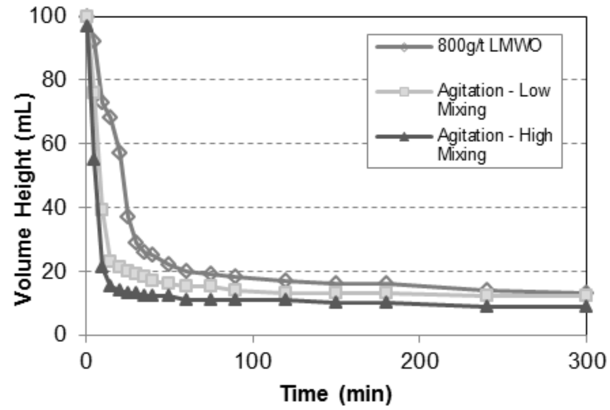

Figure 9. Initial settling rate after re-agitation of settled beds.

Note that although the ISR did improve slightly after the addition of shear, the reduction in settled bed height was the result of sampling for oven solids analysis. The mechanism for LMWO of providing a charge screening in order to aggregate solids would be largely unaffected by re-agitating the sample and the conditions would still exist for the formation of aggregates. The slight improvement in ISR indicated that there was a benefit to increasing the solid particle collisions. The subsequent settling experiments did not affect the solids content as all three experiments contained 7.7-7.8% solids.

Combination with Flocculant

The use of a flocculant alone was able to provide an improvement in ISR as shown in Figure 10, although the supernatants shown in Figure 11 demonstrated that a fraction of the fine solids were unaffected by flocculant addition.

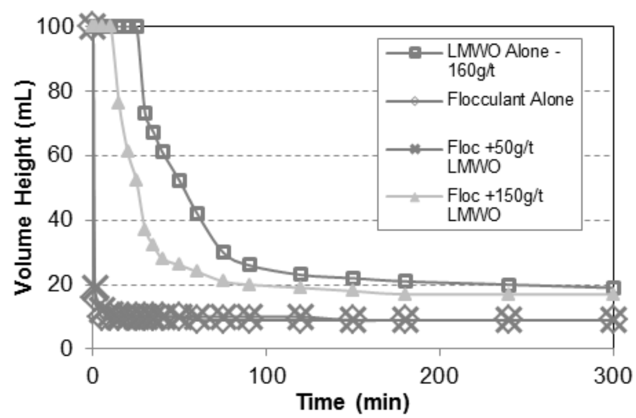

Figure 10. Initial settling rate of flocculant with and without LMWO.

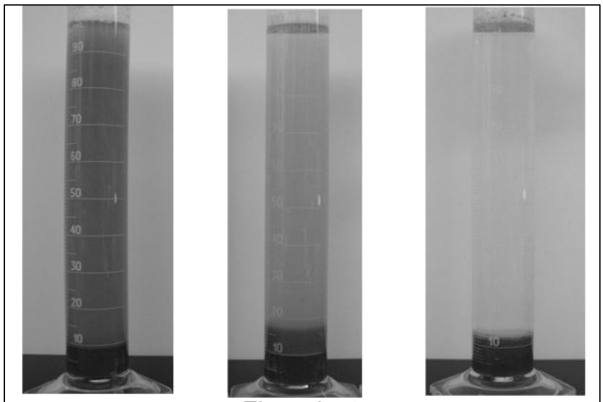

Figure 11. 24 hour tailings settling for flocculant with and without LMWO.

LMWO addition at 50g/t had previously shown poor results but was tested in combination with the flocculant to look for synergistic effects. The two chemical treatments had shown independent effects on the settling. The addition of flocculant enhanced the settling but left a fraction of the fine solids unsettled. After 24 hours the LMWO was able to assist in settling the remaining fines, but as a separate bed with low solids content as shown in Table 2.

Table 2. Settled bed solids content with and without flocculant addition.

	Flocculant Alone	Floc + 50g/t LMWO		Floc + 150g/t LMWO	150g/t LMWO Alone
		Top Bed	**Bottom Bed**		
Floc Bed Solids (%)	9.8	1.5	8.3	5.8	5.4

Comparing to the use of LMWO alone at 160g/t, the combination with the flocculant was able to improve the ISR of the tailings and still provide a clear supernatant, although the bed was not as compact as the use of flocculant alone. Although the addition of flocculants would need to be made after the bitumen extraction process, LMWO improved the treatability of the fine solids so that settled beds could be formed which contained both fine and coarse solids.

CONCLUSIONS

The addition of LMWO as an upfront process aid provided a screening of fine solid surface charge in order to facilitate aggregation without negatively impacting bitumen recovery. The existence of a critical concentration was identified. Operating below this concentration resulted in lower ISR, more turbid supernatant, and a segregated settled bed. Surpassing the critical concentration provided a clear supernatant and a non-segregated settled bed. The aggregates formed by LMWO addition would separate when subjected to agitation, but re-aggregated upon entering a static environment.

This technology can be used to improve the treatability of dispersed fine solids for post-extraction treatment. The amount of LMWO necessary to produce non-segregated tailings would most likely prove uneconomical in a large scale operation unless a cumulative effect could be observed as the water is recycled back into the extraction process.

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MIXING ENERGY DENSITY CRITERION FOR PEAK DEWATERING FLOCCULATION BY STATIC MIXERS

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ABSTRACT

The first step of mature fine tailings (MFT) dewatering processes is the addition of high-molecular-weight flocculants. The effectiveness of the final MFT disposal method depends to a great extent on the flocculation mixing. Inline static mixer flocculation can handle large amounts of non-homogenous material in a continuous process. In this study KMS and KMR type Kenics® static mixers, separately and in combination were used for MFT flocculation. Fluid velocity, V (m/s), and Reynolds number, Re, are commonly used as mixing scales. These two, along with mixing energy density were examined for predicting peak dewatering flocculation by static mixers.

The capillary suction time (CST) dependence of the inline static mixer flocculated MFT with increasing V and Re passed through a minimum. The higher the number of elements in the inline mixer, the lower was the V and Re at which the peak dewatering rate occurred. The same V or Re did not produce similarly dewatered flocculated MFT for different inline mixers. On the other hand, the CST dependence on the mixing energy density per swept static mixer volume occurred within the same narrow range for all the three static mixers used. These indicate that as long as equal mixing energy density is applied, a variety of static mixers and configurations can be selected to produce peak dewatering flocculated MFT. In short, mixing energy density can be used to scale flocculation by inline static mixers.

INTRODUCTION

Current industrial dewatering technologies of mature fine tailings (MFT) from oil sands extraction include centrifugation, thin lift drying, and rim-ditch dewatering. All of the MFT dewatering processes require preconditioning using high-molecular-weight polymer flocculants (e.g., > 1 x 10^6 g/mol). A second pretreatment step in which inorganic compounds could augment the polymer action may also be used to create large flocs and aggregates necessary for the dewatering process.

It is well known that the flocculation outcome is affected by the mixing process. Mixing by agitation of impellers in a tank is commonly used for batch and small-scale applications. The capital and operation cost of such dynamic mixers capable of handling the large inventory of MFT in a timely manner is bound to be very high. An alternative that can be used for MFT flocculation mixing is inline static mixers. In comparison to conventional dynamic mixers, static or motionless mixers are small, low in cost, consume less power, and do not have any moving parts that require regular maintenance. The material residence time is short and good mixing is achieved owing to near plug flow behaviour.

Static mixers consist of specially designed rigid structures inserted end-to-end in pipes in order to enhance both convective transfer and turbulence. The mixing elements split and recombine the mixture radially while the flow continues axially. This repeated action ensures uniformity in composition and concentration. Often, inline static mixers are employed as single pass through mixers and are therefore well suited for continuous tailings disposal processes.

Literature and expertise of manufacturers in the application of inline static mixing for flocculated MFT dewatering is limited. Within the oil sands industry, the relationship between MFT treatment, ease of dewatering and mixing operations are often empirical. This trial and error method is costly due to the numerous trials required. Such an approach is normally fraught with start-up problems, loss of time, and no route to help solve upset conditions and unusual feed property changes. Hence, this study aims to increase the present understanding of MFT flocculation using inline static mixers and suggests a property on which basis an inline static mixer can be selected.

EXPERIMENTAL

A schematic of inline static mixing units is shown in Figure 1. The experiments were conducted in a horizontal 1 inch diameter pipe, total length up to 4.5 m. The MFT was fed from a 1 m^3 tank using a progressive cavitation pump. The polymer solution was pumped using another, smaller capacity (10 L/min at full scale), progressive cavitation pump. The required flow rates of these pumps were set by their variable frequency drives. An Andreas-Hauser™ Coriolis, Promas 83E model flow meter was incorporated at the inlet of the mixing line prior to the polymer solution injection point to confirm the calibrated pump flow outputs.

The flocculant used was a commercial anionic, high molecular weight, polyacrylamide. The flocculant dosage was calculated based on dry polymer mass per MFT solids mass. A stock flocculant solution of 0.2% (w/w) was used to treat MFT that was diluted to about 25 wt% solids concentration. The polymer was injected at the centerline of the 1 inch diameter pipe, prior and close to the static mixers for rapid polymer solution dispersion.

Combinations of two different types of stainless steel statics mixers from Chemineer-Kenics, namely 1" diameter KMS and 3/4" diameter KMR types, were used in this study. Up to four Crystal Engineering™ digital pressure gauges with data logging capability, model XP2i, were used to monitor the pressure along the mixing line (see Figure 1).

The evaluation test for a mixing process is dependent on the goal. For these tests the flocculation process was gauged by the capillary suction time (CST) as the goal was dewaterablity. The CST measurements were conducted using a Triton Electronics meter, type 319 single-radius cell heads, 10 mm in diameter. Whatman paper #14 filter paper were used to measure the CST. Results given are the mean of triplicate CST measurements.

Particle size distribution (PSD) of the solids recovered from Dean-Stark extraction was measured using sedigraph analysis. The clay quantity was determined using methylene blue titration of the solids obtained from the MFT after Dean-Stark extraction, as well as MFT slurry.

RESULTS AND DISCUSSION

MFT flocculation in stirred tanks using a Rushton turbine, a pitched-blade turbine, vane, and airfoil impellers were examined for MFT flocculation as reported earlier (Demoz and Mikula, 2012). The polymer-MFT mixing outcome was evaluated based on the amount of water released and capillary suction time (CST). The CST is known to be inversely related to water release tendency (Sholz, 2006), Meeten and Smeulders, 2004). There was a clear peak in the rate of water release and a minimum in the CST as a function of mixing time. The mixing energy input for a series of MFT flocculation tests in which other conditions were held constant was found to be proportional to the mixing time. The CSTs and the amount of water released after 24 hours of settling clearly showed that there is an optimal mixing energy that corresponded to the most rapid rate of flocculated MFT dewatering.

The power dissipated by the agitation (mixing energy) was shown to be the pervading property between changes in mixing conditions and the dewatering rates of the flocculated MFT. It follows from the above that other mechanical mixing systems meeting these conditions can be used for MFT flocculation. Here, we extend the optimal mixing energy concept for peak dewaterability to inline static mixers, which are the second-most dominant mixers employed in a host of industries (Thakur et al. 2003).

A limited number of parameters are mentioned in association with the flocculation of MFT by inline static mixers. These parameters are well established in many other applications but there are not sufficient studies as to their relevance for MFT flocculation. The most often cited value for inline static mixing flocculation is the slurry velocity. For plug flow, the velocity is obtained by dividing the flow output of the calibrated progressive pump by the effective cross sectional area of the pipe containing the static mixer assembly. Due to reduction in the volume to flow within the static mixers, the average velocity of the MFT is higher than that of the pipe used for the mixer assembly.

The goal of the process dictates the type of test for evaluating the mixing quality. Mixing for blending or distributive mixing or as feed to a solid bowl decanter centrifuge is different from mixing for dewatering at discharge. For thin lift and rim ditch methods the last step before deposition is flocculation and tests of dewaterability are useful indicators of success. The CSTs of the flocculated samples were measured immediately after discharge from the three different inline static mixer setups shown in Figure 2. Figure 3 shows the CST dependence on the fluid mean velocities through the static mixers. The mixing gave poorly dewatering flocculation at smaller and higher velocities with better dewatering flocculations occurring in the intervening range for all the static mixers. The minimum in CST shifted to lower velocities with increasing number of static elements. These results indicate that velocity is not an adequate property for scaling MFT flocculation.

Inline mixing is achieved by both stretching along the elements and turbulent mixing radially. The Reynolds number, Re, characterizes the flow zones and is a common scale-up parameter. Re can be calculated using:

$$\mathrm{Re} = \frac{\rho V D}{\eta} \qquad [1]$$

where, ρ is the slurry density (kg/m^3), η is the viscosity (Pa·s), V is the fluid velocity (m.s^{-1}), and D is the pipe diameter (m). The experimental Re were all greater than 500 and the transition from laminar flow is reported to occur starting from Re around 50 for these types of static mixers (Godfrey, 1992, Kumar and Upadhyay, 2008),. The laminar to turbulent flow transition Re specific to our mixing system will be reported after analyzing the ΔP dependence across the static mixers on fluid velocity. Under laminar flow ΔP is linearly dependent on velocity while in the turbulent zone it shows a greater than 1.7 power dependence. The CST dependence on Re for all the mixers are presented in Figure 4. The Re at which the best CSTs were obtained varied with the number of static elements. Longer static mixers started to give lower CST products and hence better dewatering flocculated MFT at lower Re. A single curve CST dependence on Re was not obtained and this indicates that Re is also not an adequate inline static mixer MFT flocculation parameter.

Dynamic mixer studies have shown that the dewatering rate is well correlated to the mixing energy input. Despite the fact that static mixers have some different characteristics, it is the same mechanical energy input that drives the flocculation processes. The kinetic energy supplied by the pump provides the mixing energy input and appears as pressure energy loss across the inline mixers. The energy density of the inline static mixer flocculation, ε, can be given by;

$$\varepsilon = \Delta P * \rho^{-1} \qquad [2]$$

where, DP is the pressure difference in N.m^{-2}, and r is the fluid density in kg.m^{-3}. The mixing energy density contains information about the mixer and its configuration and residence time. This is energy per swept inline static mixer hydraulic volume. Figure 5 shows the CST as a function of the mixing energy density. It was not only the number of static elements that were varied, but the two different types of mixers were also used in combination. Even though both the measured CST and calculated energy densities have experimental error the peak dewatering flocculation, as shown in Figure 5, was produced around the same energy density window. These results show that the mixing energy density is a reasonable parameter to scale well dewatering flocculation processes. The dewaterability tests reported here did not cover laminar flow inline mixing or the effect of the solids content of the MFT: these will be considered in future studies.

CONCLUSION

Inline static mixers are the second most dominant mixing tools besides rotating impeller type mixers. In particular, their capacity for continuous operation at very low capital cost makes them an appealing choice for large scale MFT flocculation processes. Manufacturers have developed and provided a variety of inline static mixers. However, methods of comparing a process for different designs even by a single manufacturer have not been developed. Therefore, the selection of inline static mixers for flocculation has thus far mostly been done by trial and error.

Similarity of fluid velocity in static mixers has, on occasions, been used to characterize MFT flocculation. Re is another familiar dimensionless parameter used in fluid flow characterization. These two, together with optimal mixing energy density; as was verified with dynamic mixers, were examined for the characterization of inline static mixer flocculation. The evaluation test for the mixing process is dependent on the mixing goal. In MFT flocculation, the goal is to accelerate the dewatering rate and so CST test can be used to evaluate the mixing quality.

Three different inline static mixers were setup using one inch diameter and ¾ inch diameter KMS and KMR type Kenics static mixers. The CST of the flocculated MFT as a function of increasing flow rate passed through a minimum for each mixer setup. It was observed that the higher the number of static mixing elements, the lower was the fluid velocity and Re at which peak dewatering rate flocculated MFT were produced. However, the peak dewatering rates were different for each mixer setup indicating that neither fluid velocity, nor Re, can adequately characterize the MFT flocculation mixing. On the other hand, the CST dependence on the mixing energy density per swept static mixer volume occurred within the same narrow range for all the three static mixers. This is evidence that as long as equal mixing energy density is applied, a wide range of inline static mixers and configurations can be used to produce the desired MFT flocculation. In other words, mixing energy density can be used to scale flocculation mixing.

ACKNOWLEDGEMENT

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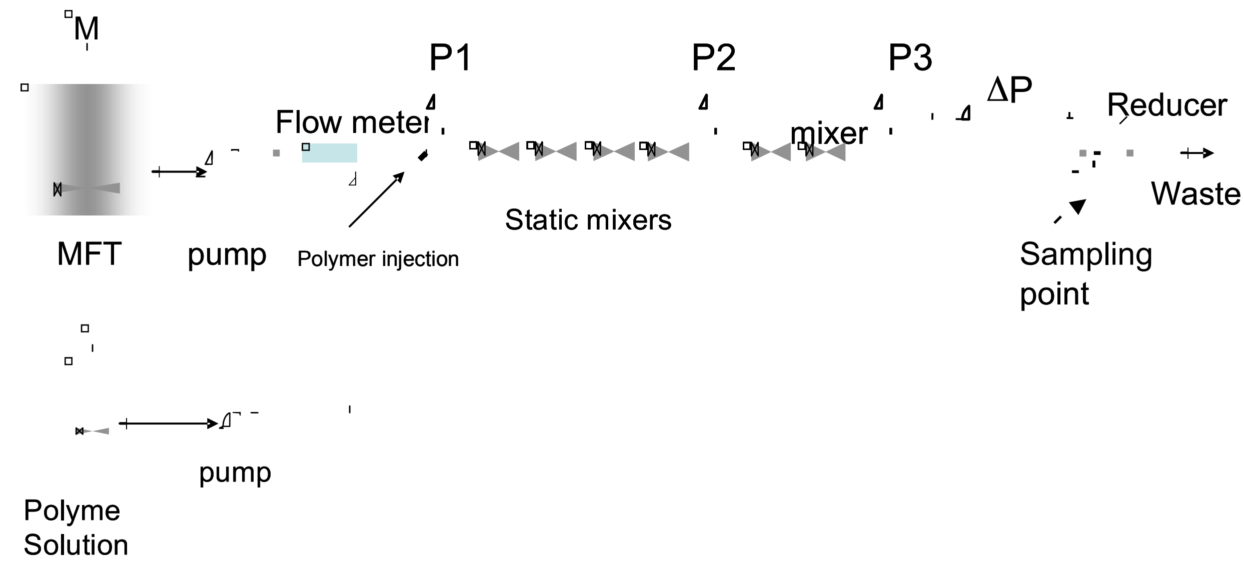

Figure 1. **Schematics of the mixing test units. 'P' stands for digital pressure gauge.**

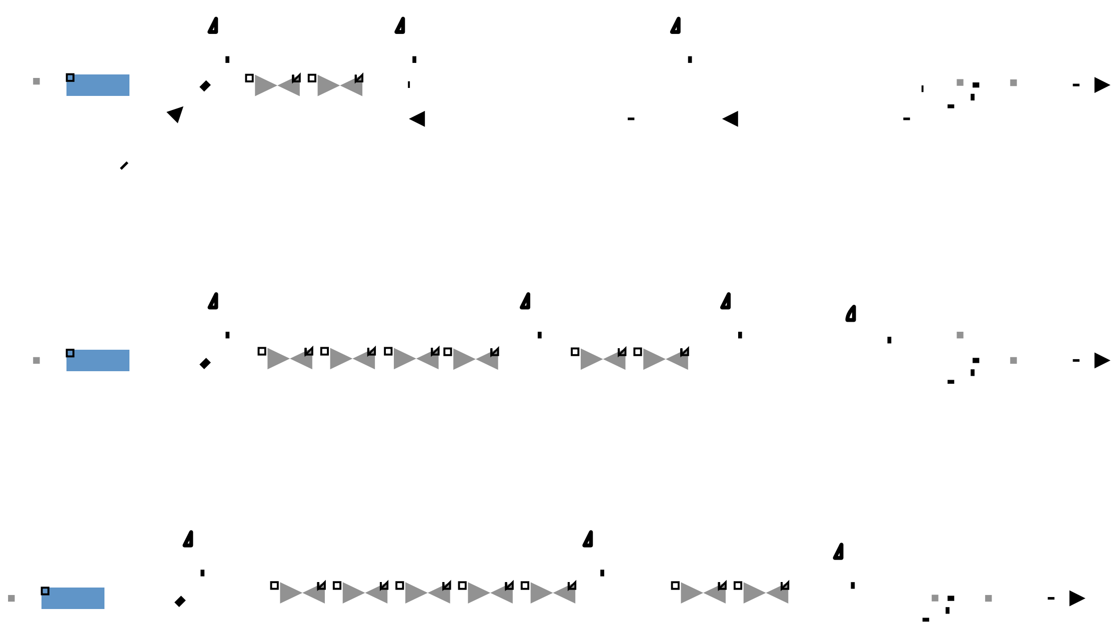

Figure 2. **Schematics showing the three inline static mixing setups. ¾" KMR mixer; 12, 6, and 4 elements mixers were combined. 1" KMS mixers; M1=M2=M3; each with 4 elements; M4 and M5 had 8 and 12 elements each. 'P' stands for digital pressure gauges.**

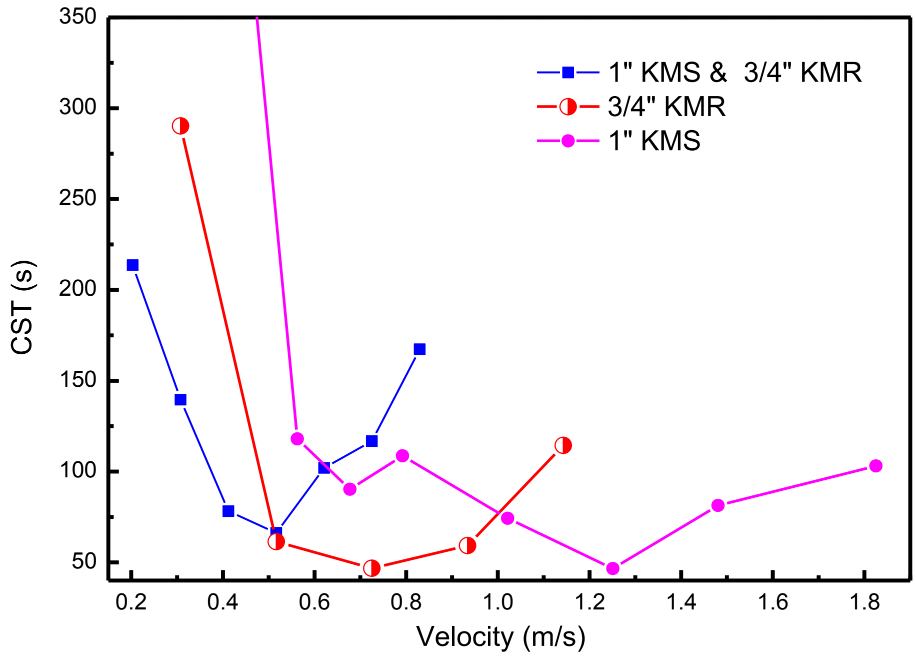

Figure 3. **Dewaterability rate dependence on fluid velocity through inline static mixers.**

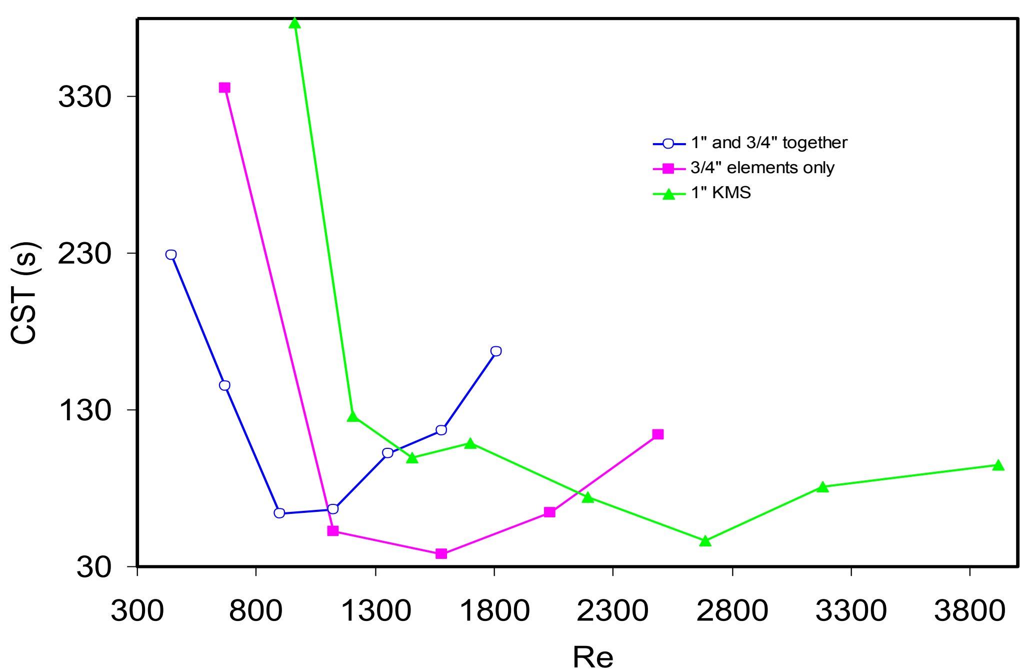

Figure 4. **Dewaterability rate dependence on Re for flocculation done using the three inline static mixers.**

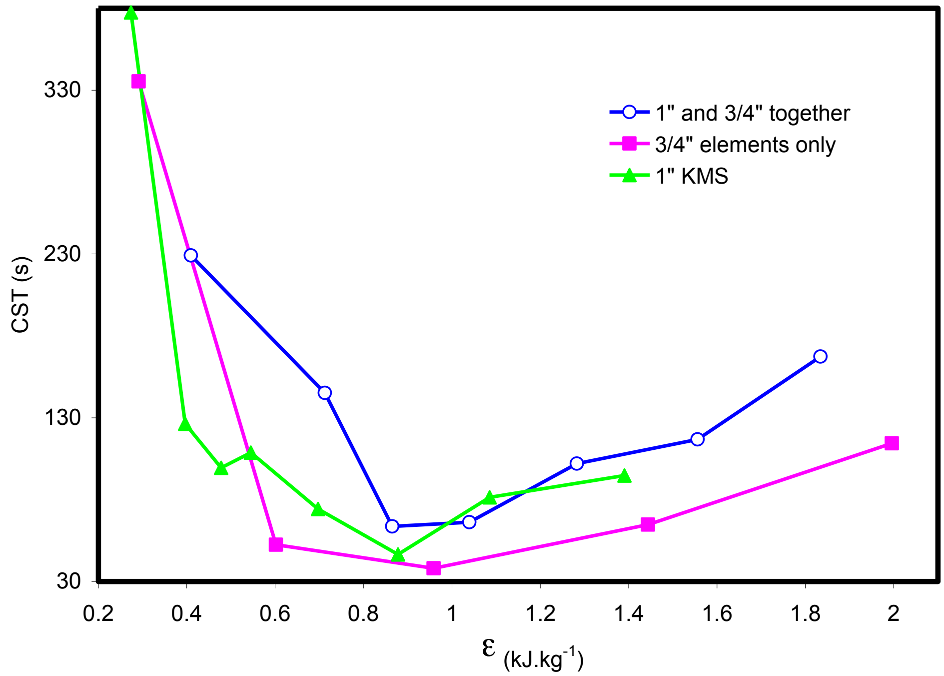

Figure 5. Dewaterability rate dependence on mixing energy density of inline static mixers.

MFT, CHEMICAL ADDITIVES, AND THE "GOLDILOCKS ZONE" OF MIXING

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ABSTRACT

An estimated 700 million m^3 of mature fine tailings (MFT) have accumulated, and continue to accumulate, in a growing number of containment ponds. Various combinations of mechanical and chemical treatments have been proposed for reclaiming tailings ponds, but a common theme is that they involve a critical amount of mixing. If too little mixing energy is introduced, the chemical additives are not adequately dispersed into the tailings, resulting in poor performance. If too much mixing energy is added, the shear tears flocs apart, regardless of the chemistries used. The "Goldilocks zone" of proper mixing can be small and elusive, and missing it can severely hamper dewaterability. However, the chemistries employed can affect the size of this window.

In this research, MFT samples were treated with different anionic polymeric flocculants. A rheometer equipped with a helical stirring shaft was used to record mixing speed and torque as a function of time, allowing mixing energy inputs to be carefully measured. It was observed that flocculant charge and viscosity had a large effect on mixing, with low-to-medium charges and viscosities resulting in more stable dewatering as a function of mixing energy.

INTRODUCTION

In order to remove fine solids from a liquid, the particles must be formed into larger aggregates that can be removed by filtration, centrifugation, or sedimentation. Chemically, this can be accomplished by altering the particulate surfaces according to the schemes illustrated in Figure 1. Surface charges can be neutralized by small inorganic coagulants, allowing Van der Waals forces to bring the solids together. Alternately, coagulant polymers can adsorb onto a surface, creating an oppositely-charged patch that can stick to other particles. Finally, large polymeric flocculants can bridge particles and bring them together. (Fuerstenau, 1995) Selecting the proper treatment chemicals is essential for efficiently dewatering MFT, or any other substrate.

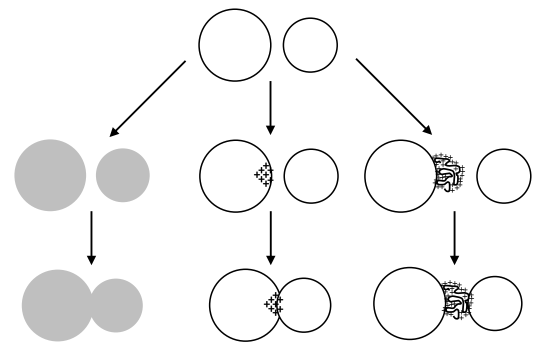

Figure 1. Flocculation of two charged particles via charge neutralization (left), charge patch formation (middle), and bridging by a flocculant (right).

Chemical selection, however, is not the entire story. The way coagulants and flocculants are introduced into the tailings is equally important, and improper mixing can render even the best additives ineffective. (Jarvis, 2005; Spicer, 1996) The process of mixing in a flocculant or coagulant polymer occurs over several steps:

1. A concentrated polymer solution is added to the tailings suspension.
2. The polymers are diluted and dispersed as the suspension is mixed.
3. As the tailings continue to be agitated, the polymers collide with particles and adsorb onto the surfaces.
4. Agitation causes the activated particles from Step 3 to collide with other particles, creating aggregates.

By Step 4, the flocculation process generally is complete, although some flocs have been reported to change into a more optimal shape with some additional mixing. (McFarlane, 2005) Beyond this point, however, extra agitation – even shear caused as the slurry is transported along a pipe to a deposition area – can cause the flocs to disintegrate, which results in poorer dewatering. (Watson, 2011)

If complete polymer dispersal is not achieved, too many solids will escape flocculation and not dewater. At the other end of the spectrum, if too much mixing is applied the flocs will begin to break apart and become too small to dewater. (Demoz, 2010; Demoz, 2011) Staying within this “Goldilocks zone” is crucial to successful flocculation, and maximizing its size would facilitate recovering as much water as possible from MFT. Fortunately, flocculant choice appears to impact the mixing window size.

EXPERIMENTAL

Samples of mature fine tailings and tailings pond water (TPW) were obtained from a major oil sand producer in Alberta, Canada. The MFT were diluted to 20 wt% with TPW and treated with a series of flocculant-grade anionic polymer solutions that had been prepared at concentrations of 0.2 wt% in TPW. Flocculant specifications are summarized in Table 1.

Table 1. Flocculant charge and viscosity specifications.

Name	Viscosity	Charge
A	Low	Low
B	Medium	Low
C	Medium	Medium
D	Medium	High
E	High	High

Mixing measurements were conducted on an Anton Paar MCR 300 rheometer equipped with an ST 60-2HR-90/188.5 helical impeller. For mixing experiments, an MFT suspension (650 g) was poured into a 1-L beaker, which was then placed under the rheometer and mixed at a constant speed while the torque was measured over time. After 60 seconds of mixing, a flocculant was injected (1500 g/t) at the bottom of the beaker and the changes in torque were followed.

Flocculant addition took approximately 20 s. Ten seconds later (90 s after the test started; 30 s after beginning flocculant addition) a series of aliquots (16 g on average) were collected at regular intervals and allowed to sediment for 16 hours in glass vials. The net water loss and solids contents of the supernatantants and flocculated beds were calculated gravimetrically.

Mixing energies were calculated from torque vs. time curves according to Equation 1,

$$\varepsilon = 2\pi N \int_0^t (M - M_0)dt \qquad (1)$$

where ε is the mixing energy (J), N the mixing speed (revolutions per second), M the torque (N·m), M_0 the no-load torque (N·m), and t the time (s).

Net water release was calculated according to Equation 2,

$$NWR = \frac{W_R - W_A}{W_0} \times 100 \qquad (2)$$

where NWR is the net water release (%), W_0 is the initial mass of water in the sample (g), W_R the mass of water released (g), and W_A the mass of water added when the flocculant solution was dosed into the MFT suspension (g).

RESULTS AND DISCUSSION

Calculating mixing energy from the torque vs. time curves allows mixing information to be more easily analyzed. For example, Figure 2 shows the torques for MFT treated with 1500 g/t of Flocculant C and mixed at 100, 200, and 300 RPM, and Figure 3 shows the net water release and solids in the recovered water for each run as a function of time. No clear trend is visible in these data.

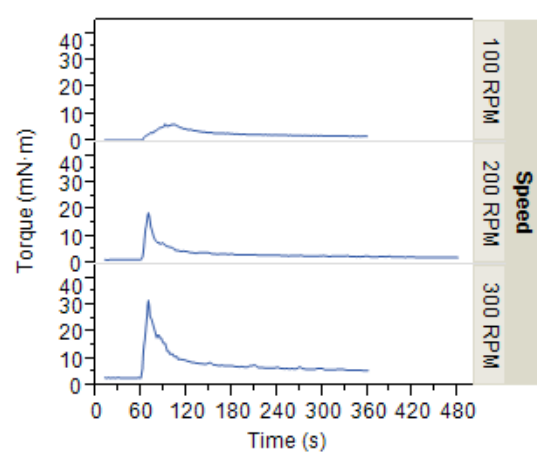

Figure 2. Torque curves for 20 wt% MFT treated with 1500 g/t of Flocculant C and mixed at different speeds over time.

Recasting the dewaterability results as a function of mixing energy in Figure 4, however, allows the underlying trend to be seen: All three series line up along the same curve and describe different phases of the mixing process. At 100 RPM the polymer is still being dispersed throughout the suspension and flocculation is only beginning to occur, evidenced by the increase in net water release as the mixing energy input increases. Optimum water release is achieved during the 200 RPM run and stabilizes throughout the 300 RPM run.

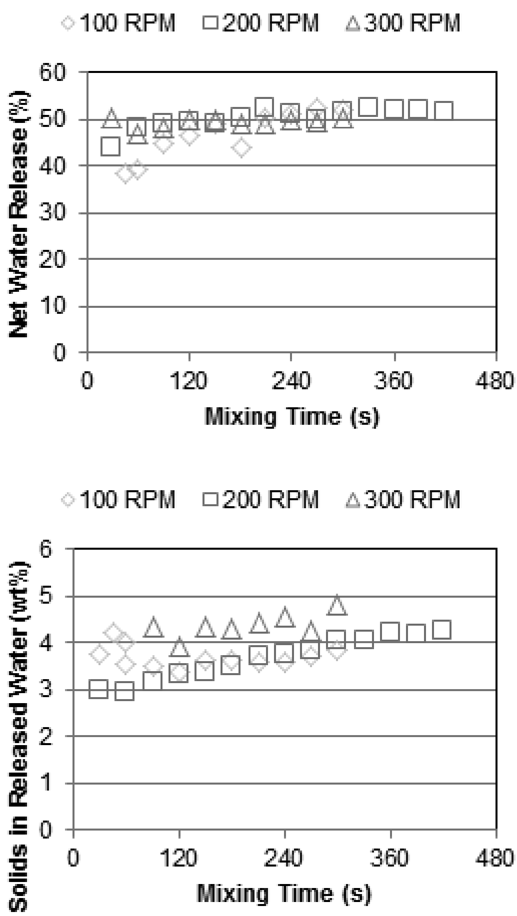

Figure 3. Net water release (top) and solids in the released water (bottom) as a function of mixing time for mixing speeds of 100, 200, and 300 RPM.

Even though the water release is approximately constant throughout the 200 and 300 RPM tests, signs of overmixing are present. The concentration of solids in the recovered water slowly increases as the mixing energy input exceeds 17 J, indicating that the flocs are beginning to disintegrate. Flocculant C appears to have a very narrow window for optimum mixing – about 10-17 J, or about 60-150 s at 200 RPM.

Figure 5 shows the torque vs. time curves of MFT treated with different flocculants and mixed at 200 RPM. For all the flocculants, a torque of approximately 1.6 mN·m is needed to achieve the desired impeller speed. After the polymer is injected at 60 seconds, the torque reaches a maximum, and then begins to return to the baseline torque value. Flocculant B produces the sharpest torque spike at 53 mN·m, while Flocculant A gives the most sustained torque change. The higher viscosity and higher charge flocculants, conversely, produce less of a change in torque. By the end of the test, the torque for Flocculants B-F had fallen to approximately twice the starting torque, while the torque for Flocculant A was five times the initial value.

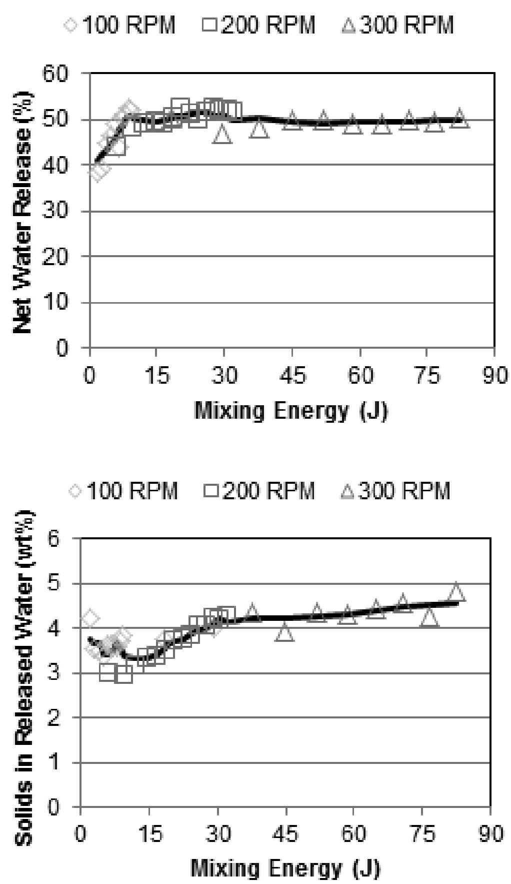

Figure 4. Net water release (top) and solids in the released water (bottom) as a function of mixing energy. The solid line is a moving average.

Aliquots taken at regular intervals from these tests were allowed to sediment for 16 hours, and the net water release and solids in the released water were measured gravimetrically. In Figure 6, Polymers A, B, and C showed little change in net water release as a function of mixing energy, while the dewaterability of more viscous and higher charge polymers D and E deteriorated with additional mixing.

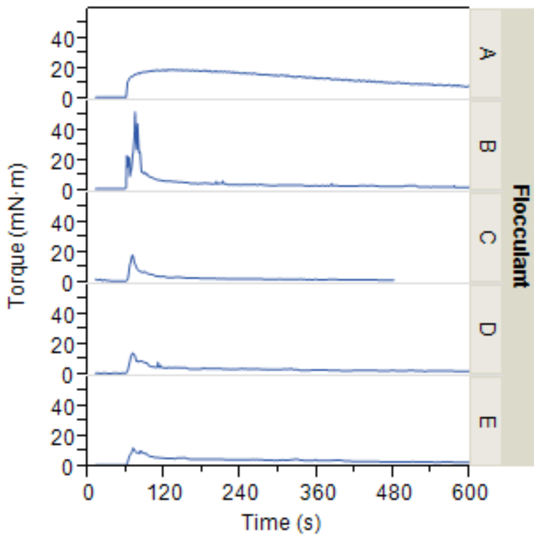

Figure 5. (Top) Torque vs. time curves at 200 RPM for 20 wt% MFT that were treated with 1500 g/t of different polymeric flocculants at t = 60 s.

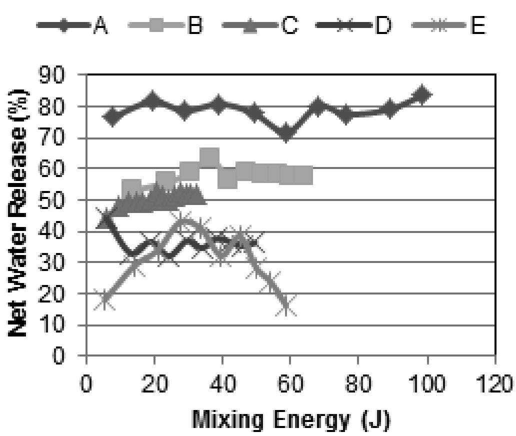

Figure 6. Net water release as a function of mixing energy and flocculant type.

The amount of solids in the released water also changed as a function of the mixing energy added to the system. In Figure 7, the performance of flocculant E was particularly affected by the amount of shear, with the solids content in the released water rapidly approaching that of the untreated MFT. Flocculant B kept the solids concentration steady, while C slowly increased over time as flocs were sheared apart. D took a little longer to stabilize, while the amount of suspended solids in A was still decreasing by the time the test ended, indicating that mixing the MFT treated with A for a little longer might give better results.

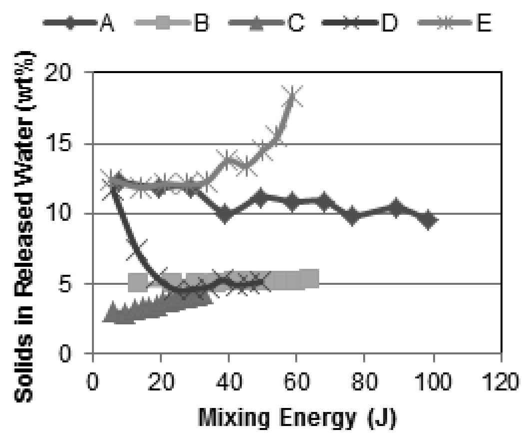

Figure 7. Amount of solids in released water as a function of mixing energy and flocculant type.

Next, the dewaterability of MFT treated with blends of Flocculants A and C were evaluated to see if there was some way to combine A's water release with C's solids containment. First, a 50/50 blend was compared with sequential addition of the flocculants – A followed by C, or C followed by A. Figures 8 and 9 illustrate the results.

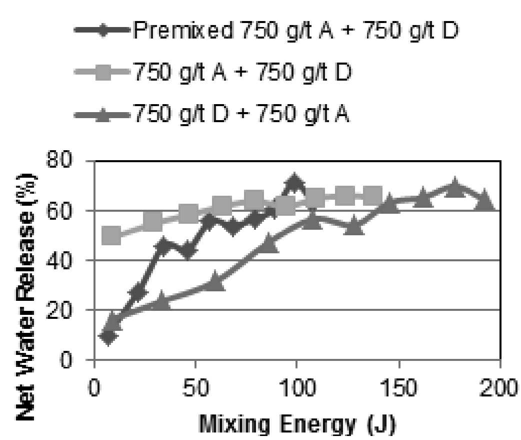

Figure 8. Effect of flocculant order of addition on MFT dewaterability.

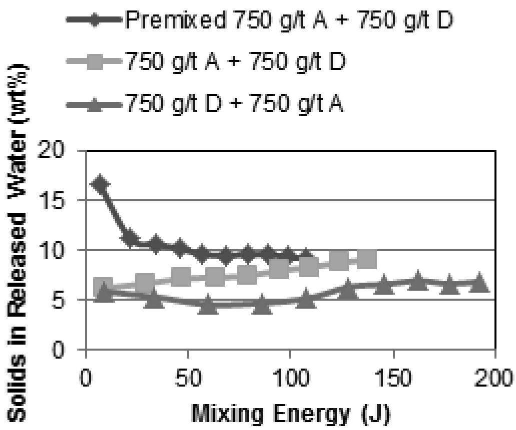

Figure 9. Effect of flocculant order of addition on the amount of solids in recovered water.

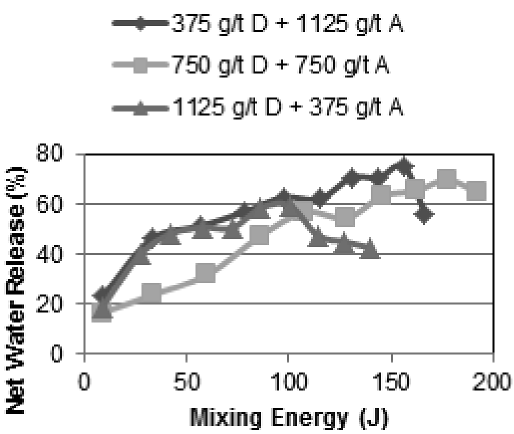

Figure 10. Effect of flocculant ratio on dewaterability of MFT treated first with Flocculant C, then Flocculant A.

The dewaterability of all the samples increased as they were mixed, with the sample treated with A first increasing the most quickly and the sample treated with C first increasing the most slowly. MFT treated with a mixture of A and C performed between these two extremes. Ultimately, all achieved between 60 and 70% net water release, however.

The biggest differentiator lay in the concentration of solids in the recovered water. Treating with C followed by A removed the most solids from the water, while premixing A and C removed the least. Also, the amount of solids in the recovered water appeared to be insensitive to the amount of mixing energy applied when C was added before A.

With the order of addition set, other ratios of Flocculants A and C were tested, with the results presented in Figures 10 and 11. Treatments consisting of 50/50 C/A and 75/25 C/A released more water than MFT treated with the 25/75 C/A blend. The 50/50 ratio, however, resulted in the lowest amount of solids in the recovered water, as well as was the most insensitive to mixing.

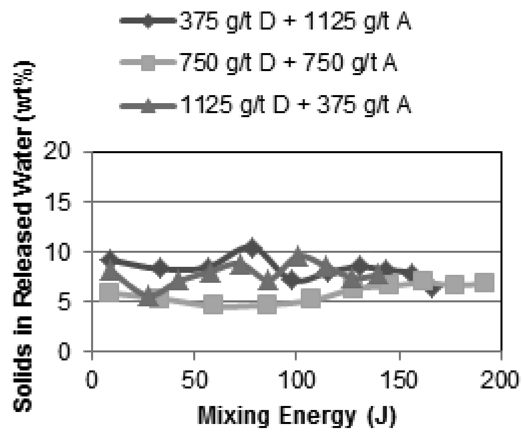

Figure 11. Effect of flocculant ratio on the amount of solids in recovered water. First Flocculant C was added, followed by Flocculant A.

CONCLUSIONS

Successful MFT dewatering requires not only selecting correct treatment chemicals, but also successful mixing. Flocculants and coagulants must be dispersed throughout the suspension and then allowed to contact and aggregate solids, which requires adding mechanical energy into the MFT. Too little energy, and the solids will not flocculate homogeneously. Too much energy, and the flocs will shear apart.

In this research, a method to measure the amount of mixing energy added to a tailings slurry was developed, and the effects of mixing were observed on a suspension of 20 wt% MFT treated with different polymeric flocculants. It was observed that different polymers not only affected how the solids flocculated, but also affected the mixing window.

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Session 4

Tailings Properties Measurement

SOLID CONTENT OF OIL SANDS TAILINGS MEASURED OPTICALLY

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ABSTRACT

Real-time solid content measurements in oil sands tailings are required to better manage tailing disposal and reclamation. Moreover, inexpensive real time measurements may improve understanding of tailing behavior and offer improved control. Here, we present results from measurements of solid contents in tailings using a light scattering technique.

Results from measurements of different solid content MFT Albian MFT from 15 m depth (AL15), Albian MFT from 7.5 m depth (AL7.5), Albian thickner undeflow tailings (TUT), gold mine tailings, Kaolin and Devon silt are presented here. These measurements have been conducted both in cuvettes and in large cylinders to study solid content change during sedimentation. The scattering behavior is determined by the sample's particle size distribution and processing history. Both increasing and decreasing scattering light with increase in solid content is observed.

INTRODUCTION

During bitumen extraction from oil sands, large volumes of tailings consisting of clay, silt and sand, water and small quantities of bitumen are produced. Tailings are collected in settling ponds and take several decades to settle naturally due to very slow consolidation rates [K.L. Kasperski, 1992, M.D. MacKinnon, 1989]. One of the main challenges of oil sands management is to increase settling rates to improve water recycling. A real time detector which can measure the solid content is an important instrument which can aid dewatering studies. As the settling rate and the carrying capacity of tailings depends on solid contents, it is important to monitor the solid content of tailings both with time and space. Tailings treatment and disposal options also depend on the solid content of tailings.

Established solid content measurement techniques include gamma ray detection (Dromer et al. 2004 and Kennedy et al. 2006) and gravimetric determination (Smith et al. 1991). Other methods for moisture content detection include neutron scattering for hydrogen nuclei detection, frequency domain methods (dielectric constant, capacitance or impedance) and time-domain methods (time domain reflectometry). Most of these methods are either off line techniques, require sample preparation or are very expensive.

Optical techniques potentially can be used for in-situ real time monitoring of solid content in oil sands tailings facilities. Potential optical techniques include near infrared light reflectivity with and without fibre optics (Pamukcu et al. 2006, Mouazen et al. 2005, Belisle et al. 1996, Garrido et al. 1999) and light scattering (Kotylar et al, 1996).

In this paper, we focus on the study of the light scattering technique as a means to monitor tailings solid content. This is a relatively simple and economical technique, which can be used to obtain real-time measurements with no sample preparation. The sensor footprint is small and hence multiple sensors can be used to obtain data from different levels in a tailings pond or settling tank. It can also be used to measure time-resolved single-point data in a flow tube. Our instrument can provide real time spatial and temporal profiles of tailings settling processes.

MEASUREMENT TECHNIQUE

Light scattering consists of three different mechanisms: Rayleigh, Mie and Tyndall scattering. Scattering from particles depends on the wavelength of light (λ) and the sample particle size (φ). Rayleigh scattering is elastic scattering from the particles which are much smaller than the excitation wavelength ($\lambda / \varphi > 10$). Mie scattering is scattering from particles on approximately the same scale as the wavelength of light ($0.01 < \lambda / \varphi < 10$). For particles much larger than the wavelength of light ($\lambda / \varphi < 0.01$), the Tyndall scattering mechanism dominates.

Based on the price and power for lasers at different wavelengths and preliminary measurements (not reported here), a 405 nm laser diode was chosen for this application. Figure 1a shows particle size distributions Albian 7.5 (AL7.5), Albian 15 (AL15) and Albian thickness underflow tailings (TUT), Figure 1b shows the same for Devon silt, Kaolin and gold tailings. Based on the particle size distribution, the laser wavelength, and the scattering mechanisms described above, the boundaries between the different scattering domains are indicated in the Figure 1.

From the particle size distributions shown in Figure 1, Kaolin, Gold tailings, Devon silt and AL7.5 are expected to exhibit mainly Mie scattering. MFT samples TUT and AL15 have wider particle distribution with larger particles above 40 μm and are thus expected to have an additional contribution from Tyndall scattering.

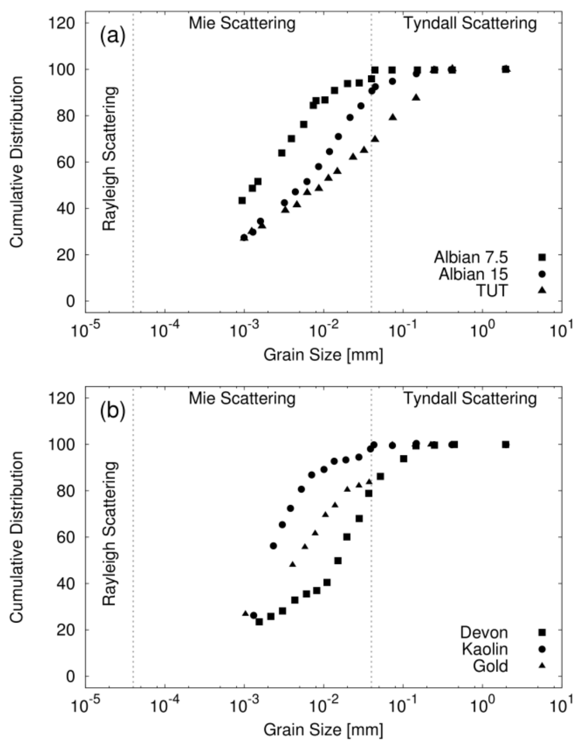

Figure 1. Particle size distribution of samples used in this study: (a) shows Albian tailings (AL7.5, AL15 and TUT), (b) shows Devon silt, Kaolin, and Gold tailings. Based on the particle size the different scattering regimes are indicated. It can be seen that for these samples Mie scattering is the dominant mechanism.

Transmission for an AL15 sample with 38% solid content with a spectrophotometer has been measured. The absorption coefficient (cm^{-1}) versus wavelength for AL15 is shown in

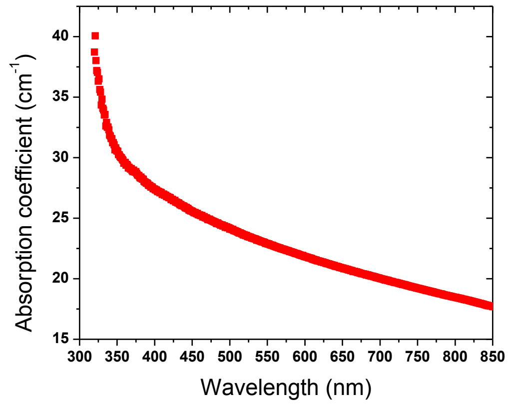

Figure 2. Absorption coefficient of 38% solid content MFT over different wavelengths, indicating a measurement depth of 370 μm for 405 nm wavelength.

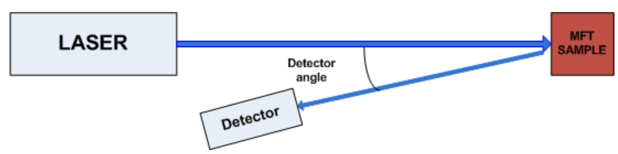

Figure 3. Solid content measurement setup in cuvette; showing the laser and detector position with respect to the sample. The detector angle was optimized to minimize specular reflection and still collect the scattered light.

Figure 2. It is observed that at 405 nm the absorption coefficient is very large, 27 cm^{-1}, which indicates that the laser has a very small measurement depth of ~ 370 μm. Thus, the scatter from the particles is primarily from the sample's surface. This is indicative of our sensor's measurement depth for all the samples used in this study.

The sensor used in this study for the light scattering measurements includes a light source and a photodetector which detects the scattered light from the sample. In our prototype we used a 405 nm laser diode and a silicon photodiode for the measurements.

Figure 4. Settling tank with MFT and six levels of solid content measurement. Inset shows the beam splitter design used for this measurement along with six photodetectors.

Figure 3 shows the schematic for the cuvette measurements. The MFT sample is measured in a square cuvette of 1 cm x 1 cm x 10 cm. The laser's beam size can be used to average the particle size over the beam's spot area. Thus, a larger optical beam can be used to obtain scatter information from that volume of the sample. We use the laser beam spot of 4 mm to 5 mm in diameter for the results presented here.

Figure 4 shows the picture of a real time settling experiment of MFT with solid content measurements being conducted at six levels. The figure inset shows the schematic drawing for measurements of the solid content of tailings in a settling tank. A series of beam splitters are used to split the laser beam into multiple beams. This simple design allows us to use one laser diode to excite several different levels. Different photodiodes are used to measure the solid content at each level. Each beam splitter reflects 8% of the light into the sample; the remainder is transmitted through to the next level. This laser is operated at a frequency of 5 Hz – 10 Hz with a 50% duty cycle. The signal from each level can be acquired at a desired rate, up to 1 Hz.

RESULTS

Cuvette measurements

Figure 5 shows the solid content measurements conducted with samples including Kaolin, Gold tailings, Devon silt, TUT AL15 and AL7.5. These measurements were conducted using the setup shown in Figure 3.The plots show three sets of measurements conducted on different days. The reproducibility is excellent for a prototype; improved mechanical bracing and optical blocks are expected to reduce variability.

Experiments have also been conducted to ensure that no significant fluorescence is collected in this detector configuration. Using filters to remove the fluorescence of the MFT samples, we determined that the signal collected by the detector was dominated by scatter light. The standard deviation of the measurements from the cuvette is between 0.5% - 3%, corresponding to solid content sensitivity on the order of a few percent.

For Kaolin Gold tailings and Devon silt, Figure 5a, b and c, signal intensity increases with solid content. While forward scattering is expected to dominate, a measureable scattering signal is still observed. Moreover, the signal intensity is linearly related to particle density. We expect this is due to an increase of particle number in the sample volume, leading to more scatter signal.

The Albian tailings exhibit a profoundly different behavior; all three show an inverse relationship with increasing solid content. Despite very similar particle distributions, the observed scattering behavior for AL15 (Figure 5e) and the gold tailings (or Devon silt) samples show opposite dependencies on solid content. This may be due to agglomeration of particles in higher solid content samples.

As particle size increases (may be due to agglomeration), forward scattering is expected to increase (S.He et al., 2009). As the density of the large particles increases, this enhanced forward scattering may lead to higher absorption of the laser light by the sample, producing the inverse relationship that we observe here for TUT, AL15 and AL7.5. We also note that the particle distributions reported above may not capture chemical interactions between MFT particles in solution. The distributions in the cuvette may not reflect the sieve experiments.

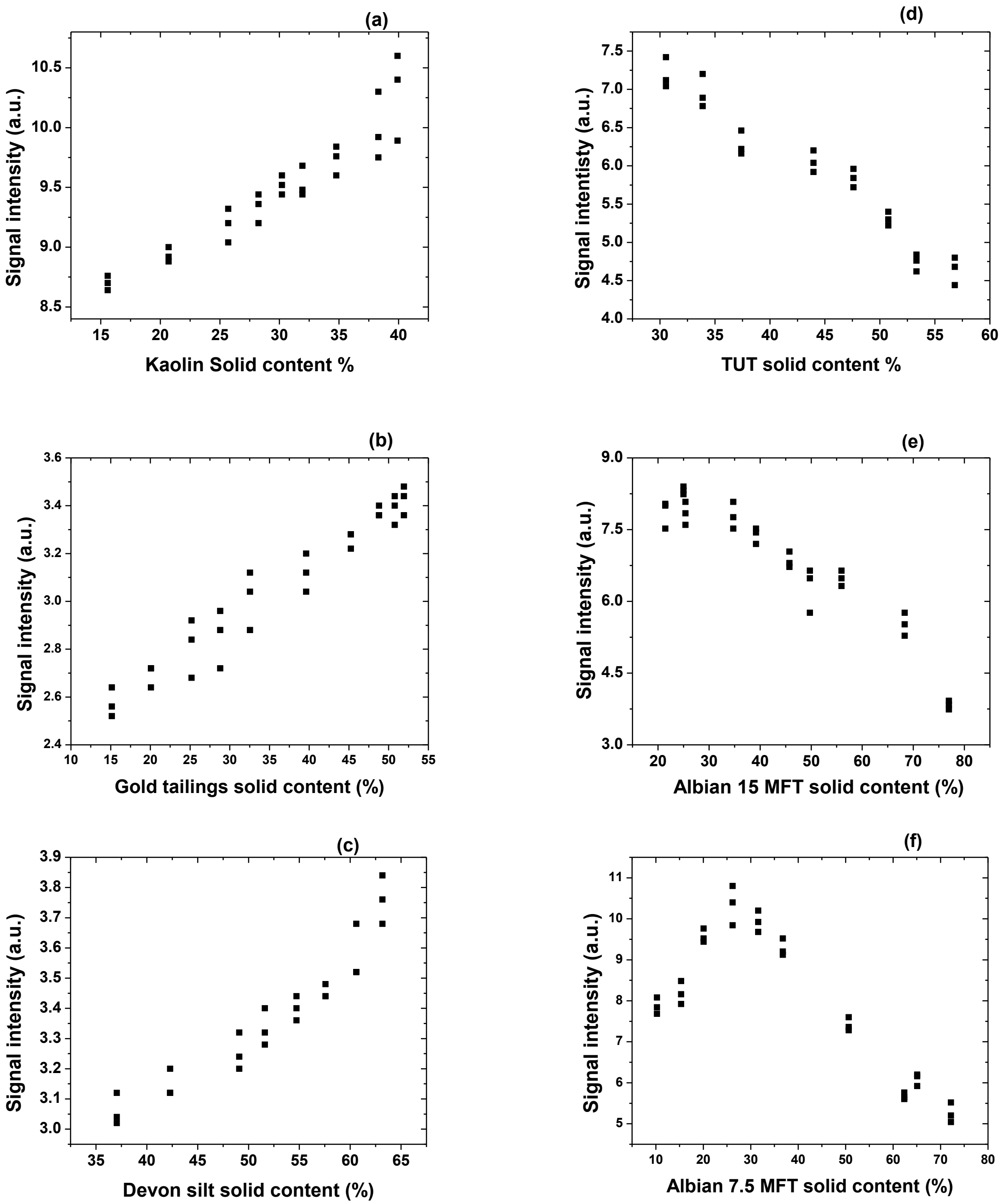

Figure 5. Scatter signal intensity measured for different solid contents for (a) Kaolin, (b) Gold tailings, (c) Devon silt, (d) TUT, (e) AL15 and (f) AL7.5. Kaolin, Gold tailings and Devon silt all indicate a linear increase in the scattering signal intensity with solid content. All three MFT process samples show a decrease in the scatter intensity with increase of the solid content.

The detailed modeling required to confirm this hypothesis is beyond the scope of this paper; however, it is clear from these experiments that the scattering technique is capable of measuring solid content for a wide range of particle sizes, and from different sources. Thus, we have proceeded with our efforts to demonstrate a system capable of measuring multiple points simultaneously. We have designed a beam splitter system for conducting simultaneous solid content measurement in a settling tank at multiple points.

Multiple level solid content measurements

Solid content measurements were conducted at multiple levels during settling experiments. These were conducted for the Kaolin, TUT and AL15 mixed with gypsum. Samples were stirred to achieve a homogeneous solution and uniform solid content at the experiment's start.

Figure 6a shows the solid content measurement for TUT sample in a settling experiment. The plots are shown for light scattering signals (which is proportional to solid content) versus settling time. For this experiment the sample height was 18 cm and the detectors were placed at 3.5 cm from each other. Level 1 is the detector close to the top of the cylinder for all the measurements. Level 2 is closer to the bottom of the cylinder.

From Figure 6a, it is observed that Level 1 exhibits an increase in the scattering signal with decrease of solid content and vice versa for Level 2. This can again be explained by the increase in forward scattering with increase in solid content which corresponds with the cuvette measurements (Figure 5d).

Figure 6b shows the AL15 solid content measurement for three levels. It is observed that unlike the cuvette measurements we do not measure a decrease in the signal with increase in solid content.

Figure 6c shows a three level solid content measurement of AL15 mixed with gypsum (1500 ppm). Gypsum increases the rate of settling of the solid. It is observed that the Level 1 and 2 see a solid to clear phase transition, whereas Level 3 is starting to observe a settling trend. Both AL15 and AL15 with gypsum exhibit Kaolin, Gold tailings and Devon silt like behavior unlike the cuvette measurement for AL15 (Figure 5e), which shows a decrease in signal with increasing solid content. Further work is required to understand this behavior. If these measurements are continued for a longer period of time we will observe a constant signal for the levels that have settled completely and are at a steady state.

To demonstrate higher resolution settling measurements, we have used Kaolin because it settles faster. The settling column height was 23 cm. The sensors were placed at 1 cm distances. Four levels were measured for this settling experiment.

Figure 7a shows signal intensity measured from four different levels from a Kaolin settling experiment, with an initial solid content of 35%. An exponential settling process is observed, with the scattered signal decreasing with time. Using the Kaolin light scattering vs. solid content data shown in Figure 5a, this correlates to increasing Kaolin density. Figure 7b shows a more detailed plot for measurements near the settling column's surface. The higher levels settled very quickly, (detector collects low scatter intensity), whereas the lower detector is capturing the slower settling process in the lower portion of the settling column. All the four levels shown in the plot do observe settling from solid content to the clear water phase.

The data presented here is the raw data collected from these proof-of-concept experiments. To correlate the solid content to light scattering signals a similar cylinder filled with samples with known solid content under homogenous unsettled conditions can be used for calibration studies. Moreover, cross-calibration with an industry accepted technique such as a gamma ray densitometer will be desirable.

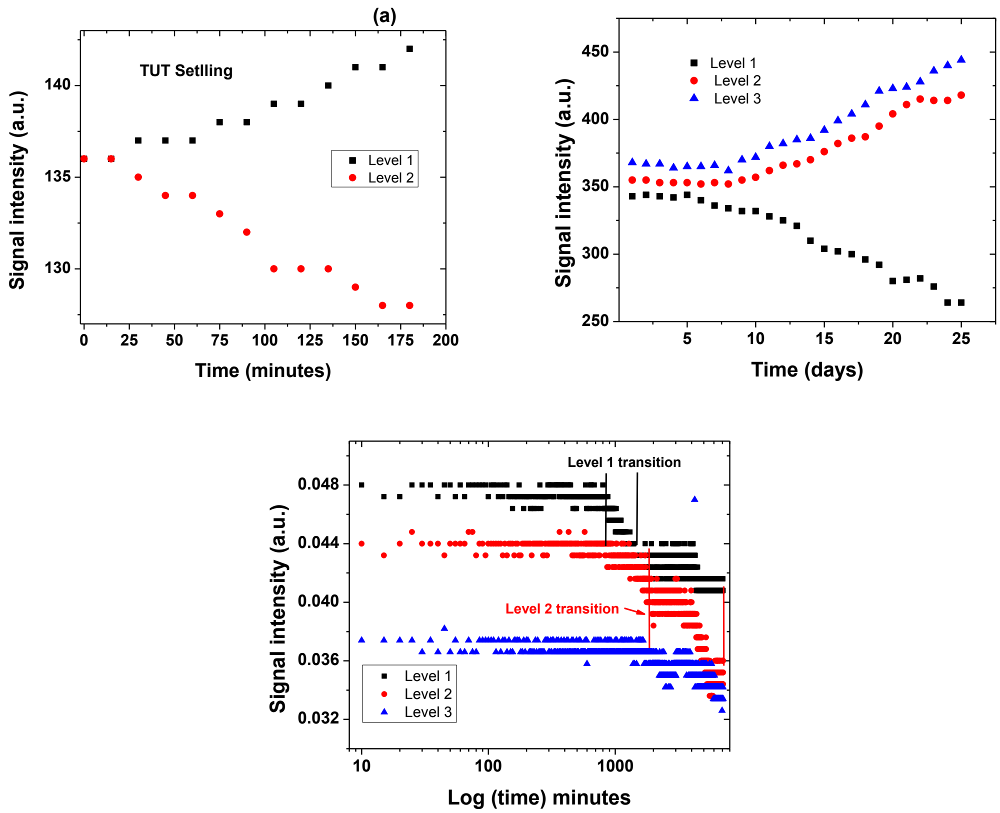

Figure 6. **(a) Two level TUT settling experiment, showing column settling with signals consistent with cuvette experiments. (b) Three level AL15 settling experiment with signals inconsistent with cuvette experiments. (c) Three-level AL15 accelerated settling experiment with added gypsum. Level 1 and 2 demonstrate the transition from solid to clear water phase, where Level 3 is starting to observe settling.**

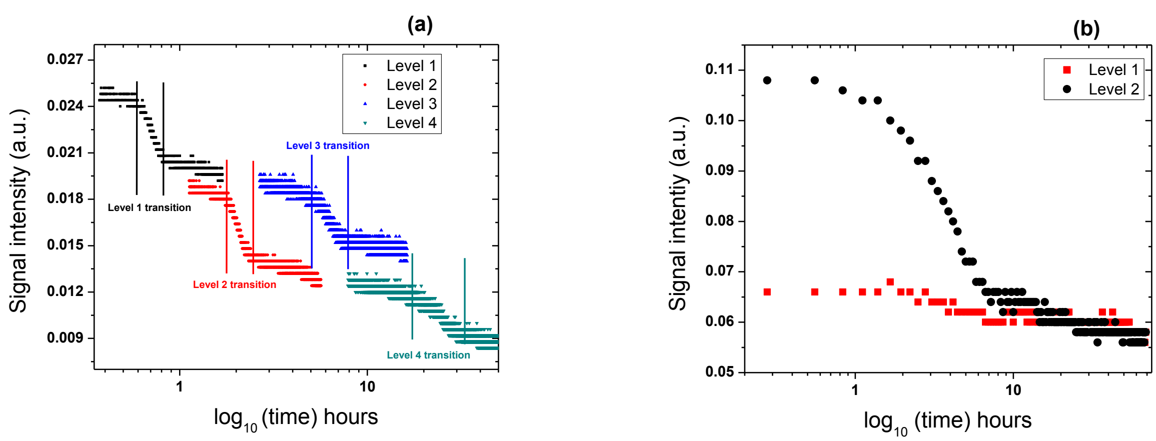

Figure 7. **(a) Multi level Kaolin solid content measurement – the detectors are place at 1 cm distance from each other. It can be observed from the measurement that the transition of each level from solid to clear water is observed when the signal level (scatter intensity) decrease. Thus both spatial and temporal data for the settling can be obtained. (b) Two level measurement of Kaolin indicating that the Upper Level 1 has settled is in water phase and lower level 2 is transitioning from solid phase to clear water phase.**

CONCLUSIONS

A simple and economical instrument based on light scattering technique has been developed to measure changes in solid content of tailings. Cuvette measurements of different solid content samples of Kaolin, Gold tailings, TUT and AL15 show that small changes in solid content can be measured using this technique, for MFT samples over a wide range of range solid content (20% - 80%). Depending on the sample particle distribution and processing history either an increase or decrease of the scatter intensity with solid content was observed with the similar experimental conditions. Preliminary multilevel solid content measurements have been conducted using light scattering in settling tanks. Correlation between the solid content and raw scatter intensity needs to be derived.

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FLUORESCENCE SPECTROPHOTOMETER FOR CONE-PENETROMETER APPLICATIONS

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ABSTRACT

Detection of hydrocarbons and naphthenic acids (NAs) contaminants in soils is important for environmental applications. We have developed two fluorescence spectrometer prototypes for use with a cone-penetrometer. This environmental screening instrument may be useful for soil contaminant detection and for use in testing oil sands tailings.

Both instruments have a compact 25 mm diameter. The first prototype uses fiber-coupled ultraviolet LEDs as an excitation source and a Littrow prism for dispersing the fluorescence spectrum. The second prototype uses a 405 nm laser diode for excitation of soil pore fluid for fluorescence. In this case sample fluorescence is observed through five different long pass optical filters (435 nm, 515 nm, 550 nm, 590 nm and 610 nm) and subsequently imaged on a charged couple device camera. The sample's fluorescence signature can be reconstructed by analyzing the intensity observed through each optical filter. The device has no moving parts and hence is robust for field applications.

Fluorescence results have been obtained for analytes such as naphthenic acids in mature fine tailings, crude oil and diesel. Current experimental results and instrument validation will be presented.

INTRODUCTION

Due to hazardous chemicals and by-products from industrial processes, it is necessary to monitor lands surrounding a plant for contamination before it has a large negative impact. Naphthenic acids (NAs) are both responsible for toxicity in water (Allen, 2008) and are corrosive to industry equipment (Slavcheva et al. 1999). Traditional detection methods require a soil or water sample to be taken from the site back to a lab, and the results reported at a later time. In the case of large scale projects occurring in remote areas, it may take a substantial amount of time before the results can be obtained. Therefore it is ideal to have a compact, field-portable tool that can be easily used to obtain real-time measurements on site.

Fluorescence is a common technique used to determine chemical composition. Generally, fluorescence can be described as the light emitted due to relaxation of a compound excited by higher energy photons. Traditional fluorescence spectroscopy uses an immobile spectrometer with a highly sensitive detector to measure the fluorescence spectrum with high resolution. If the signature fluorescence spectrum of a chemical is known, it can be used to identify a contaminant. Alternately, the fluorescence can be simply compared to that of 'clean' soil to instantly determine if further testing is required.

Portability in fluorescence instrumentation has been continuously studied till the present (Alarieet al., 1993; Baird and Nogar, 1995; Hart and JiJi, 2002; Obeidat et al., 2008; Taschuk et al. 2010). In particular, work has been done in developing a portable water sensor using multiple excitation wavelengths to create an excitation-emission matrix (EEM) for detecting NAs in processed affected water (PAW) (Taschuk et al. 2010). We continue this work by miniaturizing it and applying it to soil testing.

Here we design, build and test a compact (25 mm diameter cylinder) fluorescence spectrometer for use with a cone penetrometer. We are interested in hydrocarbon and NA detection. Cone penetrometer technology is an in-situ technique that allows quick onsite measurements of soil's physical and chemical properties. A narrow probe is hydraulically pushed deep into the ground and the sensorized tip collects data as it travels, sending data back to the controller above ground. Current technology uses UV induced fluorescence (ConeTec 2012) to detect the presence of any fluorescence. Here we present a fluorescence

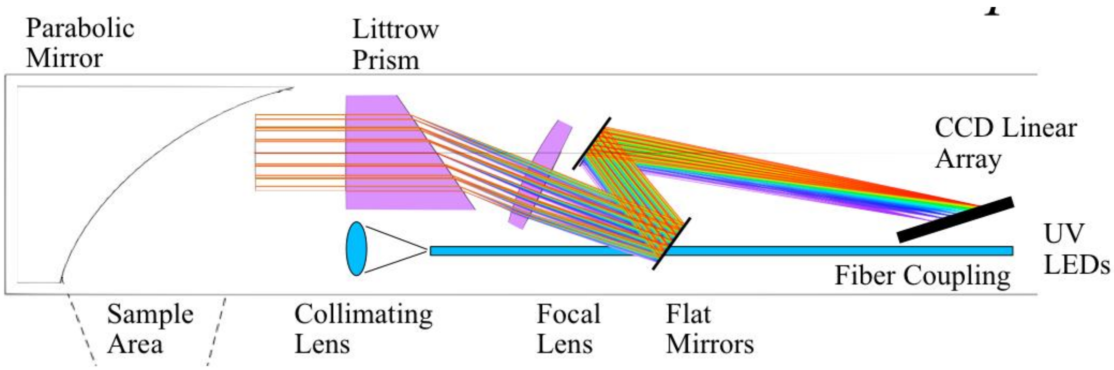

Figure 1. **Schematic and ray tracing simulation of Design 1. Parabolic mirror focuses the excitation light from fiber coupled LEDs and collected light, and collimates the emitted fluorescence. The collected light is dispersed into its spectrum using the Littrow prism and is focused onto the linear CCD array.**

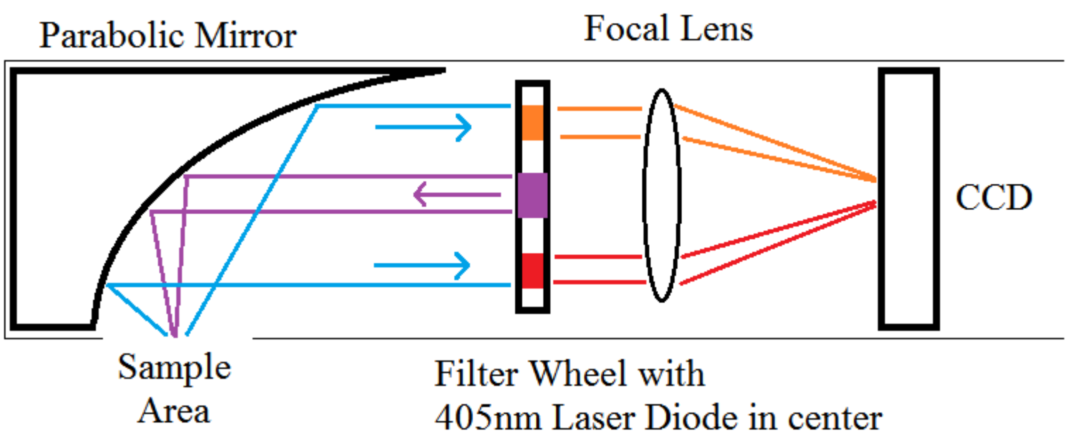

Figure 2. **Schematic of Design 2. The filter wheel holds the excitation 405 nm wavelength laser diode and the 5 long-pass filters. A parabolic mirror focuses the excitation light and collects fluorescence. The collimated fluorescence passes through the long-pass filters and is focused onto a CCD for analysis.**

spectrometer to add spectroscopic capabilities to the fluorescence cone penetrometers.

INSTRUMENT DESIGN

Design 1

Excitation

The device's initial design was a high resolution spectrometer with multiple excitation wavelengths to acquire an EEM. Seven UV LEDs (245 nm – 350 nm) which can be individually activated are used to provide this range of excitation. Custom fiber couplers were developed to efficiently transfer the UV LED emission to the fiber. The LED light is then brought to the collimating lens through the fiber, collimated and focused onto the sample using a parabolic mirror, as can be seen in Figure 1.

Collection

To collect fluorescence, the same parabolic mirror used in focusing is used to collect and collimate the sample's emitted light. Reusing the focusing optics reduces the device's footprint. The collimated light passes through a Littrow prism to spectrally disperse the fluorescence, which is subsequently focused on a Hamamatsu CMOS linear array detector (1 x 256 pixels) for a spectral range of 200 nm – 800 nm.

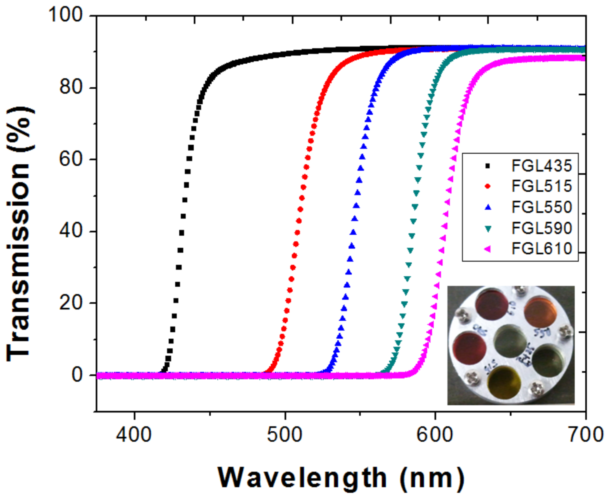

Figure 3. Transmission response of the long-pass filters. Inset shows a picture of the 'filter wheel'; center slot would have the 405 nm laser diode.

Design 2

Excitation

In the second design, the excitation source was a 405 nm wavelength laser diode. Typically one would expect weaker fluorescence when using lower energy photons, but laser diodes (250 mW) are more economical and available at a much higher output power than LEDs (≤1 mW). Another difference from the original design is that only one excitation wavelength is used. Currently this is the shortest wavelength with high power output that is economically available; however this could be easily redesigned for additional wavelengths in the future. Using the same principle as Design 1, the laser light could be focused onto the sample using a parabolic mirror as can be seen in Figure 2. For simplicity initial testing uses an externally focused laser diode.

Collection

Once again the fluorescence light is collected and collimated by the parabolic mirror. In this design the collimated light is passed through the 'filter wheel'. The filter wheel is composed of 5 Schott glass long-pass filters with increasing cut-on wavelengths (435 nm, 515 nm, 550 nm, 590 nm, 610 nm) as shown in Figure 3. The filters are 7 mm in diameter and spaced equally in the filter wheel, which has an overall diameter of 25 mm. The filters had approximately step transmission functions, and the design cut-on wavelength was defined at the 60% transmission wavelength. The filtered pattern is then focused onto a 1/3" 2D Chameleon monochrome CCD. The focusing has a dual purpose of reducing the CCD footprint and increasing the intensity of the fluorescence, thus increasing signal to noise. The CCD has a 1296 x 964 resolution with a 12 bit ADC and is USB controlled. The pattern's intensity is analyzed and can be used to reconstruct the fluorescence spectrum.

Spectrum Reconstruction

The light observed through each filter varies in intensity based on the sample's emitted fluorescence spectra and the filter's cut-on wavelength. The shorter the cut-on wavelength, the more light it will transmit and the higher intensity will be detected. The difference of intensity from two adjacent filters gives the intensity of the wavelength band between the two filters. This is similar to a band pass filter in this application.

The sample's fluorescence spectrum can then be reconstructed in bands of relative intensity in ranges between the filter cut-on wavelengths. This method's resolution is directly related to the filter number. With 5 filters we obtain the spectrum's general shape and show if there is a clear peak in the fluorescence spectrum.

RESULTS & DISCUSSION

Design 1

Optical ray tracing simulations of the design shown in Figure 1 were carried out to determine the expected spectrometer resolution. Detector placement and orientation were optimized for spectral resolution and bandwidth. One example of the simulation results is given in Figure 4, which shows a 128 pixel linear detector used to detect a spectral bandwidth of 275 nm to 607 nm.

Figure 5 shows the fluorescence spectrum of orange fluorescence paper and acridine orange dye using the 405 nm laser diode as excitation.

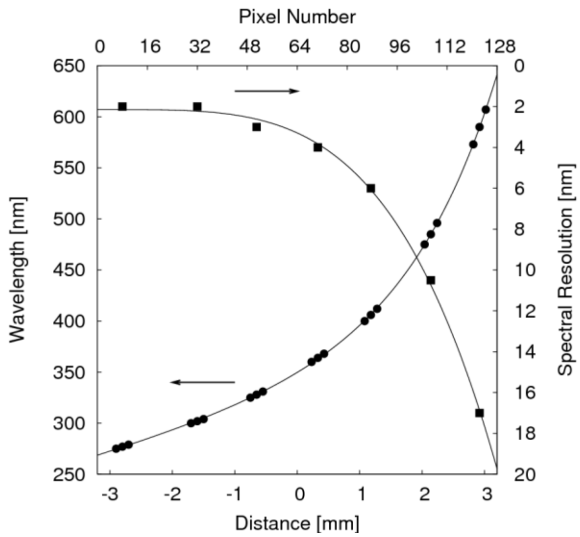

Figure 4. Optical simulation of Design 1 showing spectral resolution along 128 pixel linear detector.

This is overlapped with the spectrum measured using an Ocean Optics spectrometer.

Using a collimated white light source, 15 nm18 nm resolution was demonstrated. Unfortunately, obtaining the performance predicted by the simulation proved impossible. We attribute this to the extremely demanding alignment specifications combined with the constraints imposed by Design 1's very small form factor.

Therefore, to further develop the technology, we sacrificed potential performance for much less demanding alignment requirements; this tradeoff resulted in Design 2.

Design 2

This design was tested using various analytes such as orange fluorescence paper, crude oil, and mature fine tailings (MFT). We also tested organic compounds such as naphthalene, phenanthrene, and pyrene mixed into clay, which are 2-, 3- and 4-ring polycyclic aromatic hydrocarbons, respectively.

The measured intensities of light transmitted through adjacent filters are subtracted to find the intensity contributed by a band of wavelengths as described above. Given the filters used, the spectrum was divided into four bands: 435 nm to 515 nm, 515 nm to 550 nm, 550 nm to 590 nm, and 590 nm to 610 nm.

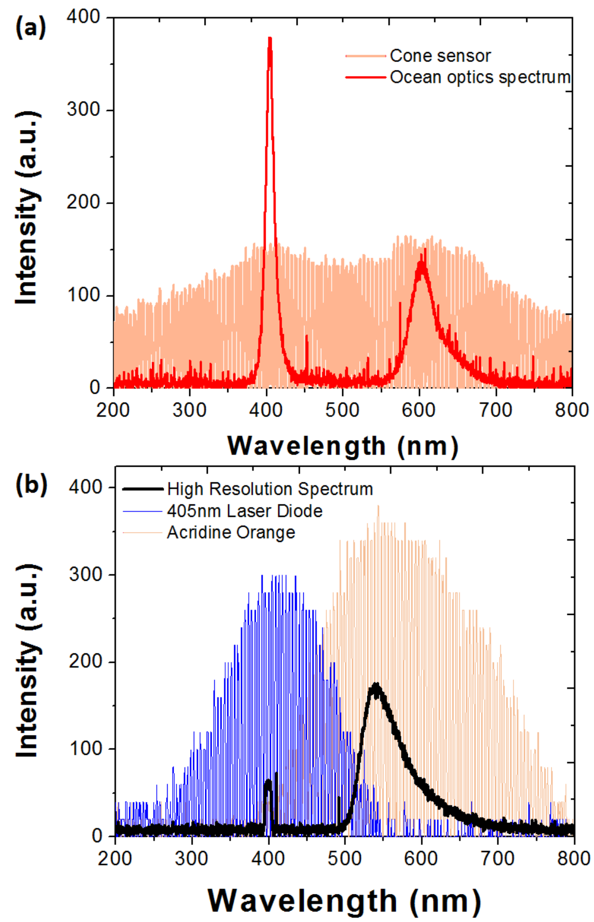

Figure 5. Fluorescence spectrum of (a) Orange Fluorescent Paper and (b) Acridine Orange dye measured using Design 1 overlaid with high resolution spectrometer measurement from Ocean Optics.

For example, if the value measured through the 515 nm cut-on filter was subtracted from the 435 nm cut-on filter, the remaining intensity value would be the contribution from photons in the 435 nm to 515 nm range. The band intensity value was then corrected using the average sensitivity of the CCD in each wavelength range. Therefore the values plotted for the device output are the normalized intensities measured in each wavelength band.

For comparison the high resolution spectrum was divided into the same wavelength bands as above. Each band was integrated, allowing direct comparison with measurements taken with our prototype device. While using adjacent filters allows us to compose an expected spectrum based on the peaks and troughs of the spectrum, additional pairings may yield additional information. Further work will be required to evaluate this potential avenue for extracting more information from the type of system prototyped here

The raw Ocean Optics spectrum is plotted for reference. In the case of the organic compounds, Ocean Optics spectrum was measured from the fluorescence of the pure reagent rather than mixed into soil due to very weak fluorescence.

Initial testing was done with fluorescent paper as the fluorescence efficiency is very high. Figure 6 shows testing results for Orange Fluorescent Paper and Crude Oil (6.3×10^{-3} concentration).

Ocean Optics measured spectrum is plotted for reference and the measured filter intensity is compared with the integrated area of the Ocean Optics spectrum in the same wavelength range. The peaks of both the paper and crude oil are clearly represented within the device's bandwidth and they are clearly distinguishable from one another. Intermediate wavelengths follow the trend of the spectrum.

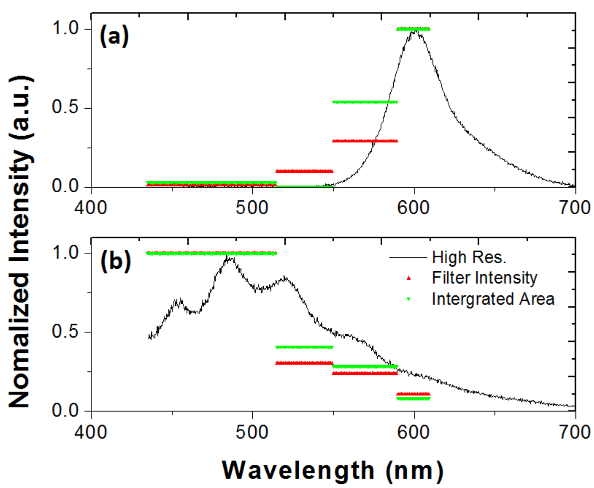

Figure 6. Comparison of analyzed filter intensity spectrum against integrated area of the high resolution spectrum of (a) orange fluorescent paper and (b) crude oil (6.3×10^{-3}). High resolution spectrum plotted for reference.

Design 2 was also tested with sample made by dissolving naphthalene, phenanthrene, pyrene in chloroform and mixing it with clay. Fluorescence from pure chloroform was not detected at the excitation wavelength and power. The mixture coats the clay and the chloroform is allowed to evaporate leaving behind contaminated clay. The clay is then packed into a cuvette and is measured. All three preparations were of approximately 10^{-3} concentration by weight. Due to very weak fluorescence of this mixture, the Ocean Optics spectrum was obtained by irradiating pure reagent with the 405 nm excitation laser. This was not a problem for the instrument and sufficient contrast was obtained with a maximum integration time of 130 ms.

Figure 7 shows the results of these three samples. The resolution of Design 2 is currently not capable of distinguishing between the naphthalene, phenanthrene and pyrene since their fluorescence emission features are mostly in the region of 435 nm – 515 nm, where there is only one filter. In particular, pyrene would be expected to be distinguishable if there was at least one more long-pass filter in this range since from the high resolution spectrum, the fluorescence peak is centered near 500 nm as opposed to naphthalene or phenanthrene where the peak is not visible.

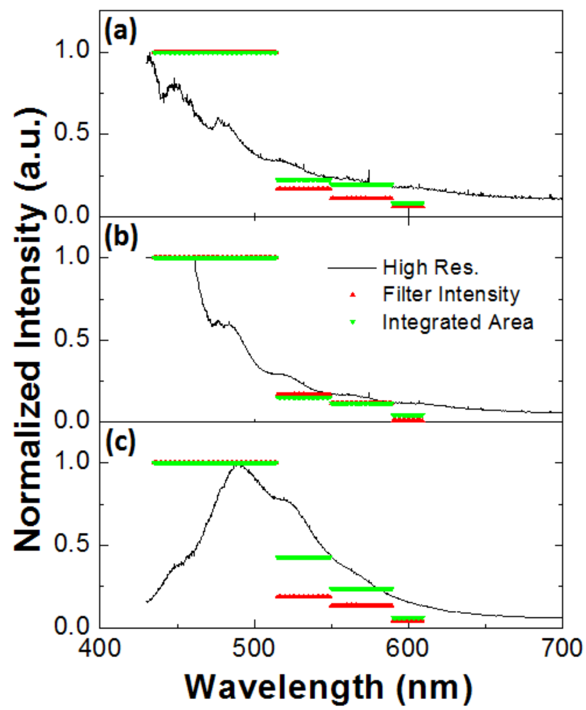

Figure 7. Comparison of analyzed filter intensity spectrum against integrated area of high resolution spectrum of: (a) naphthalene, (b) phenanthrene and (c) pyrene, all $\sim 10^{-3}$ concentration. High resolution spectrum plotted for reference.

Finally, mature fine tailings (MFT) from Albian's tailings pond were tested. The MFT had a solids content of 55% and a bitumen concentration of $3.5x10^{-3}$. Figure 8 shows the results obtained. Measured results show the same trend of having a wider fluorescence spectrum thus higher intensity in 500 nm – 600 nm spectrum range than the pure organic compounds, which are much sharper.

Limit of detection can be roughly estimated as ~1 ppm assuming linear extrapolation of the signal to noise ratio and integration time when measuring $\sim10^{-3}$ concentration samples.

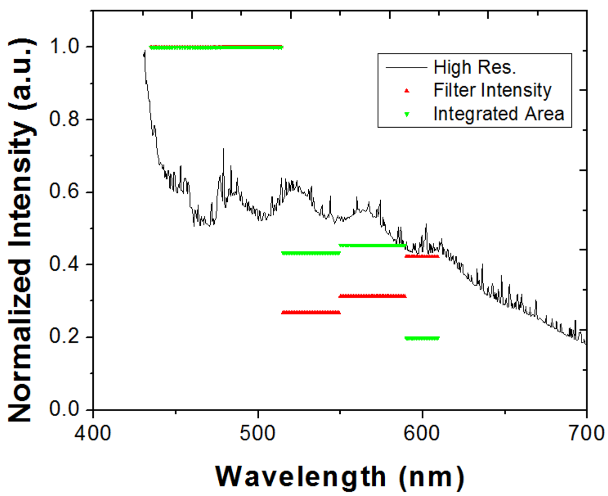

Figure 8. Comparison of analyzed filter intensity spectrum against integrated area of the high resolution spectrum of 55% solids content MFT with $3.5x10^{-3}$ concentration bitumen. High resolution spectrum plotted for reference.

CONCLUSIONS & FUTURE WORK

Two portable spectrometers for cone penetrometer applications have been demonstrated. The initial design uses a Littrow prism to achieve higher resolution and multiple excitation wavelengths to enhance chemical fluorescence differentiation; however, device performance is extremely sensitive to small alignment adjustments.

In the second design, long-pass filters provide some spectral resolution and using one excitation wavelength limits the ability to distinguish chemicals with similar fluorescence spectra. However, the device is much more robust in that it is not very alignment sensitive. Future work involves using more excitation wavelengths using laser diodes and/or LEDs and increasing the number of filters to improve fluorescence differentiation.

Based on current results, differentiation can be made between various specimens. Orange fluorescent paper is clearly distinguishable from any other spectrum. The bitumen in MFT is distinguishable by its wider fluorescence emission spectrum versus pure organic compounds which was much sharper. This offers the advantage of spectral resolution for small, portable, fast measurement field devices.

ACKNOWLEDGEMENTS

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CORRELATIONS OF SHEAR STRENGTHS OF SOFT OIL SANDS TAILINGS ASSESSED BY DIFFERENT IN SITU METHODS

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ABSTRACT

This paper presents correlations of shear strengths measured by three different field testing methods in tailings deposits at the Shell Canada Tailings Testing Facility at the Muskeg River Mine site. The testing was conducted using the Cone Penetration Test, the Ball Penetration Test and the Field Vane Test. The materials tested consisted of: (a) two Non-Segregating Tailings deposits with different gradations, and (b) two Thickened Tailings deposit with different gradations.

Characterization of the deposits was performed while they were at different stages of consolidation, but generally soft, with shear strengths lower than 20 kPa. The strength conversion factors N_{kt} and N_{ball} were calculated for Cone and Ball Penetration Test results, respectively, for each tested deposit, with the Field Vane Test adopted as the reference measurement technique. The 'customized' strength conversion factors were, on average, close to the commonly applied values N_{kt} = 15 and N_{ball} = 10.5 for the Non-Segregating Tailings, but deviated from typical values for the Thickened Tailings. For all tailings types evaluated, the scatter of data was significant with the coefficients of variance of up to 50% for the strengths calculated.

It was concluded that more work is needed to collect additional experience with the field methods for strength measurements in soft oil sands tailings. Caution should be exercised when making generalizations, which should be limited to specific tailings types.

INTRODUCTION

This paper presents correlations of shear strengths measured by three different field testing methods in tailings deposits at Shell Canada Ltd. (Shell) Testing Tailings Facility (TTF) at the Muskeg River Mine (MRM) site. The testing was conducted using the Cone Penetration Test (CPT), the Ball Penetration Test (BPT) and the Field Vane Test (FVT).

TAILINGS TYPES AND TESTS

Table 1 presents summary data on Shell's tailings deposited at TTF [1]. The materials tested consisted of:

- two Non-Segregating Tailings (NST) deposits with different gradations (Cells 5 and 6);
- one Thickened Tailings (TT) deposit (Cell 4); and.
- one Treated Thickened Tailings (TTT) deposit, of a paste consistency (Cell 1).

Both TT and TTT were produced in a paste thickener, the former with the paste thickener operating in a high-rate thickener regime. The NST was made by mixing the paste thickener underflow with dewatered coarse tailings. The sand-to-fines ratios were originally determined on the 74-micron basis, from the fines contents FC_{74} that were obtained by wet sieving. The fines content FC_{44} and the sand-to-fines ratios SFR_{44} on the 44-micron basis were calculated from a material-specific correlation with the SFR_{74}.

Characterization of Shell TTF deposits was performed at the following approximate ages:

- 10 months for NST in Cell 6 and TTT in Cell 1,
- 3 months for NST in Cell 5 and
- 1, 8 and 10 months for TT in Cell 4.

Strength testing was performed at three locations in each cell: upstream (A), middle (B) and downstream (C), approximately at the quarters of a cell length. The strengths were investigated using three field methods: FVT, CPT and BPT.

The FVT testing was performed by Geoforte, Edmonton, using custom equipment with vane sizes D:H (diameter to height) of 4 x 8 cm and 6 x 12 cm. Geoforte also performed sampling of the deposits for determination of geotechnical index properties, which was conducted by Shell's Calgary Research Centre (CRC).

The penetration tests were performed by Conetec Investigations Ltd., Vancouver. The CPT sounding was performed in general accordance with the ASTM standard D5778, using a cone equipment with a maximum tip capacity of 200 bars, tip area of 15 cm^2 and friction sleeve area of 225 cm^2. The BPT sounding was performed using a standard cone pushing frame with a maximum BPT tip capacity of about 80 bars, and the standard ball of 11.3 cm in diameter, with projected area of 60 cm^2. The BPT penetration rate was a commonly used value of 2 cm/s.

CHARACTERIZATION RESULTS

The calculated strength data for all three test methods are presented in Figures 1-3 for Cells 4, 5 and 6. All three locations in a cell are shown in the same diagram, for comparison.

Strength data for TTT in TTF Cell 1 are not presented. The FVT testing was not performed at the mid-cell location B, and it had to be repeated on location C. The vane results in Cell 1 were assessed as invalid – they were very noisy, the curves wavering, and the peak strengths generally low - below 2 kPa, so it was decided not to include them in this analysis.

No BPT peak testing was performed at location A in Cell 6.

Forty five NST and fifteen TT strength data points were used in this analysis. Only peak strengths were considered.

Calculation of strengths

The strengths presented in Figures 1-3 were calculated using the formulae:

$$s_u^{BPT} = \frac{q_{net}}{N_{ball}} = \frac{q_b - \left[\sigma_{v0} - u(1-a)\right] \cdot \dfrac{A_s}{A_p}}{N_{ball}} \qquad [1]$$

where: q_b is the ball penetration resistance, σ_{v0} the total vertical overburden stress, u the pore pressure measured just behind the joint between the ball and push rods, A_s the cross sectional area of the cone shaft, and A_p the projected area of the ball;

$$s_u^{CPT} = \frac{q_{net}}{N_{kt}} = \frac{q_t - \sigma_{v0}}{N_{kt}} \qquad [2]$$

where: q_t is the measured tip resistance corrected for unequal end area pore pressure effects on the cone tip, and N_{kt} is the undrained shear strength conversion factor; and

$$s_u^{FVT} = \frac{6T_{\max}}{7\pi D^3} \qquad [3]$$

where: T_{max} is the maximum value of measured torque (corrected for the apparatus and rod friction), and D is the vane diameter.

The CPT and BPT strengths were calculated from the net tip penetration resistances using the common values of the strength conversion factors N_{kt} = 15 and N_{ball} = 10.5, respectively.

No correction for the rotation rate was applied to the vane data. It should be noted that the vane test were performed with non-uniform rotation rates, so that a correction for shear rate sensitivity should be applied to normalize the data set. The vane rotation rates were, in general, higher than the recommended standard rates; occasionally, they were up to an order of magnitude higher. Therefore, the normalization with respect to shear rate would reduce the strength data used. However, the reduction would be small, about 10%, and would not substantially affect the correlations.

DATA SELECTION

Preliminary Data Screening

Initial screening of the TTF data included the following:

- *Indurate surface crust*: None of the data points fall within a strong, partially dried surface crust, extending to about 0.4 m of depth.
- *Disturbed materials zones during testing*: Some of the strength tests were located near the elevations of CPT pore pressure dissipation tests, which were found to affect

the response of soil in continued penetration. Similar is valid for the cycling tests in BPT for determination of the remoulded strengths. It was difficult to estimate to what extent the tailings response was affected by the proximity to these, potentially disturbed zones, so no points were discarded.

- *Test performance*: Similar comments are valid when pushing rods were added or removed during CPT and BPT, with characteristic spikes in the penetration resistance curves at regular 1-metre intervals. Although some vane tests were located in the vicinity of these depths, none of the CPT/BPT points were discarded from further analysis.
- *Pore pressure response as indicator of mode of deformation (undrained / drained)*: Dynamic pore pressure curves were inspected for visible deviations from the hydrostatic pore pressure lines, but it was concluded that this may not be decisive, as explained in Section 4.2.

Only one Shell TTF strength test point was actually discarded: the lowest point at the upstream location (A) in Cell 6, which was suspected to be a mixture of NST tailings with the cell bottom material (lean oil sand or overburden soil).

Estimation of Mode of Stress-Strain Behaviour

No specific screening for the stress-deformation mode of behaviour (drained, undrained or partially drained) was performed on the vane data. The penetration results were screened for the mode of stress-deformation behavior.

The majority of CPT data shows generation of positive pore pressures during penetration, which should correspond to the expected volumetric deformation behavior of the contractant structure of soft NST tailings. Sometimes, this response is not apparent, but the data cannot be discarded because of a possibility that pore pressure generation can be suppressed by dilation of coarse granular structure (coarser material is typically found in the neighbourhood of a discharge point). Therefore, individual data points were not examined for pore pressure response, but two more general, global approaches were chosen.

It is generally accepted by the geotechnical community that, at a standard rate of CPT penetration of 2 cm/s, undrained response will occur if the permeability of the soil is less than about 10^{-5} cm/s [2]. Using the pore pressure dissipation tests during CPT in the actual deposits [3] and applying four different methods for estimation of the permeability from CPT data [3, 4 and 5], it was found that the TTF NST permeability should be within the range of 10^{-8} to 10^{-6} cm/s, which is an order of magnitude below the undrained behaviour limit value.

An alternative approach to an estimate of the NST mode of behaviour was taken from reference [6]. For shallow circular foundations in calcareous silts and sands, it was found that the limits for undrained and drained behaviour corresponded to the 'non-dimensional velocities' V of about 0.01 and 30, respectively. It was found later that these limits may be used as a first estimate of the degree of drainage in the CPT field tests [7]. The following formula is used for the CPT *V* factor, and a similar one for the BPT V factor:

$$V = \frac{v \cdot d}{c_v} \qquad [4]$$

where: v is the penetration rate, d is the cone diameter and c_v the coefficient of consolidation.

The values for c_v in Shell NST tailings were estimated from CPT data and varied within the range 0.7 – 3.8 cm^2/min. The normalized velocity range for CPT was calculated as V_{CPT} = 356 – 1934, which is much above the undrained behaviour criterion. The corresponding *V* range for the BPT test was determined as V_{BPT} = 138 – 749, again higher that the undrained behaviour criterion.

Presence of Air Bubbles (Gassy Soils)

The longitudinal velocities v_p in Shell's NST tailings were consistently low (200 - 400 m/s or about 15 - 25% of the velocity in water), which was explained by partial saturation of tailings stream due to air bubbles entrained during pumping from the TTF plant and discharge into the cell. These bubbles were visible in the cores taken from the tailings, but the amount of gas could not be measured during sampling. The corresponding air volume fraction was numerically estimated to be within the range 0.1-0.5% and the degree of saturation greater than 99%. However, these results were taken as an illustration of low sensitivity of the applied model rather than as an indicator of the undrained behaviour.

It should be noted that the stress-deformation of gassy soils can exhibit phenomena like subdued excess pore pressure response, which will imply different resistance to penetration and other field strength tests. That effect could not be investigated during NST tailings characterization. However, the volume of air bubbles could be more precisely estimated in TT in Cell 4 based on stereological image processing techniques. The air volume was found between 2% and 6% and the corresponding degree of saturation between 92% and 97%. Based on certain literature data on gassy sands, for example [8], the specimen with the initial degree of saturation greater than 90% responded in a strain-softening manner similar to that of fully saturated specimens. Therefore, the measured strengths were likely undrained, with a possibility that some results were obtained in a partially drained mode of behaviour.

STRENGTH CORRELATIONS

The vane results were assumed as the primary data set, i.e. the FVT was adopted as the reference method. The basis for this decision is that the vane method measures strength directly, while the penetration methods estimate strength through correlations.

The following approach was adopted and consistently implemented in data interpretation and comparison. For each testing location A, B, C in a cell, the CPT and BPT penetration resistances, corresponding to the elevations at which the vane tests were performed, were calculated as the averages over a 20-cm interval centered at the vane test elevation.

The correlations between FVT and CPT / BPT data were established separately for the two NST cells. The correlations were carried out using inverted forms of the equations for undrained strength:

$$N_{ball} = \frac{q_{net}^{BPT}}{s_u^{FVT}} \qquad [5]$$

$$N_{kt} = \frac{q_{net}^{CPT}}{s_u^{FVT}} \qquad [6]$$

Figures 4-7 present the plots of correlation data: undrained vane strengths and the net tip penetration resistances. The strength conversion factors N_{kt} and N_{ball} were obtained as the slopes of the linear best-fit lines for these data sets. They are presented in Table 2.

Calculated N_{ball} factors for two Shell NST materials are very close to the commonly used value of 10.5 and, on average, identical to it. Therefore, using the standard value of N_{ball} = 10.5 for BPT data interpretation results in strengths consistent with the FVT measurements.

The N_{kt} factors for Shell NST are a little below the commonly used value of 15, which means that the 'standard' CPT strength prediction with N_{kt} = 15 will, on average, slightly underestimate the shear strengths obtained by the FVT.

ANALYSIS AND CONCLUSIONS

Comparison of Strength Correlations for Shell NST and Natural Sandy Soils

The strength conversion factors N_{ball} and N_{kt} for Shell's NST materials are closer to the commonly applied values, and their variation is smaller than in natural sandy soils [5]. This can be expected in the pilot deposits, where a tighter control over tailings production and deposition can be exercised. It is also consistent with a relatively high solids content of Shell NST and a smaller variation of its SFR compared with natural soils, where soft and widely graded deposits show higher variation in properties and behaviour than compact and uniform deposits. A caveat for the use of the pilot data for the larger scale operations is that the larger variation of properties at a larger scale of operations may diminish the predictive capability of a model developed at a smaller scale.

Uncertainty in Strength Conversion Factors

The range of variation of N_{ball} and N_{kt} factors for Shell NST material is too wide for a single value to be chosen as representative of the whole tailings type. Take for an example the BPT penetration data. Although the N_{ball} = 10.5, consistent with the commonly used value in practice, could be recommended, the variation of the N_{ball} values in the range of 6 to 15 implies that the coefficient of variation for the calculated strengths would be about 50%. This inherent variation in the parameters obtained from statistical considerations (correlations) may present risks without cautious application.

Therefore, the field penetration tests may be safely applied for strength assessment of a tailings deposit when:

- the deposit is well understood, including: zonation of different tailings, possible stratification, geotechnical index properties, strength development processes (sedimentation, consolidation, desiccation, etc.) and similar;
- a strong correlation is established between a penetration test results and actual, operational strengths obtained or confirmed by other, different methods;
- there is sufficient understanding of influences on the tailings strength and the adopted strength correlation, such as: tailings type; particle size distribution; plasticity of fine fraction, rate sensitivity, etc.;
- a value of N_{ball} or N_{kt} is selected such that it ensures that the strength equal to or greater than the calculated value is present in the majority of the deposit, within a margin of error sufficient for the purpose of strength assessment or another specific application;
- updates of the above data are performed on a regular basis.

Representativeness of Field Vane as Reference Method for Strength Measurement

The FVT method is probably the most direct means available to measure strength in soft tailings deposits. The method involves uncertainties in performance and data interpretation, including: rotation rate sensitivity of certain soils/tailings, soil anisotropy, progressive failure, empirical corrections for actual in situ strength, vane size, operator competence, etc.

Back calculations from case histories - actual failures of geotechnical structures - showed that in many cases vane strengths overpredicted the strength values at failure for natural soils. This led to the concept of correction factors that should be applied to raw measured FVT data to adjust calculated FVT strengths and obtain reliable strength values for design stability analyses; for example see [9]. However, practical implementation of this concept faced numerous difficulties, so that application of correction factors has not become an element of the FVT standardization. Although vane is extensively used for strength measurement in the oil sands industry, the authors are not aware of any assessment of the need for correction factors for FVT data when applied to oil sands tailings problems.

Insufficient information exists on application of FVT in soft oil sands tailings. The magnitudes of the mentioned influential factors in soft tailings have not been quantified. Additional factors affecting FVT results in oil sands tailings should be considered, such as the percentage of bitumen and chemical treatment of tailings. It is our opinion that more work is needed to collect additional experience with FVT application in soft oil sands tailings. Generalization should be postponed, and FVT should be used as the only strength testing method only after careful consideration of all available information for an individual deposit. It is recommended to regularly use FVT in conjunction with the penetration methods.

Particular attention should be paid to the performance of the vane test as seemingly the adopted reference method in oil sands industry. It is suspected that occasional large differences between FVT strengths and the CPT / BPT estimated strengths can be attributed to the variability in the execution of the FVT.

Strength Correlation for Shell's TT (Cell 4)

The conversion factor N_{ball} = 13.5 for peak strength of TT in Cell 4 is significantly higher than the commonly used N_{ball} = 10.5. This variation may be attributed to finer gradation, and the presence of air and bitumen.

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Table 1. TTF Tailings Deposits, Deposition and Characterization Data.

Tailings	TTF Cell	End of Deposition	Date of Testing	Solids content (%)	SFR_{44}
Shell TTT	1	Dec. 04, 2007	Oct. 15-18, 2008	65 - 80	0.4 - 1.3
Shell TT	4	Nov. 09, 2009	Dec. 7-11, 2009; Jul. 12-16, 2010; Sept. 20-25, 2010	28 - 51	0.2 – 3.0
Shell NST Nominal SFR_{74}=2.8 (measured) Nominal SFR_{44}=4.8 (estimated)	5	Jun. 25, 2008	Sept. 20 – Oct. 20, 2008	82.5 - 84.5	4 – 5
Shell NST Nominal SFR_{74}=2.4 (measured) Nominal SFR_{44}=4.1 (estimated)	6	Nov. 21, 2007	Sept. 24-28, 2008	83.5 - 85	2.5 – 5

Table 2. Strength Conversion Factors for TTF NST and TT.

Tailings	TTF Cell	CPT strength conversion factor N_{kt}		BPT strength conversion factor N_{ball}	
		Average	Range	Average	Range
Shell NST SFR_{44}=4.8 (estimated)	5	14.5	10 – 23	10.1	6 – 15
Shell NST SFR_{44}=4.1 (estimated)	6	13.4	8 – 19	10.8	8 – 14.5
Shell TT	4	n/a	n/a	13.5	11.5 - 20

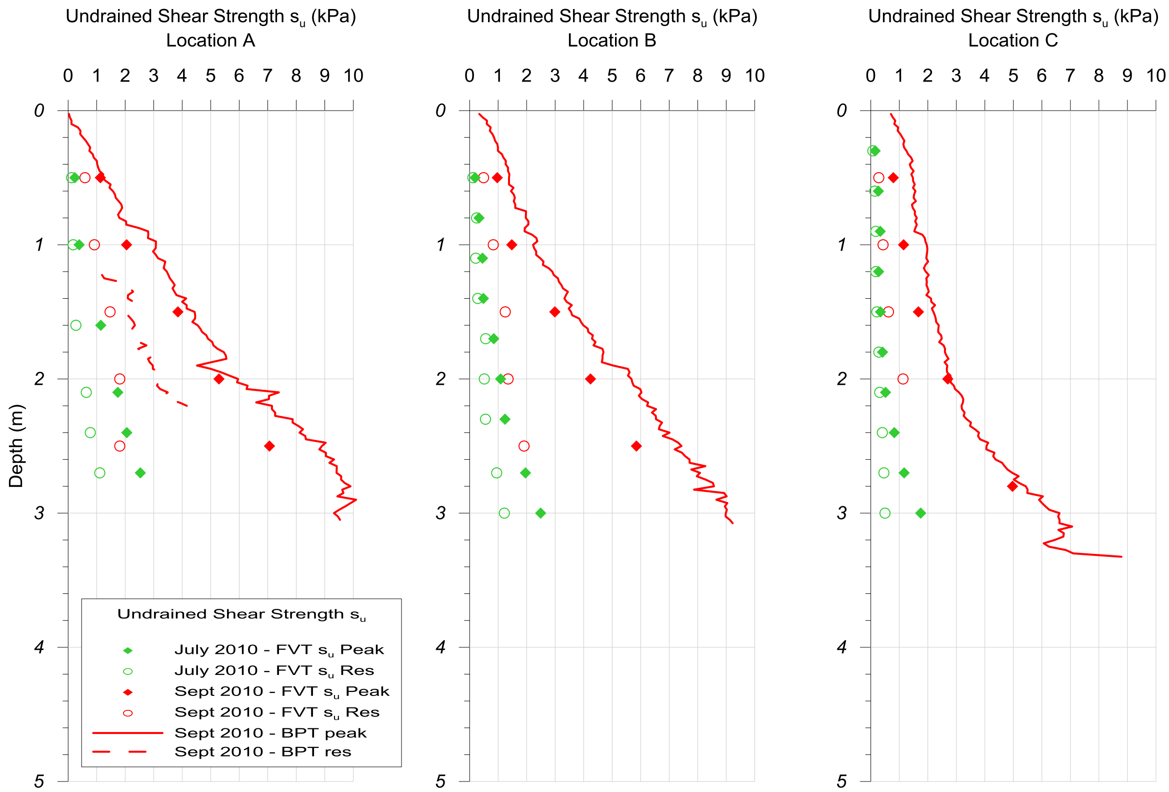

Figure 1. Shear Strengths in TTF Cell 4 (TT).

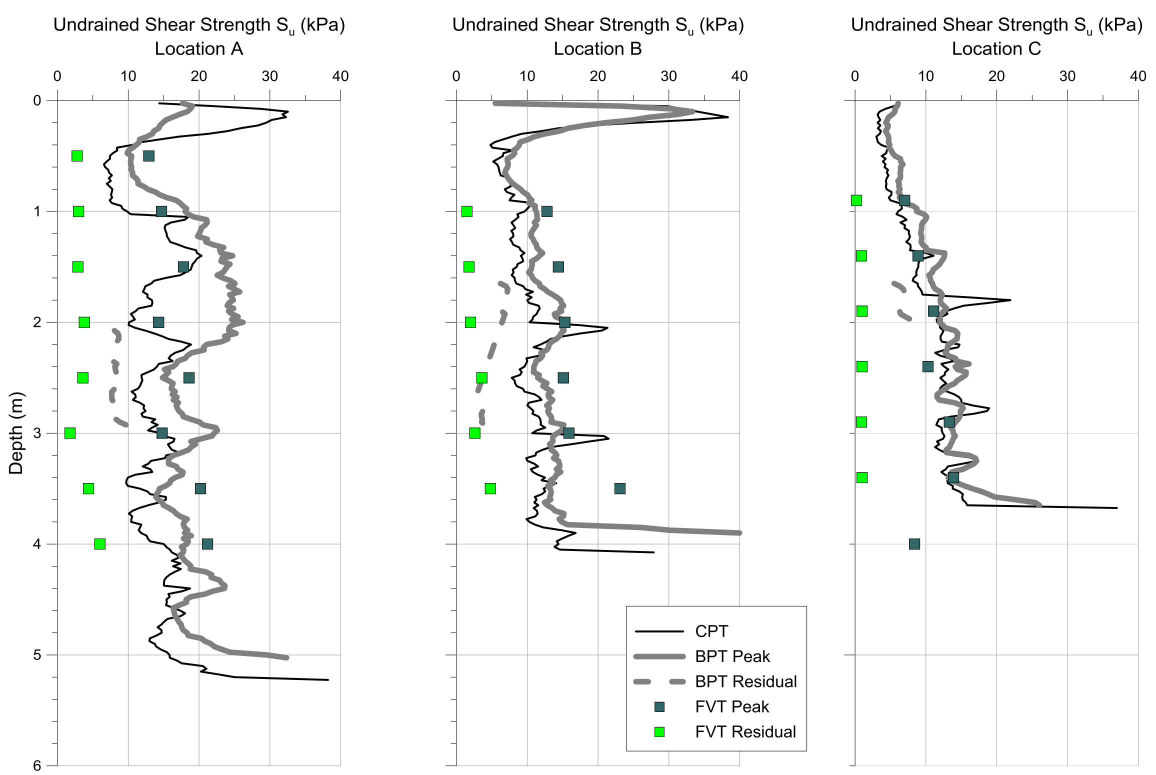

Figure 2. Shear Strengths in TTF Cell 5 (NST, SFR_{44} = 4.8).

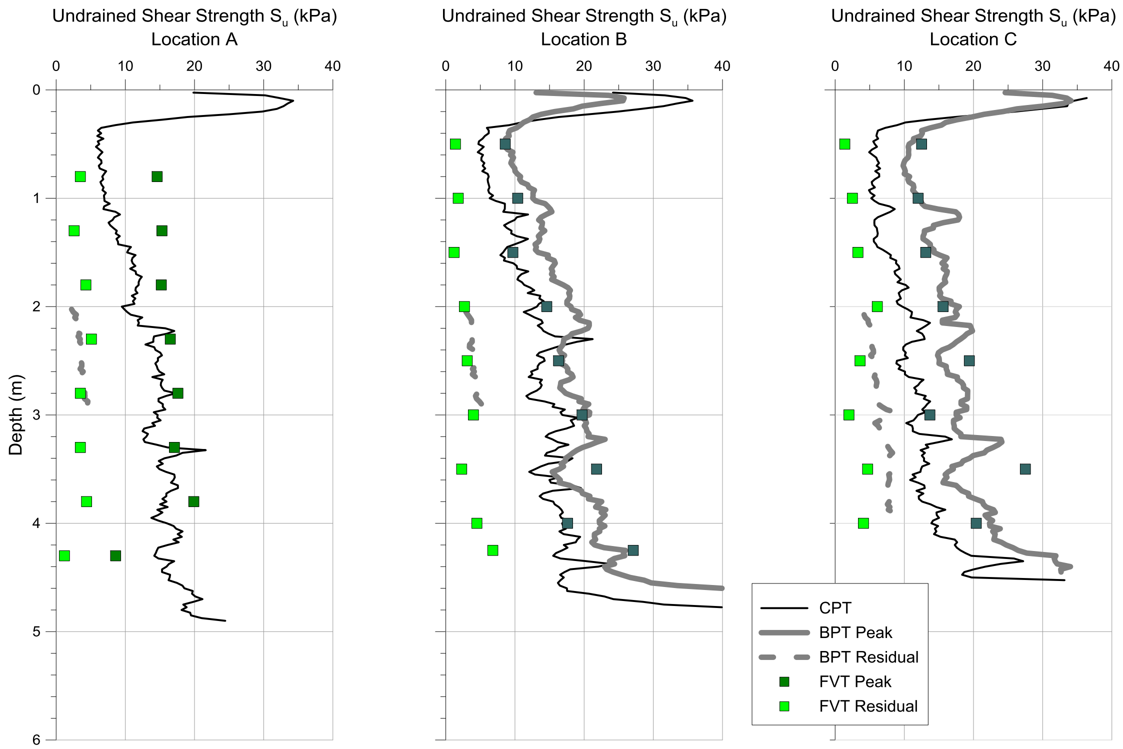

Figure 3. Shear Strengths in TTF Cell 6 (NST, SFR_{44} = 4.1).

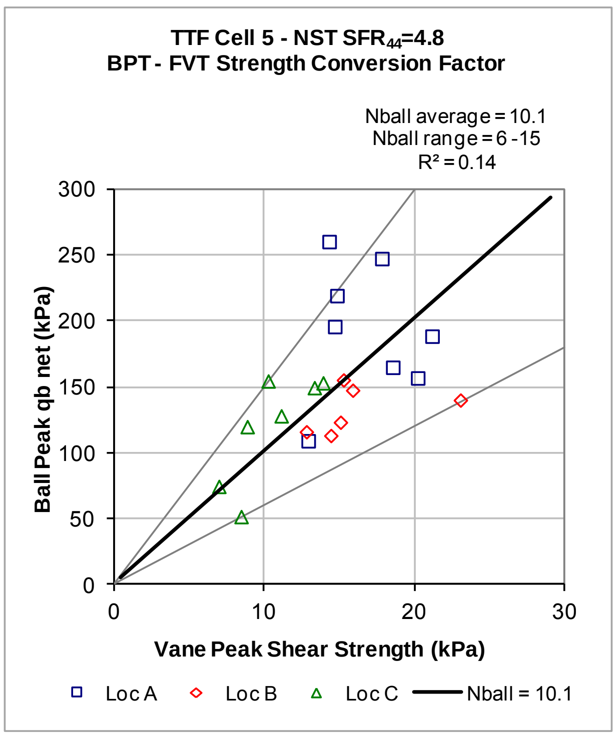

Figure 4. Cell 5 - Determination of N_{ball} factor.

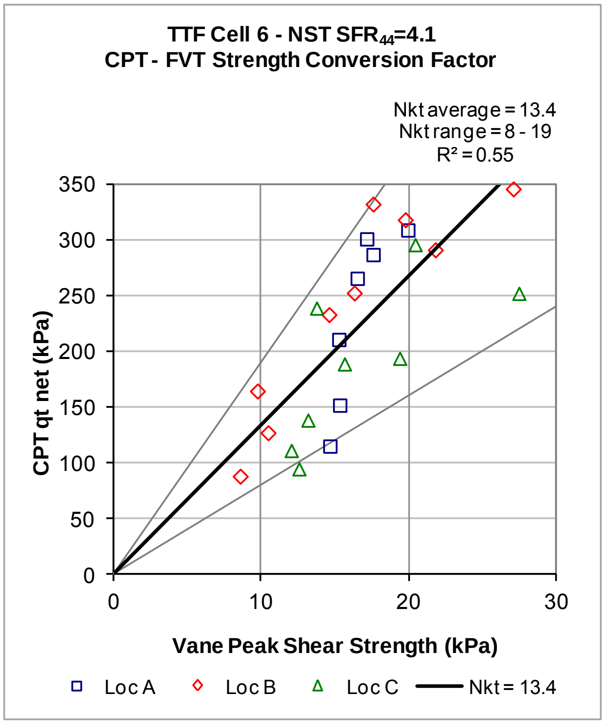

Figure 5. **Cell 6 - Determination of N_{ball} factor.**

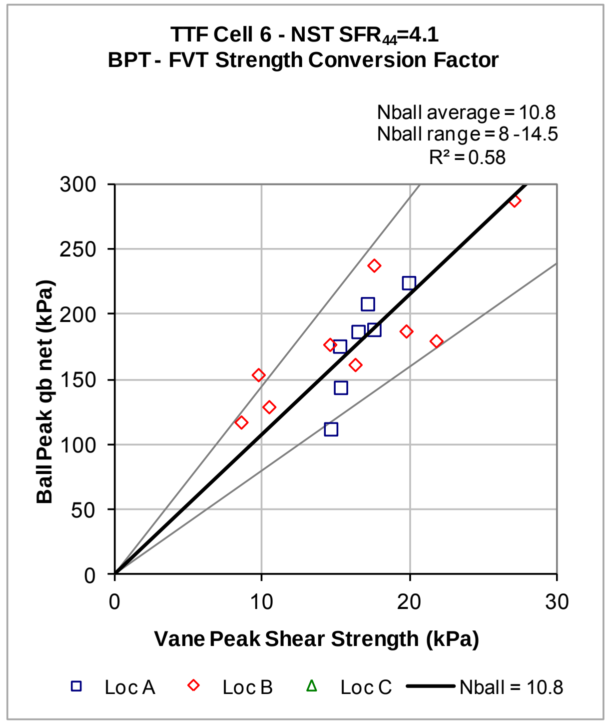

Figure 7. **Cell 6 - Determination of N_{kt} factor.**

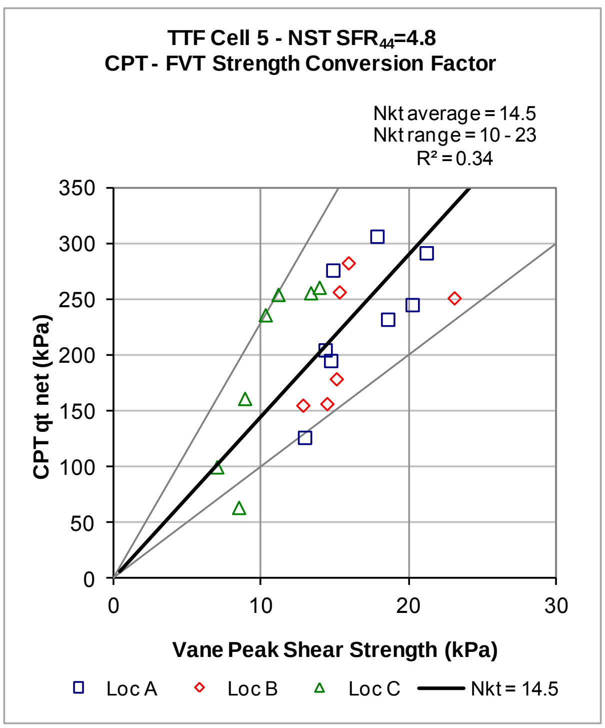

Figure 6. **Cell 5 - Determination of N_{kt} factor.**

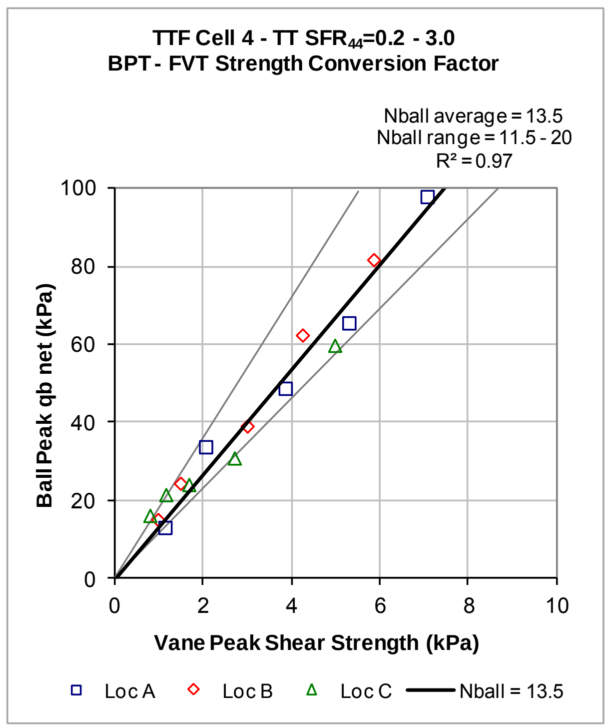

Figure 8. **Cell 4 - Determination of N_{ball} factor.**

TIME DOMAIN REFLECTOMETRY (TDR) OIL SANDS TAILINGS WATER CONTENT MEASUREMENTS: TEXTURE EFFECT

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ABSTRACT

Soils texture effect on time domain reflectometry (TDR) water content measurements of oil sands tailings are investigated in this paper. The results indicate that the TDR water content measurement is influenced by the residual bitumen and percent clay in the tailings. At a given solids content, the dielectric constant of high clay content and/or high bitumen content tailings are less than for a slurry without residual bitumen or tailings with low percent clay. The dielectric constant-volumetric water content relationship of slurry significantly differ from the commonly used Topp's et al. (1980) calibration equation, Topp's equation underestimate the volumetric water by as much as 28%.

INTRODUCTION

The space and time variability of water content are important parameter for tailings characterization and management. Traditional methods such as gravimetric, neutron probe, γ -ray and X-ray probes are time consuming and are expensive for a continuous water content profile measurements (Herkelarth et al. 1991). Time domain reflectometry (TDR) have been used quite extensively to measure the water contents of soils in water resource, soil physics, agricultural and geotechnical engineering professions (Benson and Bossher, 1999; Yu and Drnevich, 2005). The TDR is a non-destructive, cost effective, fast, easily automatable and a sufficiently accurate method for short or long-term water content monitoring (Yu and Drnevich, 2005; Benson and Bosscher, 1999; Herkelrath et al. 1991). The large contrast in the dielectric constant between water and soil solids makes the TDR a viable method for water contents measurements (Topp et al., 1980; Benson and Bosscher, 1999). The apparent dielectric constant of water is around 81 (at 20° C) and soils a solid is 3 to 7 (Drnevich et al., 2005; Topp et al., 1980). The TDR is an electromagnetic technique that measures the dielectric constant of the soil via the travel time of an electromagnetic wave within the soil (Siddiqui, 2000, ASTM 6565 (05)).

The TDR can have an important application in oil sand industry both in the field and in the laboratory. In the field, the temporal and spatial variation of water content in tailings ponds/disposal pits can be monitored continuously by installing TDR's at different depth and locations. Installing TDR's at strategically location along the extraction tailings streams helps in monitoring the variability of water contents in the extraction tailings. The solids contents of cyclone overflow, cyclone underflow and thickener under flow tailings can be monitored and controlled by use of TDR's installed at the discharge and intake of these tailings treatment units. TDR installed at different depth and locations in the in-situ (field trial) tests can be used for fast and continuous water content measurements. Since water content can be related to shear strength and degree of consolidation, the TDR measured water content can be used to indirectly monitor the progress of consolidation and the development of shear strength in the storage ponds/disposal pits. In the laboratory, the TDR can be used to monitor the water content profile of tailings during the sedimentation and self-weight consolidation tests (such as settling column and centrifuge tests) for a better understanding of the settling behavior of tailings and for deriving consolidation parameters from the self-weight consolidation tests.

The composition of oil sands tailings varies with quality of ore, the extraction method and efficiency (FTFC. 1995). The presence of residual bitumen, the addition of solute, variation in composition may affect the TDR measurements of water content. Because of the important potential application of TDR in the oil sands industry and the variability of tailings compositions the study reported on in this paper focuses on studying theses influences on oil sands tailings water content measurement. The objective of the study is also to define and examine the dielectric constant-water content relations for use with oil sands fines and fines-sand mixture tailings. Identifying and quantifying the influencing parameters can broaden the use of TDR for measuring the water content of oil sands tailings both in laboratory and in field application.

MATERIALS

Oil sands fines tailings from Albian Sands Energy Inc, commercially available Kaolinite and Devon silt are used to study the effect of soil texture on TDR water contents measurements. The Albian tailings are: mature fine tailings from Muskeg River Mine tailings pond from a depth of 7.5 m (Albian_7.5) and from a depth of 15 m (Albian_15); Albian thickener underflow tailings (TUT); and fines-sand mixture at the sand content of 70% (AL_70) and of 50% (AL_50). The fines-sand mixtures were prepared by mixing Albian beach sand with Albian mature fine tailings from 15 m depth (Albian_15). Devon silt and commercially available pulverized kaolinite are used to study TDR water content measurements of slurry with no residual bitumen for comparison with oil sands tailings containing residual bitumen. To broaden the clay and silt composition in the oil sand fine tailings in study of soil texture effect on TDR water measurements silt rich and clay rich fines tailings were prepared using Albian_15. The clay rich and silt rich MFT were prepared by diluting Albian_15 to 5% solids and letting the slurry sediment to form layers of material of different composition; the top portion of sediment yields a clay rich sediment and the middle layer results in a silt rich fine tailings. The physical characteristics of the materials used in the study are summarized in Table 1.

Table 1. Characteristics of Materials.

Material	Bitumen [%]	Sand [%]	Silt [%]	Clay [%]
TUT	3.5	35	30	35
Clay rich MFT	2.8	2	44	54
Silt rich MFT	2.5	5	55	40
Kaolinite	___	2	36	62
Albian_15	7.0	9	49	42
Albian_7.5	1.0	1	42	57
Devon silt	___	20	55	25
AL_50	3.0	50	27	23
AL _70	2.2	70	16	14

TDR MEASURING PRINCIPLE

The TDR measuring principle is based on generating a fast time pulses which propagate along coaxial cables to the TDR probe installed in the soil (Benson and Bosscher, 1999). Due to the difference in the impendence characteristics of the metallic TDR probe and soil, wave forms are reflected along the cable to the analyzer as illustrated in Figure 1; the first reflection is when the pulse from the metallic probe in contact with the soil, the second inflection is of the signal travel from the soil back (O'Connor and Dowding, 1999). The apparent dielectric constants (Ka) of the soil/slurry can then be determined from the apparent distance between the inflection points of the waveforms (La) and the physical length of the TDR probe (L) (Figure 1 and Equation 1). The water content of soils is then derived from soil-specific or 'universal' apparent dielectric constant-water content calibration equations (Topp et al., 1980; Benson and Bosscher, 1999).

$$Ka = (\frac{La}{L})^2 \qquad [1]$$

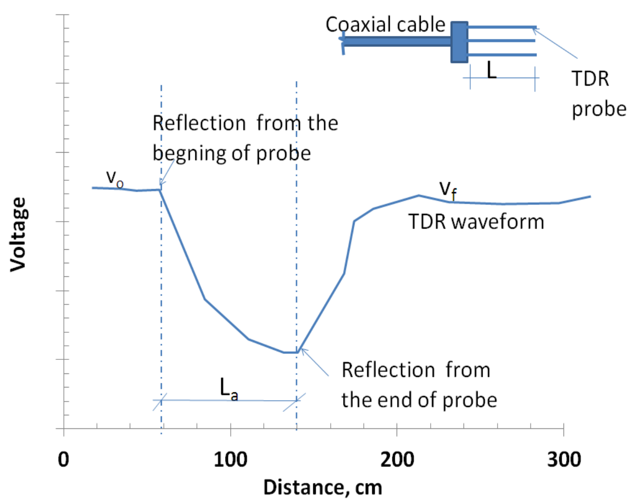

Figure 1. TDR Probe and Waveform modified from (Benson and Bosscher, 1991).

TDR COMPONENTS

The components of the TDR that were used in this study are shown in Figure 2. The TDR device includes: Reflectometer , TDR probe (rods), a connection coaxial cable, multiplexer, data processor and data logger (Figure 2). The Reflectometer is used to generate the pulses; the multiplexer is used to monitor multiples probes using a single reflectometry (Benson and

Bosscher, 1999). The RS-232 ports are used to transfer data to personal computer. The TDR probe (rods/waveguide) consists of three metal rods used to measure the dielectric constant when inserted in to slurry/soils.

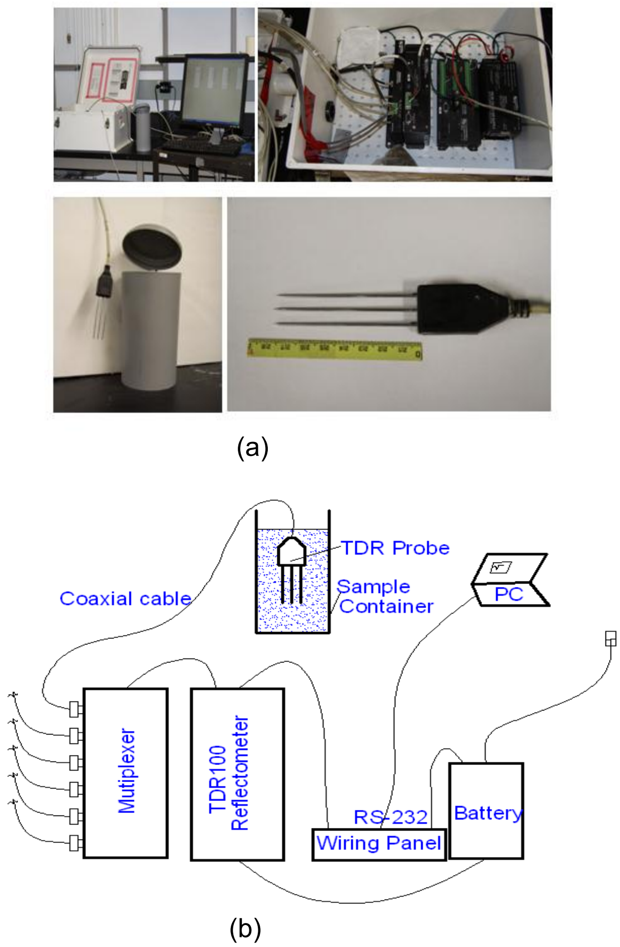

Figure 2. a) Photographs of TDR components; b) Schematic setup of soil moisture measurement in TDR technique.

SOIL/SLURRY TEXTURE

Topp et al. (1980) established an empirical relationship between an apparent dielectric constant and volumetric water content of soil (Equation 2). The equation set the foundation for use of TDR for soils water content measurements. The equation was derived from tests conducted on four different agricultural types of soils (sandy loam to heavy clay) and glass beads. The equation is considered "universal' and was later adapted by ASTM D6565 (05) standards. Following Topp's equation, different forms of the calibration equation have been proposed either based on tests conducted using different materials or based on the principles of mixing the dielectric constant of air, water and solids according to their volume proportion. Herkelrath et al. (1991); Nadler et al. (1991); Liedu et al. (1986) for example proposed different form of the calibration equation based on tests conducted on different types of soils. Mixing formulas were for examples suggested by Dirksen and Dasberg (1993); Whalley (1993); and Roth et al., (1990). The mixing law suggested by Roth et al. 1990 given in Equation 3 is based on three phases (water, air and solids) and assumes no interaction between water and soil. Wright et al. (2001) summarizes some of the calibration equations proposed by different authors.

$$\theta_v = 4.3x10^{-6} K_a^3 - 5.5x10^{-4} K_a^2 + 2.92x10^{-2} K_a - 5.3x10^{-2} \quad [2]$$

Where θ_v is the volumetric water content (the ratio of volume of water to total volume of soil) and Ka is the apparent dielectric constant of the soil.

$$Ka = [\theta_v K_W^2 + (1-n) K_s^{\beta} + (n-\theta_v) K_{air}^{\beta})^{1/\beta} \quad [3]$$

Where Ka is the apparent dielectric constant of the composite mixture; θ_v is the volumetric water content (the ratio of volume of water to total volume of soil); n is the soil porosity; K_w, Ks, and K_{air} are dielectric constant of water, soil solids, and air respectively; and β is a geometric factor that depends upon the spatial arrangement of the mixture and its orientation in the electric field.

Even though a number of calibration equations are available in literature, most of the calibration equations are based on much lower water content than common in oil sands tailings. The equation proposed by Topp et al. (1980) for example are limited to a volumetric water content of less than 53% (solids content of greater than~ 68%) and most of the data are less than 40% volumetric water content (solids content of greater than ~79%). Many researcher's such Drnevich et al. (2005); Yu and Drnevich (2005); Siddiqui and Drnevich (1996); Siddiqui (2000); Ponizovsky (1999) evaluated different forms of the calibration equation but their study are also limited to a volumetric water content of 50% or less. TDR is used more in geotechnical earth works; and in agricultural and soil science professions, where partially saturated materials are common, thus previous studies reported on extensively in the literature were concentrated on soil of volumetric water content less than 50 %.

The accuracy of the TDR measurements depends on the calibration equation (apparent dielectric constant-water content relationship) and it is important to derive soil specific calibration or have a calibration equation that best describe the materials being studied (Yu and Drnevich, 2005; Gong et al., 2003). The objective of the calibration study in this research is to evaluate the suitability of calibration equation suggested in literature for oil sands tailings and to propose calibration equations that best describe the dielectric constant – solids/water content of oil sands tailings. The effects of soils texture, solute concentration, residual bitumen, temperature on the calibration equation are also addressed in the study.

The calibration procedure consists of preparing non segregated samples at different initial solids content and measuring their apparent dielectric constant by fully inserting the TDR probes in the samples. Materials at initial solids content of 15 to 78% were used in the investigation. Nine different types of materials, listed in Table 1, were used during the investigation. For each material, 6 to 14 samples were prepared at different solids contents. For each sample prepared at particular solids contents, six to nine TDR measurements were made to investigate the repeatability and reliability of the TDR readings. The averages of the TDR readings were used to correlate the apparent dielectric constant to the water/solids contents of the samples.

Three rod metal TDR probes (rods) were used for the tests. The free length of the TDR probes, the length of the probe that has direct contact with the surrounding soils, is 75 mm. The diameter of each rod is 2 mm and the spacing between the rods is 8 mm. The spacing between the rods and the radius of the rod fulfill the Knight (1992) recommendation that the ratio of the radius of the rod to the spacing between the inner rods should be greater than 0.1 and the radius of the rod as large as the average pore size of the material.

The samples for calibration tests were contained in PVC cylinders of 205 mm height and 96 mm diameter. According to Topp and Davis (1985), the sensitive region of TDR probe is a cylindrical surface surrounding the probes of diameter 1.4 times the spacing between the rods. Area just beyond the tip of the probe does not have an effect on TDR (Baker and Lascano, 1989). The size of the container used for the tests is greater than the size of the influence zone, diameter of container is 12 times the spacing of the rods and the height of container is about 3 times the length of the probe. TDR readings were taken by fully inserting the probes in the sample in a manner that minimizes air gap and the effect of soil containers boundary effects. The probe is extremely sensitive to the area immediately surrounding the probe and the presence of air along the rods significantly underestimates the dielectric constant of the material (Siddiqui, 2000; ASTM 6565-05).

The results of the dielectric constant- volumetric water content relationship of samples used in the study together with the calibration equation proposed by Topp's et al. (1980) are shown in Figure 3. The results in Figure 3 indicate that the dielectric constant-volumetric water content relationship of slurry/soft soil significantly deviate from the widely used Topp's et al. (1980) equation at volumetric water content greater than 45% (at solids content less than ~ 75%). The use of Topp et al. (1980) calibration equation for predicating the volumetric water content of tailings would results in as much as 28% lower values than the actual water content (Figure 4). The error in using the Topp et al. (1980) equation increases with an increase in volumetric water content of the samples but the error tends to decrease for volumetric water content above 83 %(for solids content of less than ~ 32%) (Figure 4). The decrease in error for volumetric water greater than 80% is because the Topp's equation was constrained to pass through the apparent dielectric of water (81.5, 100), though the volumetric water content of their soil samples were generally less than 53% (greater than~ 68% solids content).

Whalley (1993); and Herkelrath et al. (1991) suggested a linear form of calibration equation that relates the volumetric water content (θv) of the soil to the square root of the dielectric constant ($\sqrt{ka}$) (Equation 4). The constant a and b in Equation 4 are found from the regression analyses of laboratory test results but generally the constants a is around 12 and b is around 18. The linear form of calibration equation gives the same results as Topp's polynomial equation when constants a =11.81 and b = 18.41 (Yu et al. 1997).

The theoretical investigation of Ledieu et al. (1986) and Herkelrath et al. (1991) suggest that the calibration equation between volumetric water content and the travel time should be linear. The results in Figure 5 indicate that linear form of the calibration equation are applicable for the materials used in this study but with the constant a = 15.6, and the constant b = 34.1, are significantly differ from the values reported in literature. The results indicate that the constant a and b for high water content material differ from soil with low water content.

$$\theta_v = a\sqrt{K_a} - b \qquad [4]$$

A closer look at the dielectric constant-volumetric water content of the materials in Figure 3 indicates that the relationships depend on the texture of the material. At given volumetric water content, high clay content and/or high bitumen content materials tend to have lower dielectric constant than high sand fines ratio oil sands tailings or non-bitumen's materials (Devon silt and Kaolinite). AL_15 has high bitumen content while AL_7.5 is clay rich oil sands tailings both have lower Ka than the high sand content oil sands tailings (AL_70 and AL_50). The dielectric constant is a measure of the ease with which molecules can be polarized and oriented in electric field (O'Connor and Dowding, 1999). The presence of residual bitumen in the fines and the diffused double layer near the clay surface may limit the orientation and polarization of these molecules leading to a lower Ka for high clay content or high bitumen content tailings. The coating and bonding of bitumen on soil solids may affect the interaction of soil particles surface and water that leads to lower Ka compared to the value of bitumen dielectric constant. The dielectric constant of bitumen is 2.7 at 25 °C (Whiteoak and Read, 2003), close to the value for soils solids.

The dependency of the calibration equations on soil type is consistent with the findings of Bohl and Both (1994); Driksen and Dasberg (1993). Because the dielectric constants of soil are a function of particle specific surface, particle shape and the interaction between soil surface and water, TDR water content is soil type dependent (Driksen and Dasberg, 1993). Bohl and Both (1994) based on tests conducted on a number of samples concluded that the dielectric constant versus water content relationship varies over a wide range depending on the soil type. O'Connor and Dowding (1999) indicate that there is no universal calibration equation that can be applied for all materials to give an absolute measurement of water content.

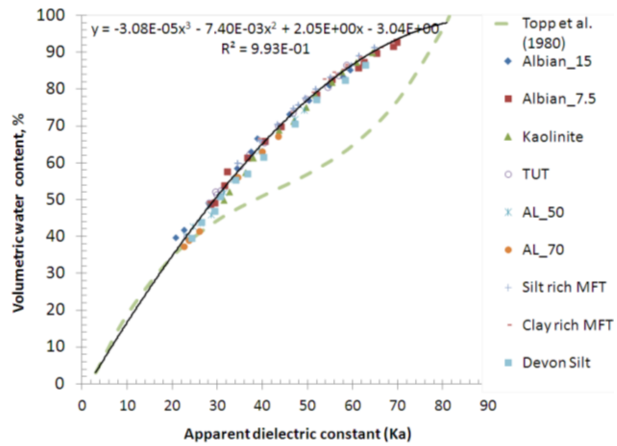

Figure 3. Apparent dielectric constant-volumetric water content relationship.

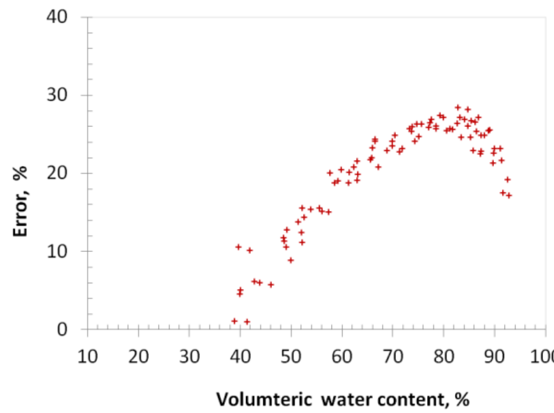

Figure 4. Measured Errors for the Topp et al. (1980) equation in calculating the volumetric water content of slurry and soft soil.

The apparent dielectric constant is better to correlate with the gravimetric water or solids content than the volumetric water content of soil/slurry for the geotechnical application in the oil sands industry. The apparent dielectric constant versus solids content relationship of the oil sands tailings are presented in Figure 6. The solids content (S) of the tailings is bounded by linear equations S = -1.47Ka + 114 ± 4, the lower boundary for high clay contents and/or high bitumen content tailings and the higher boundary for high sand fines ratio fines-sand mixture tailings as shown in Figure 6.

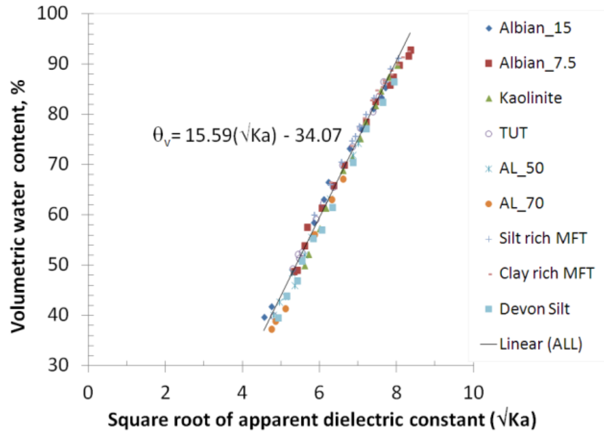

Figure 5. **Square root of dielectric constant versus volumetric water content relationship.**

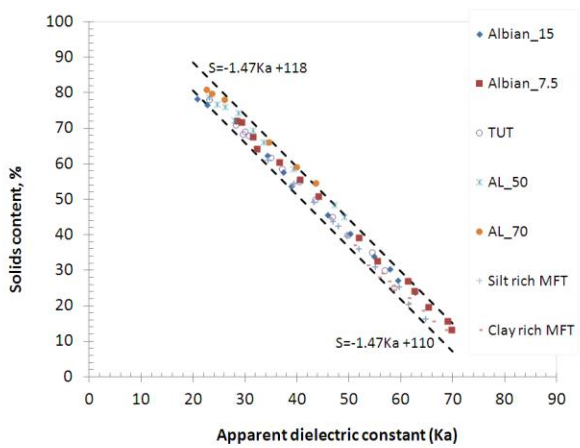

Figure 6. **Apparent dielectric constant-solids content relationship of oil sands tailings.**

The repeatability of TDR measurements at different solids content are evaluated using repeated TDR measurements made at each solid contents. The results are summarized in Figure 7. The standard deviation in solids content measurements from repeated TDR measurements is less than 0.6%, but the standard deviation in TDR solids content measurements tend to increase to 1% when the solids contents of the tailings are less than 25%.

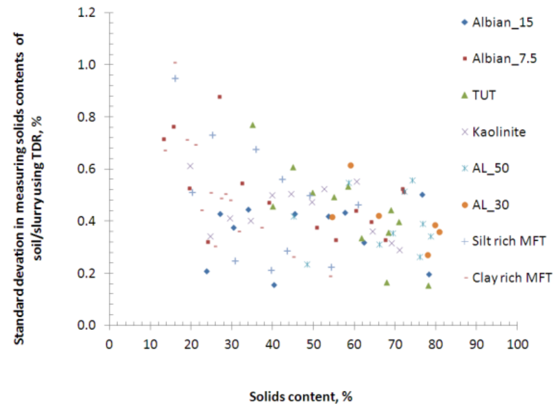

Figure 7. **Standard deviation in solids content measurement using TDR.**

CONCLUSION

Samples at various solids contents and fines content were prepared to study the texture influence on the TDR water content measurements. The results indicate that TDR can be used to measure the solids content of tailings over a wide range of solids content. Detectable measurable reflections were observed over all solids contents range and for all materials used in the study. The percentage of clay and bitumen in tailings affects the TDR water content measurements. The TDR has a good repeatability in measuring the solids content of the tailings. The use of TDR avoids the inherent radiation hazards in using the X-ray and neutron methods and the methods gives reliable direct results that minimize cost of tailings sampling for water content monitoring. Site specific calibrations are an important element of using the TDR but the study in this paper highlights the effect of texture on the calibration equations. The study shows the use of calibration that were derived for very high solids content and unsaturated materials would results a significant under estimation of water content when applied for very high water content materials.

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Session 5

Tailings Technology Roadmap 1

STRATEGY FOR DEVELOPING TAILINGS ROADMAP – A CASE FOR NON-COMPETITIVE COLLABORATION

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ABSTRACT

How did competing tailings management consultants combine forces to develop a tailings roadmap in response to an invitation by Alberta Innovates? Consulting engineering in tailings management is highly competitive often involving intellectual property and corporate advantage related to unique project experience. Not surprisingly, cooperation and collaboration by competing consulting firms did not occur without a recipe. The first ingredient was common acceptance that implementation of effective tailings technologies that result in trafficable tailings surfaces is the most important goal - even more important than winning. A group of five tailings management consultants and the UofA Geotechnical Group understood this priority and agreed to work together in responding to an invitation by Alberta Innovates to prepare the winning proposal. Other ingredients included a willingness to share information, transparency, acceptance of 'best-in-class' staffing regardless of remuneration, teamwork, a strong management team and commitment to a process that equally involves all participants.

INTRODUCTION

When Alberta Innovates issued a request for proposal (RFP) in November 2010 to develop a tailings roadmap, consultants quickly realized that this was a benchmark project that would require the most qualified tailings management specialists with broad, world-wide tailings management experience along with specific knowledge of oil sands tailings issues. Golder Associates Ltd. (Golder) immediately considered this to be a high profile project that would place the successful consultant in a spotlight; a project that might lead to an increase in future consulting assignments, or, conversely, lead to intense scrutiny, friendly disassociation and undesirable characterization of the process and technical outcomes.

Golder evaluated the RFP at the corporate level where it was characterized as a 'No-Go' (no bid) as a sole proponent due to the high risk of negative characterization by sectors of the oil sands mining industry, consulting community, NGO's and media, regardless of the quality of technical work completed. Within Golder, this negative perspective on the Tailings Roadmap project was based on precedent where previous oil sands industry-wide reviews of tailings management options had not generated favourable endorsement by stakeholders; however, Golder's 'No-Go' decision pertained only to submission as a sole proponent. Golder therefore began to investigate the interest of competing consultants to submit a combined proposal submission and learned that other specialist tailings consulting firms had similar views, and that while there was little interest in submitting proposals independently, other firms were enthusiastic about the submission of a combined proposal. Gauging interest was a simple matter because each of the collaborating consultants belonged to a relatively small and friendly community. A strong endorsement of a combined proposal submission was quickly confirmed as each consultant immediately agreed to the consortium approach proposed by Golder.

A total of five competing tailings consultants and the University of Alberta (UofA) Geotechnical Group formed the Consortium of Tailings Management Consultants (CTMC). The five tailings management consultants comprised of AMEC, BGC, Golder, NorWest Mining Services and Thurber Engineering. The UofA Geotechnical Group was represented by Dr. Dave Sego and Dr. Ward Wilson.

The strategy involving a large consortium for proposal submission was mirrored by the proponents of the Tailings Roadmap project that comprised a consortium of government agencies (Alberta Innovates, Alberta Environment, ERCB) and a consortium of oil sands firms (CNRL, Imperial, Shell, Suncor, Syncrude, Teck and Total) that had formed the Oil Sands Tailings Consortium (OSTC). Non-competitive collaboration was adopted by three types of stakeholders including the oil sands industry, government regulators and private consultants.

TIMES WHEN COMPETITION NEEDS TO BE AVOIDED

Competition amongst resource industry producers, and even suppliers of consulting services is often used to achieve best performance, minimize costs and provide for excellent client service. The engineering consulting industry is a highly competitive environment where consulting firms invest in staffing, knowledge and experience, without realizing rapid returns on investment. The downside of this competition is a degree of secrecy, partial disclosure and reluctance to share learnings and new discoveries with the larger community. The upside of competition in the consulting industry is commonly expected to include a potential for increased investment in specialized staff, a highly motivated team that is focused on superior technical performance, client service excellence and innovation.

For the tailings roadmap project, the collaborating consultants readily recognized that competition on this important project would compromise not only the project outcome, but possibly the endorsement of the outcome by stakeholders. They also recognized the risk to the oil sands industry's international reputation that might result from a failure to develop a widely endorsed path forward for improved tailings management leading to sustainable land reclamation. It was clear that the optimum path forward may not be achieved through the usual competitive process.

A CASE FOR NON-COMPETITIVE COLLABORATION

For the Tailings Roadmap project, the collaborating consultants readily agreed that full disclosure, sharing of learnings based on combined oil sands experience, and common endorsement of the outcomes was needed to advance best practices in oil sands tailings management. Accordingly, consultants forming the CTMC readily agreed to form the consortium led by Golder, who had initiated the collaboration.

It was their commitment to the environmental benefit to the oil sands industry that motivated members of the CTMC to set aside competition in favour of full disclosure, sharing learnings and mutual benefit. The goal is to achieve excellent oil sands tailings management practices that will result in rapid and progressive trafficability of oil sands tailings deposits and lead to sustainable reclamation. This is considered to be one of the most pressing environmental issues in the province of Alberta and therefore deserves a response without self-interest.

MANAGING COLLABORATION

Having decided upon a non-competitive association for accepting the assignment, the CTMC needed to follow through and establish a non-competitive collaborative environment for conducting the work. The strategy for avoiding competition and maximizing collaboration was developed during preparation of the proposal when association agreements were negotiated and execution plans were formulated. Key elements of the strategy are summarized as follows.

- **Association of equals in prime consultant and sub-consultant relationships.** The goal was to develop an association of equal partners. Conceptually, this would most readily be accomplished by a legal Joint Venture arrangement and the creation of a separate legal entity. The sole purpose of the separate legal entity would be to accomplish the work effectively and profitably, relying on the most qualified inputs irrespective of company affiliation. However, such an entity is not readily established because of the high cost, extensive legal inputs and significant collaboration by the corporate staff of each firm. Furthermore, the size of the Tailings Roadmap project did not ordinarily warrant such an investment and therefore a simpler arrangement was adopted. The partners of the CTMC agreed to a prime consultant – sub-consultant relationship in which the sub-consultant partners trusted the prime consultant to manage the project fairly while respecting the inputs by other partners equally to those of the prime consultant.

- **Every participating firm valued and given a role.** The associated consulting firms and university affiliation were assembled into the CTMC because of their significant contributions to tailings management in the oil sands industry as well as the international mining industry. The CTMC's response to the RFP for the Tailings Roadmap project contained combined inputs from all the associated firms, with each given a valued role that was presented with equal importance to others in the proposal.

- **Dedicated project manager.** The CTMC decided that the project would be best managed by a dedicated project manager that would look out for the interest of all members of the CTMC, not only the interest of the prime consultant. This proved to be the unifying force during project execution.

- **Clarity of association.** The dedicated project manager was focused on project delivery based on a Memo of Understanding (MOU) that united the members of the CTMC in the common goal of achieving excellence in the proposed Tailings Roadmap.

- **Team selection based on best in class specialists.** Key members of the technical team were selected early during preparation of the proposal in a meeting of CTMC member companies. Each tailings specialist was selected individually based on his/her expertise, not necessarily company affiliation.

- **Agreement on disagreements.** The strong potential for serious disagreements during project execution was recognized, discussed and addressed in the MOU that bound the members of the CTMC. Each member would have the right to express differences of opinion and document such differences in the formal report submissions. Although the right did exist and did lead to good relations amongst all, this right was not exercised by CTMC members at any point in the project.

- **Frequent internal meetings during preparation of the proposal.** Meetings of senior, experienced staff are costly because of their high charge rates and travel disbursements. However, CTMC meetings proved to be valuable tools for developing an effective project execution plan, maximizing efficiency of the work and providing focus and priority of selected staff.

- **Frequent meetings with stakeholders.** The work involved four large meetings of stakeholders and CTMC members in addition to many meetings of technical staff. Much of the work of project components 2, 3 and 4 was conducted at meetings of technical specialists representing member companies of the CTMC, government agencies and the oil sands mining companies.

- **Work and budget allocated based on expertise.** Work and associated budget was apportioned based on the expertise of the selected team members, not on the basis of company affiliation.

- **Internal technical reviews.** Each member of the CTMC was encouraged to review reports by others with a view towards a common endorsement of the final roadmap deliverable.

- **Equal recognition.** Each member company/group of the CTMC was recognized equally in official reports and deliverables. Primary authors of technical reports were also given recognition for their specific contribution.

SUPERIOR OUTCOMES

The CTMC resulted in a widely-endorsed tailings roadmap based on the best technical expertise available. The project team, Client, and project itself benefited significantly from the non-competitive environment that was created for execution of the Tailings Roadmap project, which subsequently facilitated the input from some of the best and most experienced specialists in the tailings management business; a team approach with a common goal of reclaimed tailings areas; and extensive internal and external collaboration that was developed by the management team. Each of the participating firms within the CTMC acknowledges the value of non-competitive collaboration that was adopted for this Tailings Roadmap project.

THE OIL SANDS TECHNOLOGY ROADMAP AND ACTION PLAN: LEVERAGING THE POWER OF COLLABORATION FOR STRATEGIC SUCCESS

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Wikipedia defines collaboration as "*a process where two or more people or organizations work together to realize shared goals, (this is more than the intersection of common goals seen in co-operative ventures, but a deep, collective, determination to reach an identical objective …. an intriguing endeavor that is creative in nature by sharing knowledge, learning and building consensus*".

Certainly there are great visionaries, but all great visions are achieved by the collaborations of focused people and organizations that brought that vision into reality - "*None of us is as smart as all of us*" – Japanese proverb.

INTRODUCTION

The development of the oil sands of northeastern Alberta has been a long hard road that began with its first commercial operation in 1967. Oil sands mines produce very large quantities of solid and fluid tailings. Mining, and the corresponding tailings production, has been ongoing for over 40 years and is expected to continue, at increasing rates, in the coming decades.

Although a water-based extraction process has been developed that achieves 90% or better recovery of the bitumen from the oil sands, the relatively high capital and operating costs of a commercial venture made it a risky investment given the see-saw oil prices of the last 30 years leading up to the end of the 20th century. As a result, technology development by the industry tended to be focused on advancements in getting the bitumen out as efficiently as possible (truck and shovel mining, mineral processing with slurry transport, more energy efficient extraction processes).

Due to the long-standing regulatory requirement to contain all process-affected water on the lease (Figure 1), research on tailings in the past has tended to focus on proper engineering to contain the tailings and secondarily the characterization and fundamental properties of the fluid clay slurry we now call mature fine tailings (MFT). The needs of the extraction process, driven by the economics of an ever-changing "business case" have tended to over-shadow the growing volume of MFT.

Figure 1. Example of a Commercial Tailings Operation (after CTMC, 2012).

With the arrival of several new players on the scene starting in 1995, coupled with a very positive trend in the price of oil, the "business case" for bitumen extraction has matured and become more profitable in a sustained sense. Along with international recognition of the resource potential came international recognition of the industry's various environmental challenges. There are ongoing concerns about the geotechnical risks, environmental risks, and long-term liability related to tailings production. In particular, there are concerns related to production, storage, and reclamation of fluid fine tailings (FFT). Because of the maturation of the business, the provincial government positioned to become more demanding of the industry and the industry was now in a position to be more responsive to the "environmental costs" of its "social license" to operate.

HISTORY

There is a history of research cooperation between the Alberta government and the oil sands industry; the foundation of which was the Alberta Oil Sands Technology Research Authority (AOSTRA) from 1974 to 1995. Since the time of AOSTRA, there have been some key collaborations (Figure 2).

Tailings management in the oil sands has evolved over the years, with three overlapping periods of tailings technology development:

- **1960s through 1980s:** Focus on waste management (materials handling and geotechnical stability). Numerous large dams were constructed from tailings to hold fluid fine tailings, space was left in mined-out pits for long-term storage of fluid tailings, the focus was on constructability, safety and cost.
- **1990s through 2000s:** The focus was on creating solid tailings landscapes capable of supporting terrestrial reclamation and land uses. Research, development, and implementation of sand-capped composite tailings, ongoing research and development of water-capped MFT end pit lakes.
- **2010s:** The focus is now on meeting new regulatory goals to reduce production and legacy volumes of fluid fine tailings and speeding up the rate of reclamation of tailings ponds. Research, development, and implementation of high-fines soft tailings treatment technologies.

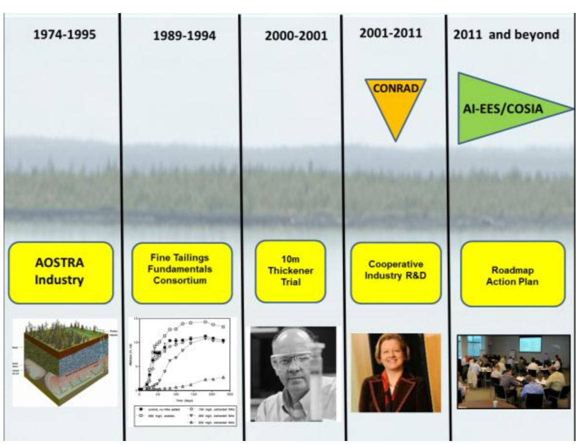

Figure 2. A timeline of Government/industry collaboration on Alberta Oil Sands.

The Oil Sands Tailings Technology Roadmap and Action Plan project was founded in the principals of collaboration and open cooperation – it was the intersection of four key events. In February 2009, the Energy Resources Conservation Board (ERCB) implemented Directive 074 (the tailings management directive). About the same time, Alberta Environment began working on a new tailings policy document now known as the "Tailings Management Framework". Concurrent with these two regulatory initiatives, Alberta Innovates-Energy and Environment Solutions began to engage both the industry and the regulators with respect to the need for more collaboration on tailings technology development. In December of 2010, the industry announced the formation of the Oil Sands Tailings Consortium (now a part of Canada's Oil Sand Innovation Alliance, COSIA).

The execution of the Tailings Technology Deployment Roadmap project has provided a renewed opportunity for the oil sands industry (operators, regulators and consultants) to work together towards a common goal. The project led to an encouraging convergence and understanding, within the industry, of the technological challenges at hand. There is a better understanding of what it might take to source and deploy new technology to successfully manage and reclaim oil sands tailings. The participants in the project behaved like a new organization that matured toward a shared vision.

The "Roadmap" (as it has come to be called) was an initiative of both the government and industry to support a broader strategy for sustainable management of tailings produced by the oil sands industry. The four primary components of the project are presented in Figure 3.

During the course of the project it became clear that the approach and technology that how each oil sands company uses to deal with their tailings will differ according to the specific characteristics of the extraction technology, the oil sands ore and the lease conditions. However, it is clear that when it comes to "social license to operate" issues, such as the environment, a collaborative approach is the best path forward. It was also clear that for these social license to operate issues, success can only be achieved through transparent execution by a team whose mandate is not that of individual corporations but is the shared vision of the industry (OSTC) and the province (AI-EES).

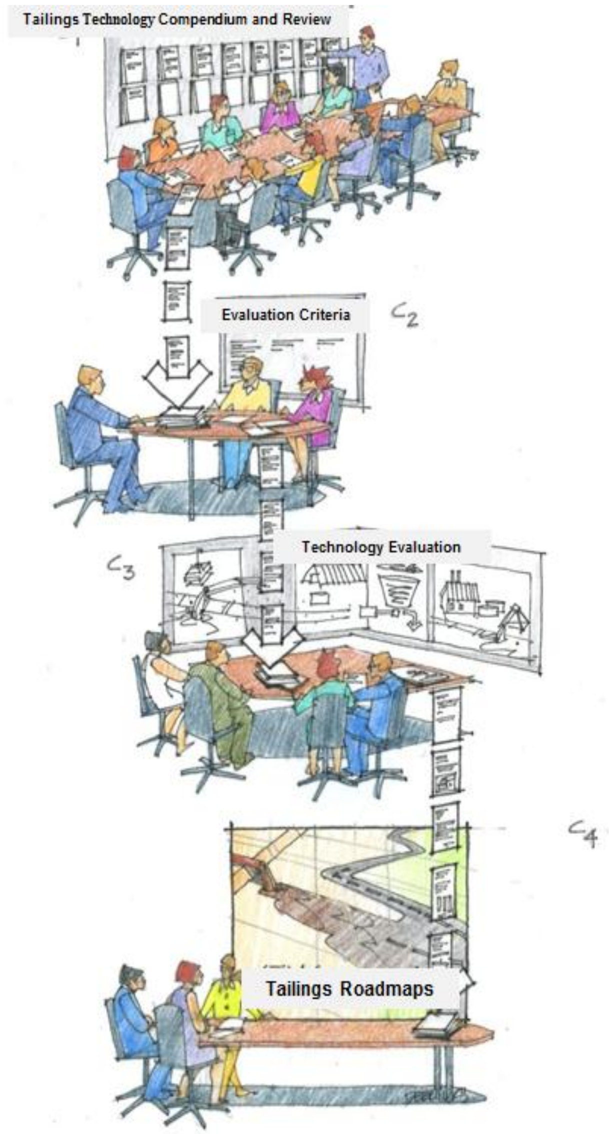

Figure 3. Four components (C1 to C4) of the tailings roadmap and Action plan study (after CTMC, 2012).

Tailings technology development in the oil sands to date has involved individual companies and consortiums of companies, often working through the Canadian Oil Sands Network for Research and Development (CONRAD), or with universities, consultants, provincial and federal government laboratories and agencies. During the evaluation process a substantial number of tailings management experts provided knowledge which was brought to bear in addressing the challenge (Figure 4). The collaboration of the Roadmap Action Plan has revitalized the vision of AOSTRA where the people of Alberta are partners with industry in the development of technologies that can achieve environmentally sustainable exploitation of our energy resources.

STUDY RESULTS

Management of oil sand tailings in Alberta is a high priority for industry and government. The Tailings Technology Roadmap and Action Plan project is a solid step towards maximizing the know-how Alberta industry already has and building on that expertise in future activity.

The Roadmap includes vital research that will inform policy and regulation of tailings in Alberta. No single technology can address all of the tailings challenges; the Roadmap is designed to evaluate and recommend the most effective technologies available at each stage of the tailings process. Over 500 tailings treatment technologies were identified during the initial review for the Roadmap project. From the initial review, 101 technologies were recommended for further study and incorporated into nine detailed tailings technology development roadmaps for the treatment of oil sands fine tailings.

Each roadmap contains a comprehensive "suite" of technologies that would allow an oil sands operator to convert their fluid fine tailings inventory into a "reclamation ready" deposit, depending on their individual characteristics of the mine operation. The roadmaps range from passive technology options such as end-pit lakes, to more engineered options such as thin lift drying and centrifuging. The technology options used by a company will ultimately be dependent on their specific mine characteristics.

The Tailings Roadmap / Action Plan initiative is providing a framework to government and industry that will:

- Help achieve more timely deployment of the end-to-end tailings technologies, and share the results and knowledge of tailings deployment activities.
- Document the current state of tailings reclamation technology to define technology pathways to reach the end goal.
- Serve as a basis for accessing government and industry funding to accelerate commercial scale

demonstration of technology and promote sharing and technology transfer.

- Identify technology options and establish a framework for operators to conduct detailed feasibility studies and deploy technology, and allow regulators to verify the performance during this process.
- Promote a collaborative approach to oil sands tailings technology that expedites technology deployment, reduces environmental impacts beyond the boundaries of the mine lease and enhances public trust.
- Provide a medium for sharing the results and knowledge of effective tailings deployment initiatives.

Some of the technologies that AI-EES and COSIA expect to advance through further R&D collaboration include:

- Hot water recovery focused technologies, such as:
  - In-line technologies
  - Centrifugation
  - Thick and thin lift dewatering
  - High rate thickeners
  - In-pit tailings processing and reclamation

- Progressive reclamation, including:
  - Solid deposits (e.g. CT, thin lift and centrifuge)
  - Wet deposits (end-pit lake, wetlands)

- Alternative technologies to eliminate or minimize tailings production,
  - Non-water based extraction processes such as retort and solvent technologies for high fines ores and rejects:

- Remove hydrocarbons from tailings to improve the efficiency and economics of treatment:
  - De-oiling of various tailings streams
  - Treatability of de-oiled tailings
  - Improvement of economics

THE PATH FORWARD

Future collaboration between the COSIA and AI-EES with respect to oil sands tailings will focus on the priorities and opportunities identified through the Oil Sands Tailings Technology Deployment Roadmap and Action Plan study (such as those listed above).

The OSTC is currently in the process of reviewing the Tailings Roadmap/Action Plan in terms of the recommendations to further evaluate a number of technologies. Of the 48 "Highlighted Technologies", 16 technologies are included in current, or planned, OSTC/COSIA projects; 22 technologies will be considered for inclusion in OSTC/COSIA projects; 8 technologies are viewed as out-of-scope relative to the OSTC/COSIA and 2 bitumen recovery-related technologies will be evaluated through a joint industry/AI-EES initiative.

Industry is committed to share the information necessary to progress the development of oil sand tailings technologies that are viewed as having the potential to help achieve the goals and objectives of industry relative to speeding up the reclamation of tailings ponds. Industry will work with AI-EES, government and Third Party Technology Developers to develop and implement tailings technologies that will ensure that oil sand tailings are successfully reclaimed on a timely basis.

Technology vendors will have greater access to make contributions through the ongoing technology development and assessment protocols that will be developed. These protocols should provide the basis for partnerships among oil sands industry operators, third party tailings technology providers and government R&D investment. A more holistic and wide ranging approach can be adopted when completing tailings technology evaluations, which involves a wider range of experts to provide technical input and advice on all components related to the treatment and management of oil sand tailings, including water treatment and reclamation.

ACKNOWLEDGEMENTS

Much of the content of this paper draws heavily upon the 2012 CTMC reports to AI-EES: "Oil Sands Tailings Technology Roadmap and Action Plan".

Figure 4: Oil Sands Tailings Roadmap and Action Plan Workshop

Session 6

Tailings Technology Roadmap 2 / Tailings Management 2

THE OIL SANDS TAILINGS ROADMAP AND ACTION PLAN: OIL SANDS TAILINGS STATE OF PRACTICE OVERVIEW

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ABSTRACT

Oil sands mines near Fort McMurray, Alberta, Canada produce very large quantities of solid and fluid tailings. Mining, and the corresponding tailings production, has been ongoing for over 40 years and is expected to continue, at increasing rates, in the coming decades. This paper documents the commercial state of practice for tailings management, in the oil sands industry as compiled during the Tailings Roadmap project (CTMC, 2012a) sponsored by Alberta Innovates and the oil sands industry.

INTRODUCTION

All current and planned oil sands mining operations have adopted water based extraction for bitumen production. Older facilities (Suncor and Syncrude Base Mine) use a form of the original Clark Hot Water Extraction (CHWE) process and newer facilities have adopted the Low Energy Extraction (LEE) process initially developed by Syncrude for use at the Aurora North project. This paper reviews the various tailings strategies that are currently in use to process water based extraction tailings, the slurry transport methods used to deposit these materials, and the practices for constructing a trafficable surface amenable for placing reclamation material.

Tailings management in the oil sands has evolved since mining was initiated in the region and three overlapping periods of tailings technology development can be considered. Tailings technologies from all three periods of development are still in use today, mostly in combination with one another:

- <u>1960s through 1980s</u>: The focus was on waste management (materials handling and geotechnical stability). Numerous large dams were constructed from tailings and to hold fluid fine tailings, space was left in mined out pits for long-term storage of fluid tailings, and the focus was on constructability, safety, and cost. Tailings technologies that have evolved from the early years of oil sands mining are referred to here as “conventional tailings.”
- <u>1990s through 2000s</u>: The focus was on creating solid tailings landscapes capable of supporting terrestrial reclamation and land uses. This included research, development, and implementation of sand capped composite/consolidated tailings, and ongoing research and development of water capped MFT end pit lakes. Tailings technologies that were developed during this period are termed “process tailings” for the purpose of this paper.
- <u>2010s</u>: The focus is on meeting new regulatory goals to reduce production and legacy volumes of fluid tailings, and to speed reclamation. This has resulted in rapid research, development, and implementation of high fines, soft tailings technologies. All these technologies currently involve managing MFT inventories and are referred to here as “emerging technologies.”

The individual technologies developed during these periods form combined “suites” of technologies that meet various performance objectives within the overall tailings plan for a project.

CONVENTIONAL TAILINGS TECHNOLOGY

All current and planned operations have or plan to use conventional tailings as part of their overall tailings management system, especially during early years when there is no space for depositing tailings in the mined out pit areas. This is the primary tailings method for managing out-of-pit tailings (whole tailings discharge) and most projects have some component of water capped Mature Fine Tailings (MFT); these are the two suites referred to here as conventional tailings.

Conventional Hydraulic Fill Construction

The tailings resulting from the Clark Hot Water Extraction process and its modern derivatives produce a segregating tailings slurry. The slurry is conveyed by pipeline and discharged into a disposal area, on the inside of a sand or overburden dyke, forming “beaches.” Upon discharge, the coarse sand settles rapidly from the slurry, leaving a water runoff that carries with it a portion of the fines (clay and silt), which forms a “pond.” In general practice, sand cell construction is used to construct the dykes; sand and fines are captured in the cells and beaches; and the pond contains process water with fines that run off as the beaches are formed. The fines settle, forming a “clear water zone” near the surface of the pond, which is recycled for use in the extraction process. An alternative to tailings sand-cell construction is to raise all or part of the pond containment dykes using earthen construction sourced from overburden or oil sands reject interburden (the lean oil sands seams within the orebody).

The basic tailings disposal method is to discharge whole tailings and/or cyclone underflow tailings as a single stream and either beach the sand to the interior of the tailings pond or use it for cell construction of the pond containment dykes. Typically, dyke cell construction uses a method called “modified centre line” or “upstream construction” where the cells are constructed over the previous beach area. As an integral component of the dam, that portion of the beach must have geotechnical integrity. Wide track dozers are used to compact the sand cells that make up the containment structure.

Conventional hydraulic fill construction is a proven and secure method to manage oil sands tailings. The major issues relate to the fluid fine tailings that segregate during deposition to form first Thin Fine Tailings (TFT), and then with time, MFT. Also, bitumen and solvent losses from froth treatment are significant contributors to floating oil on the recycle ponds and to operational difficulties with bitumen “mats” in the fluid fine tailings. Solvent on the pond surface contributes to atmospheric VOCs with HSE implications and acid generation has been reported due to pyritic sulphur. The potential for acid-generation from dewatered, near-surface Forth Treatment Tailings (FTT) deposits under reclamation needs further investigation. Increased concentration of low level NORMs (naturally occurring radioactive materials) has also been reported in FTT deposits.

Conventional tailings continue to offer benefits necessary for efficient oil sands operations:

- Low cost slurry transport.
- Mature technology with well-developed engineered containment design and construction methods.
- Segregation allows efficient cell construction.
- Handles a wide variety of conditions (low or high fines, low or high bitumen, low or high solids, can include other low volume waste streams).
- The pond is an efficient “vessel” for clarification of process water and initial sedimentation.
- New findings, from Syncrude’s Aurora Settling Basin and Suncor’s Pond 1, show that 70 to 75% of the fines reporting to the basin are trapped within the tailings sand, much as Beach Below Water (BBW) or perhaps more specifically as Beach Below MFT. This is a larger fines capture than is typically assumed during initial design stages.

There are numerous opportunities for improvement:

- Reduce large quantities of fluid fine tailings.
- Improve consolidation rates and strength gain rates for fine tailings.
- Improve sand permeability for water table control.
- Reduce propensity for erosion on downstream dyke slopes.
- Reduce head losses and wear on pipes and pumps.
- Improve fines capture in settling basins. These sandy, fines rich deposits end up sequestering most of the fines from the orebody.

In particular, there is opportunity through more engineering and operational efforts to further increase the fines capture rate, thus reducing the MFT production rate. This opportunity for “managed beaching” should be investigated through fundamental work and large trials.

Water Capped MFT

Water capping of MFT within a lake created by a completed mine pit has been under development by Syncrude since the early 1980's (see Figure 1). After 30 years of small-scale demonstration, culminating with a four-hectare pond under observation since 1984, this most studied of methods is now being tested in a commercial scale demonstration at Syncrude's Base Mine Lake Project. Transfer of some 200 million cubic meters of MFT into the 800 ha pit, west of Highway 63 South at the Mildred Lake site is underway, with water capping to be completed in 2012.

Figure 1. Water Capping Test Ponds at Syncrude.

While this project is being conducted within an operating mine, Syncrude and other operators await the assessment of this commercial scale demonstration. In-pit, water capped sequestration of MFT is a proposed component of closure plans for all mines now in operation. Predominant technical matters to be verified include:

- Lake performance.
- The rate of bio-degradation of naphthenic acids in the process water cap.
- The potential for re-disturbance of the settled tailings.
- The potential for permanent or semi-permanent stratification.
- The rate of release water from the underlying MFT at large scale.
- The potential for mixing of lake bottom sediments with surface water due to wind effects.

This technology is a sequestering technology and accepts that the MFT will consolidate over a very long time period in the range of tens to hundreds of years.

An opportunity that should be evaluated with water capping is to increase the density of the MFT placed at the lake bottom either by flocculation and placement or flocculation, centrifugation and placement. This would allow for a significant reduction in MFT sequestered providing for low-cost deposition. In addition, the greater density difference between the water layer and mud layer reduces the risk of mud turnover. Further, the higher density means there is less water, containing naphthenic acids, remaining in the deposit to migrate into the lake water layer.

PROCESS TAILINGS TECHNOLOGIES

During the 1990's tailings management practices focused on conversion of fluid fine tailings into semi-solid deposits that were amenable to capping for terrestrial reclamation. Co-disposal of coarse and fine tailings (CT) was researched, developed, and implemented at Syncrude and Suncor, and is now a major commercial technology. Thickened tailings (TT) was piloted by CONRAD at the Syncrude Aurora site, and is in use at Shell's Muskeg River and Jackpine Mines.

Both CT and TT have the potential to reach their goals of reducing fluid fine tailings inventories, creating more solid landscapes, and providing more terrestrial reclamation. A dominant challenge has been in the day-to-day, all season operation of the processing equipment to produce robust, non-segregating, fast consolidating slurries. Also there is a need for better management of off-spec materials, and improved methods to monitor field deposition processes at large scale and in difficult operating conditions.

Composite Tailings (CT) and Non-segregating Tailings (NST)

Composite / consolidated tailings using MFT as a feedstock was piloted at both Syncrude and Suncor with excellent results. Syncrude's ten year old 10 Mm^3 CT Prototype deposit has been held up as the ideal for CT that is a dedicated disposal area, single discharge point, high spec CT, with a successful sand cap and reclamation. Some elements leading to its success are difficult to duplicate commercially. These include partial separation from production, ability to discharge off-spec materials to other deposit, single discharge point, dedicated operational staff, full time technical oversight, almost entirely subaerial deposition, pre-installed instrumentation, close mass balance, shallow deposition and summer-only operation.

A limitation of CT is that the sand used in this process is unavailable for dyke construction, beaching or capping activities, and thus priorities must be set for sand use. Also, the CT must be fully contained while the deposition is underway. For these reasons the method is not seen as suitable for sites where out-of-pit or in-pit containment is unavailable or subject to high costs.

Full scale composite / consolidated tailings operation at Suncor proved disappointing, resulting in large volumes of high fines, low solids tailings that will be difficult to reclaim. The major issue was CT segregation when discharged below water and below MFT, despite the trialing of numerous discharge techniques. At Pond 5, a mechanically placed coke cap is being installed, along with tightly spaced wick drains, to speed consolidation of this segregated, soft “CT” deposit. Capping plans are in place for Ponds 6 and 7 at Suncor.

At Syncrude’s East In-Pit pond (EIP), the composite tailings was more robust and the deposit is nearing full height and being sand capped. Off-spec material was removed by an MFT / RCW reclaim barge and transported to the West In-Pit pond (WIP). Unfortunately, for a variety of reasons, much of the deposit volume is composed of tailings sand. The remaining volume of high fines CT in the distal end of the deposit is being evaluated.

Syncrude has recently started to place CT in the Southwest In-Pit area (SWIP) and is experimenting with commercial scale sub-aqueous tremmie discharge. A large CT plant is being designed and constructed for Syncrude Aurora North East Pit. CNRL is constructing facilities for NST (combined TT and tailings sand using a thickener that results in a material similar to CT) at their Horizon Project. Is it thought to have similar benefits and challenges as CT from MFT.

Some of the benefits of CT and NST technology at commercial scale include:

- Deposits strong enough and dense enough (if on-spec) to allow hydraulic sand capping.
- Rapid release of process affected water during deposition, available for recycle.
- Consolidation settlements complete in 0 to 20 years, typically with 0 to 4 m of post reclamation settlement, manageable with a well-designed sand cap.
- Some reduced toxicity of release water. Release water is clear.
- Ability to control coarse and fine inputs and coagulant dosages in real time.
- Ability to create deposits that provide about 30 to 50% upland and 50 to 70% wetland land uses.

And there are areas for potential improvements:

- There can be difficulty in creating non-segregating deposits, largely due to un-optimized discharge methods. Off-spec CT is difficult to remobilize (without dilution), or to cap, essentially behaving as sandy MFT.
- Surfaces are untrafficable to mining equipment even when consolidated, and must be capped with sand hydraulically or mechanically (or capped with water).
- Large bulking factors requiring a large containment volume per tonne of fines.
- Deposits are loose and hence essentially permanently liquefiable, requiring robust or geologic containment. The potential for liquefaction of the upper part of the deposit must be considered when designing and constructing closure landscapes.
- Calcium and sulphate in the recycle water has detrimental impacts for re-use in the plant, and impacts on aquatic organisms.
- Large quantities of tailings sand are consumed, leaving little for dyke construction and sand capping.
- Overall fines capture rates need to be improved; the loss of fines when the CT is off-spec and when one includes the losses associated with sand capping and dyke construction is higher than originally envisioned during CT development.

There is opportunity to examine the engineering properties of lower SFR CT (nominally 3:1) that would reduce the reliance on sand for CT production, increase the fines capture, and potentially provide better non-segregating behaviour. It can also be concluded that producing higher density CT discharge would increase the fines capture and make a more robust operating system.

Thickened Tailings

In a typical oil sands thickened tailings circuit the thickener is derived from a cyclone overflow stream. The underflow contains most of the coarse sand and is about 50% solids. Most of the water, carrying most of the fines, reports to the overflow. The fines stream feed to the thickener enters at 10 to 20% weight solids, at a sand to fine ratio of approximately 0.3:1 to 0.8:1 SFR. The thickener uses flocculants to enhance gravity settling of solids in water, with increasing density towards the bottom of the thickener. Clarified water from the overflow of the thickener is recycled back to the extraction plant for re-use in ore processing. The thickener underflow is densified to a solids target 45 to 55% and pumped for storage behind engineered containment. Density is limited by rake torque and pumping requirements.

The Albian Sands project operates commercial scale thickeners at both their mining projects:

- The two high-rate thickeners installed with the original Muskeg River Mine were predominantly justified for the recovery of warm water for re-use in the extraction process. Thickener underflow has been discharged into the tailings settling pond in several different configurations. While significant segregation has occurred, some of these deposits are exhibiting consolidation behaviour.
- The Jackpine Mine, which commenced operation in late 2010, included a high-rate thickener with design improvements based on experience with the Muskeg River operation. Low fines content in the opening area of the Jackpine mine has resulted in TT production below the design range of the underflow system. Increased water must be withdrawn with the TT underflow to satisfy minimum flow rates. This has prevented any meaningful assessment of the ability to reliably produce on-specification TT. Design modifications are underway to provide for a greater range of fines in the oil sand feed.

Consistent operation of thickeners to produce a non-segregating TT product has not been accomplished at commercial scale. The primary driver for design of the entire thickening and deposition process is to achieve a deposit with acceptable fines capture, consolidation rates, density, and strength. The design includes the disposal area required to contain the tonnages, lift thickness and lift frequency and achieve the deposit performance that will support timely capping and consolidation consistent with reclamation plan commitments. Design of a robust thickener must accommodate the range and variations of feed expected from mining/extraction. Consistent target solids and SFR to support a beachable TT could be a challenge with a high rate thickener.

An alternative that requires re-flocculation of the thickener underflow to achieve a permeable deposit that would release additional water has been considered but not fully explored or tested on a large scale. If a two-stage flocculation protocol could be successfully developed, this would make the method similar to the in-line thickening deposition methods discussed below as emerging technologies.

Some of the benefits of thickened tailings include:

- Rapid release of warm, process affected water suitable for immediate re-use in the extraction plant.
- Consistent underflow stream with reasonably good consolidation properties.
- Hydraulic transport to retention ponds.
- Formation of soft, but non-liquefiable deposits with properties of that of normally consolidated soils.
- No ionic loading of the recycle water stream.

Areas of improvement for TT include:

- Pre-conditioning of fines slurries for greater consistency before loading.
- Improved flocculents / flocculent additions for thickening and post deposition consolidation properties.

There is opportunity for continuous improvement of many aspects of producing thickened tailings that will reduce costs and increase reliability.

A systematic review of industry wide oil sands thickener operating and deposit performance should be considered, with a view towards a) improved reliability, and b) optimizing the SFR and density of the discharge, to maximize consolidation behaviour and form a cappable deposit. This will probably require a departure from high rate thickening towards thickener technologies that can readily produce and handle a denser underflow product, which may also have implications for pumping technologies.

EMERGING TAILINGS TECHNOLOGIES

As the operational challenges and planning limitations of CT became more evident (e.g., containment volume requirements and competition for the use of sand), methods intended to solidify fines on their own, rather than blending fines with sand, received increasing attention.

Frustration at seeing legacy MFT volumes continue to grow, and the submissions of EPEA applications for new sites that would quadruple the amount of fluid tailings at the end of oil sands surface mining in forty years (despite adoption of processed tailings technologies) caused the ERCB to introduce new regulations to stabilize if not reverse the trend. ECRB Directive 074 (D074) was announced in February 2009 and has partially contributed to the drive towards a third wave of tailings technologies.

Since the release of D074, research and development activities have more than tripled, and several new technologies are at various stages of commercial development. Some of the individual technologies that form the “commercial suites” are still at development scale. This trend of rapid technology deployment with short implementation time frames, based largely on regulatory obligations, poses new challenges on the oil sands industry.

In-Line Thickening with Thin Lift Dewatering

This method involves in-line treatment of MFT with an anionic polyacrylamide flocculent followed by thin layer deposition. This technique has benefited greatly from recent polymer advances, accompanied by new mixing / shearing / pipelining techniques. Dewatering is accomplished by a combination of sedimentation, shear during beaching, and under drainage, with additional environmental effects to increase solids content after deposition.

Following trials on the Suncor site initiated by polymer manufacturer SNF Floerger, Suncor further refined the method and adopted it as the primary component of its Tailings Reduction Operations (TRO). The current practice is to deposit a thin layer (i.e. 10 to 30 cm). After initial dewatering to about 60% solids mass, the surface is reworked to break the crust and allow for atmospheric drying to increase the solids content towards design values. After one or two layers have reached sufficient solids content, the material is removed from the drying surface and relocated to waste dumps or kept in place.

Shell is deploying a similar method at its Muskeg River Mine, which relies on re-handling the material. The current plan is to place some of the material in overburden dumps.

As originally contemplated, this method could be practiced by placing, and leaving in place, multiple layers of thin lift deposition over each summer season. Each layer would “dry” in-place to from a “solid” deposit. Subsequent lifts would be placed in succeeding years. At an annual deposition rate of 3 t/m^2 to 5 t/m^2, a placement area of 1 km^2 could support an operation of 3 to 5 million tonnes per year of MFT solids.

Figure 2. Suncor TRO Spigots.

In practice, deposition rates require more area to sufficiently dewater the material. Less than 1 t/m^2 can consistently be treated in this manner. Even this rate produces material that may be too weak for building deep, multiple-layered, free-standing deposits to depths of 10 to 20 m. Challenges with this method include:

- Re-handling adds to the cost and requires sufficient volumes of construction-grade overburden for containment.
- The area required to operate the drying process plus the increased area required for secure overburden polders is very large.
- The availability of competent overburden material to manage both weak overburden units and the low-strength dewatered MFT.

There are significant opportunities to improve the In-line thickening / thin lift dewatering technologies including process, deposition, management, and cost. As a first step to realizing these opportunities, the industry should conduct a thorough review of the performance of the field studies conducted at Suncor, MRM, and Syncrude. The review should include a common basis – consistent definitions, a field database of performance, and a common understanding on the underlying physics of atmosphere evaporation, as a building block towards improving this technology.

In–line Thickening with Accelerated Dewatering

This method, being developed by Syncrude, uses in-line flocculation of MFT followed by deposition into a containment area, either in deep in-pit deposits (> 20 to 30 m) or shallow polders (i.e. ≤ 5 m thick in sand cells). The method is an alternative for centrifugation and has similar containment efficiency (lowest volume per tonne of fines). It is expected to have the lowest cost and lowest energy intensity of any of the pond solids methods.

This method was piloted by Syncrude in 2009 in a 10 m deep, 60,000 m^3 deposit. Syncrude plans a larger trial for 2012 using improved polymer and polymer mixing technology developed since 2009 that could increase both initial deposit dewatering and subsequent consolidation.

The method relies on having the ability to decant surface water expressed from the deposit along with precipitation. Self-weight consolidation is augmented with perimeter rim ditching and natural surface cracking to lower the water table. Starting at about one metre deep, the ditch is advanced many metres until the deposit is eventually unsaturated and over consolidated to partial or full depth. Perimeter ditching is widely used in Florida in the phosphate industry.

Figure 3. Syncrude 2009 Accelerated Dewatering Pilot.

Ongoing monitoring of the Syncrude pilot should continue with a view to developing an understanding of the role of cracking and rim ditching on the densification behaviour.

Centrifuge MFT

In this method, the MFT, supplied from a pond dredging operation, is diluted to a controlled concentration (currently 20% solids), mixed with a polyacrylamide solution and centrifuged to form a “cake” at 55 to 60% solids. The cake may be transported to a large engineered disposal facility by three different methods:

- Conveyor to stacker.
- Pumped with positive displacement pumps and transported by pipeline.
- Hauled by truck.

Syncrude testing has demonstrated a capacity of up to 80 tph using a 1 m diameter, solid bowl, scroll decanter. A larger 1.4 m diameter prototype machine is undergoing testing.

The attractiveness of centrifugation is the reliability of cake water content under varying weather, terrain and feed conditions, coupled with the ability to operate for most of the year. Capital cost and large-scale depositional behaviour are the primary concerns.

This technology has a high potential for large-scale conveyor transport, both out of pit if there is room and in-pit. There is opportunity to reduce costs with development of larger diameter centrifuges and the use of a mobile or re-locatable centrifuge plant.

This technology has potential for broad application to MFT beneficiation. Testing to date at the Syncrude site indicates that a reliable cake solid is achievable at high production rate with the commercially available centrifuges. Studies are underway to evaluate conveyor stacker operations. Feasibility level evaluations would improve an understanding of the economics for different transport methods and provide an opportunity to optimize the material handling schemes based on results from the pilots to date.

MOVING FORWARD

There is a major opportunity to increase the performance and decrease the cost of existing commercial tailings technologies and suites employed at the oil sands operations. After the first draft of this report was released, the Component 4 Team addressed this opportunity by carrying out a gap analysis of the commercial suites to evaluate the potential for continuous improvement of existing technologies and addition of new complementary technologies.

There is also a major opportunity for improved management and governance of existing tailings technologies, applying the same management controls used to produce tight specs on bitumen and synthetic crude oil, to the planning and operation of tailings systems using a more conservative and managed approach to achieving desired tailings performance.

REFERENCES

All material in this paper is drawn from the following report:

Consortium of Tailings Management Consultants (CTMC), 2012. Oil sands tailings technology deployment roadmaps: Project report. Report prepared for Alberta Innovates Alberta Innovates – Energy and Environment Solutions (AI-EES) and the Oil Sands Tailing Consortium (OSTC) in partnership with Alberta Energy, Natural Resources Canada, Alberta Environment and Sustainable Resource Development, and the Alberta Energy Resources Conservation Board. June 29, 2012. 5v. http://www.cosia.ca/media-resources/resource-library/

OIL SANDS TAILINGS TECHNOLOGIES ASSESSMENT AND EVALUATION

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ABSTRACT

Although the oil sands industry has spent many millions over the years on improving their technologies to mine oil sands, extracting bitumen, disposing of tailings, protecting the environment and rehabilitating mine-disturbed land, the industry is under a microscope regarding the environmental aspects of ongoing and legacy tailings disposal. As a result, the Alberta Government, in partnership with industry, developed a Tailings Management Framework to focus on these concerns. A Consortium of Tailings Management Consultants (CTMC) was awarded a contract by Alberta Innovates – Energy and Environmental Solutions (AI-EES), to prepare an Oil Sands Tailings Technology Deployment Roadmap. The intention of the four-part project is to help achieve more timely deployment of end-to-end tailings technologies by focusing industry and Government on existing technologies with the best chances of success and on select technologies that show promise but still need further development.

The project includes four components: Component 1 (C1) compiled an extensive list of technologies used, or with potential use, in the oil sands industry, including piloted and research technologies; Component 2 (C2) developed a set of Tailings Management and Reclamation (TMR) objectives and sub-objectives to evaluate the technologies; Component 3 (C3) developed the evaluation methodology and conducted the technology evaluations based on criteria from C2, to identify gaps, opportunities, and possible improvements; and Component 4 (C4) developed road maps using the evaluation results from C3 as well as the C1 data and report, to improve the process from mining through to final remediation and closure, and improved tailings management practises.

This paper describes the methodology that was used in C3 of the project, to evaluate technologies, and to identify gaps and opportunities.

INTRODUCTION

The overall objective of the “Tailings Roadmap” project’s is to create a technology deployment roadmap and action plan that will assist regulators and industry to create and implement technology solutions that will meet the goals of Alberta Environment’s (AE) *Tailings Management Framework* and the ERCB’s *Directive 074* and, in particular, the long-term reclamation objectives (stable, reclaimable, sustainable, and acceptable).

The role of C3 was to develop a method to evaluate the various technologies provided by C1, with reference to specific criteria provided by C2, and then to evaluate the technologies using the developed methodology.

COMPONENT 3 METHODOLOGY

When discussing the evaluation process at the start of the project, it was clear that balancing the multiple viewpoints, reams of data, and opinions of the participants was going to be a complicated undertaking. After all, many of the participants have spent careers in oil sands dealing with tailings management and while these long-held beliefs could provide insight, they could also hinder the process if not handled appropriately. Many of the operators had already chosen the technology/method they deemed most appropriate for their site. Naturally there was a tendency to want some technologies to come out of an evaluation as “winners” and others as “losers”. And at the end of this “Roadmap” project were the “consumers” - the regulators and more importantly the operators of the oil sands sites, each with specific operational circumstances and constraints, who needed to be able to utilize the results and apply them to their individual situation. It was clear that a method to evaluate the technologies without assigning arbitrary titles like “better or worse” was required. A broader evaluation framework that discounted ideas like "fatal flaws" and evaluated ideas and technologies on **all** their merits was needed. The evaluation tool built to accommodate the challenges listed above ultimately became the GoldSET (Golder Sustainability Evaluation Tool) model.

The development of the C3 evaluation process, and ultimately GoldSET, consisted of a multitude of steps. Foremost, the development of the C3 evaluation process involved gathering knowledge and expertise that covered all aspects (from a sustainability perspective) of each technology and identifying the critical factors required for evaluation. These sustainability aspects consisted of technical, environmental, social, and economic factors. The technologies up for evaluation spanned all the C1 defined mining life cycle categories of:

- Mining;
- Extraction and Bitumen Recovery;
- Tailings Processing;
- Deposition and Capping;
- Water Treatment;
- Reclamation; and
- Technology Suites.

The breadth of the categories and the depth of the sustainability factors required that broad expertise from the oil sands industry, engineering consultants, and regulators be utilized in the evaluation of the technologies. It was identified that no one person could possibly have enough specialized knowledge in each sustainability factor and across the phases of the mining life cycle to make a balanced evaluation as an individual. This necessitated the development of teams of people, each with specializations in certain areas, to participate in the evaluations. The GoldSET tool was adapted for the team evaluation concept and attempted to facilitate evaluations that would be consistent and defendable, and would take into account all available information.

Input from C1 and C2

The purpose of C3 was to develop a method to evaluate the various technologies provided by C1, with reference to specific objectives provided by C2, and then to evaluate the technologies using the developed methodology.
After C1 provided the Master List of Technologies and all relevant associated documentation within the seven defined categories, the level of development of the technology was realized as a new challenge. As a result, all technologies in each category were divided into three levels of maturity as follows:

1. Commercial – Currently used in the oil sands industry;
2. Development – Currently in pilot scale testing or used in other industries and believed to have good potential for application to the oil sands industry; and
3. Research – Technologies purported by suppliers, universities or operators as having some potential for application to the oil sands industry.

These three subdivisions were very different in terms of information available, technical uncertainty, and the required effort to "optimize" or implement each technology. A unique set of evaluation considerations were required for assessing the different stages of maturity of the technologies. These considerations concerning maturity level needed to be incorporated and accommodated in the evaluation process, and ultimately the GoldSET tool.

C2 provided the list of Tailings Management and Reclamation (TMR) objectives and sub-objectives that were based on regulatory requirements, such as those found in ERCB's Directive 074. Upon review, it was identified that these objectives would need to be linked to the evaluation criteria, which would be selected to perform the evaluation. A matrix table was developed and used to cross-reference the evaluation criteria with the C2 sub-objectives. This exercise assisted in deriving the appropriate evaluation criteria; however it also highlighted another challenge in the evaluation process. It was recognized that the evaluation criteria that had been derived from the C2 objectives would vary between evaluations of the defined mining life cycle categories. Thus, there would not be a consistent list of evaluation criteria, and these variances would need to be accommodated in GoldSET. Figure 1 illustrates an example matrix table. An "x" is placed in the box where a sub-objective applies to a particular evaluation criteria. The goal of the evaluation was to objectively and collectively assess the tailings technologies identified for each of the mine life cycle categories, using an evaluation process and an analysis tool that accommodated all challenges innate to the project and those that were recognized from C1 and C2 input.

	C2 Sub-objectives	Reduce legacy tailings volumes	Maximize use of recycle water	Maintain geotechnically stable landforms	Maximize flexibility to accommodate seasonal operation
C3 Evaluation Criteria					
Technical					
Design Complexity					x
Water Recovery		X	x	x	
Environmental					
Air Emissions					
Geotechnical and Seismic Hazards / Risk		X		x	
Social					
Health and Safety					
Corporate Reputation		X	x		
Economic					
Capital Cost		X			
Operating and Cost		X	x		x

Figure 1. Example Matrix Table – C2 Sub-Objectives vs. C3 Evaluation Criteria.

C3 Methodology

The C3 process was divided into phases which are described below:

- Phase 1: Develop a tool (GoldSET) to accommodate an evaluation process that discounted ideas like "fatal flaws" of a technology; that accommodated the levels of maturity of the technologies in each of the defined mining life cycle categories; and that integrated the C2 objectives and sub-objectives. GoldSET would assist in the data analysis of the evaluated technologies considering the challenges to the project and those recognized from the C1 and C2 input. The tool was developed to use the following components:

  1. Evaluation criteria (referred to herein as "indicators") and indicator importance weightings;
  2. Indicator clarifiers (descriptors) and ranking, i.e. scoring guidelines for each indicator; and
  3. Data quality determination (based on the evaluation team perspective).

- Phase 2: Assemble teams of experts to evaluate the technologies.

- Phase 3: Carry out detailed evaluations of the technologies in accordance with the developed process and evaluation criteria. The evaluation teams were provided with the following:

  1. Master tables of indicators, master legends, review sheets, and data sheets prior to evaluation;

2. Evaluation forms to capture qualitative commentary from evaluators throughout each session; and Data quality evaluation parameters.

- Phase 4: Process data in GoldSET and provide results of the evaluations.

This structured evaluation approach defines the criteria against which to evaluate the different technologies, a scoring scheme for each indicator, a mechanism to account for the relative importance of the indicators, and a process to deal with data quality. This approach was intended to allow the evaluators to identify the strengths and weaknesses of the technologies, while accommodating the challenges discussed above.

Phase 1: GoldSET Development
Indicators and Descriptors

Indicators were selected for each technology category, i.e. mining life cycle phase. The selected indicators were organized into four aspects: technical, environmental, social, and economic. A descriptor was developed for each indicator to clarify its meaning and reduce the subjectivity associated with the indicator. This was done to reduce the "interpretation" factor during the evaluations. An example of a dual interpretation for the indicator "tailings footprint" would be as follows:

3. Does tailings footprint refer to the external area?

4. Does tailings footprint refer to internal and external area?

Providing the descriptors also provided consistency and repeatability in the evaluation. An example of the indicators and descriptors in the Master Table for the tailings processing category is presented in Figure 2.

Weighting and Indicator Relevance

Each indicator was assigned a weighting of 1, 2, or 3. The weighting represented the level of relative importance of that indicator with respect to the other indicators in a particular mining life cycle category. This allowed appropriate emphasis to be placed on particular indicators that are critical to achieving the C2 objectives and sub-objectives. However, selection of indicator weightings was

CTMC TAILINGS ROADMAP

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MASTER TABLE
Category: Tailings Processing

#	Indicators	Descriptors	Weighting	Ranking Guidelines
	Technical Criteria			
1	Feed characteristics	Ability to accommodate tailings materials with different mineralogy, PSD, bitumen content, clay content, and solids content	1	0 = Very sensitive to minor changes in the listed variables 33 = Sensitive to moderate changes in the listed variables 66 = Sensitive to major changes in the listed variables 100 = Not sensitive to changes in the listed variables
2	Fresh tailings	Applicability to fresh tailings	2	0 = Not applicable to fresh tailings 33 = Applicable to one stream 66 = Applicable to two or more streams 100 = Applicable to all streams
3	MFT	Applicability to legacy MFT	2	0 = Not applicable to MFT 33 = Applicable to few MFT constituencies 66 = Applicable to multiple MFT constituencies 100 = Applicable to all MFT constituencies
4	Bitumen extraction	Negative impact on bitumen recovery (i.e. chemistry of recycle water)	2	0 = Major impacts 33 = Some impacts 66 = Minimal impacts 100 = No significant impacts
5	Fluid fine tailings formation	Ability to capture fines and reduce the degree of segregation during production, transportation, deposition and storage	3	0 = Unable to capture fine and ultrafine particles 33 = Capture most of the fines but unable to capture ultrafine particles 66 = Potential to meet the regulatory requirements for fines capture

Figure 2. Master Table – Category: Tailings Processing.

carefully considered in order to minimize the subjectivity within an evaluation. The weighting schemes needed to be agreed upon and remain unchanged during the evaluation. Changing the schemes to fit a desired outcome was considered a “fatal flaw” analysis. The established weightings can be used in combination with a sensitivity analysis to assess the overall effects of each aspect and weighting.

Additionally, the overall weightings for each of the four aspects were also considered in the GoldSET tool and, after discussion, were established as equal weights (25%) across the technical, environmental, social, and economic factors. An example of the weightings in the Master Table for tailings processing category is presented above in Figure 2.

Ranking and Ranking Guidelines

The ranking[1] is a number between 0 and 100 that represents the ability of a particular technology to satisfy the requirements of a particular indicator. The ranking guidelines constrained the evaluators to choose rankings of 0, 33, 66, and 100. The higher the ranking number for a technology indicates the more it meets the desired indicator condition. The guidelines were developed prior to the evaluation, and selection of an appropriate ranking was performed by the evaluation teams during the sessions. An example of the ranking guidelines in the Master Table for tailings processing category is presented in Figure 2.

Data Quality and Data Quality Description Table

A column was provided on the evaluation forms to indicate the quality of the information provided for each technology with regards to each indicator. In some cases, often for technologies in the research stage, the amount and quality of information provided in the datasheets was insufficient to make a reasonable evaluation for a given indicator. The three main concerns about data quality are described as follows:

1. Data or criteria not applicable;
2. Information not available; and
3. Information not substantiated.

For some technologies, a particular indicator was not considered to be applicable and was therefore not included in the evaluation, as noted by the evaluators. There were also cases where a technology was not evaluated because the evaluators considered that there was a lack of suitable information to select a ranking/score. The last concern was in reference to “claims” made in the technology data provided that was largely unsubstantiated. The evaluation teams were instructed to evaluate based on the data provided but to use the data quality evaluation to describe the uncertainty. The data quality value was determined as shown below in Table 1.

Phase 2: Evaluation Teams

The evaluators were selected based on their knowledge and expertise in either the technical, environmental, social, or economic areas (sustainability aspects). The evaluator teams of experts included multi-disciplinary specialists from geotechnical, geochemical, process, mining, water treatment, mechanical, reclamation engineers and scientists, hydrologists, biologists, economics specialists, and government representatives.

Phase 3: Evaluation Sessions

Eight evaluation sessions were completed over a period from December 6, 2011 to February 14, 2012. For each evaluation session, one or two teams consisting of 5 to 8 evaluators each were required. Each session was assigned a facilitator and a note taker to guide the session. Ample opportunity was given to discuss the technologies, to express individual opinions, and to write down comments to gain qualitative insight as well as the quantitative evaluations.

Only one technology was evaluated at a time. A consensus ranking was recorded where possible. If a consensus could not be reached, separate rankings were recorded by the individual evaluator, and the note taker recorded the ranking of the majority. The evaluators were encouraged to write their individual comments and the note taker also recorded comments presented during the evaluation. Evaluators were encouraged to keep in mind the level of maturity for the technology they were evaluating. When evaluating a commercial technology, the goal was to identify knowledge gaps as well as areas for improvement or areas of weakness for the technology. When evaluating a development technology, the goal was to identify gap fillers and to compare with other technologies.

[1] Note that in the context of this report, the term “ranking” does not refer to an order (i.e., better or worse, etc.), but rather to a score assigned to a technology that reflects its ability to satisfy certain requirements.

When evaluating a research technology, the goal was to identify gap fillers as well as opportunities worth pursuing.

Phase 4: Data Analysis

The numerical data collected from the evaluations was entered into the GoldSET tool, from which it was possible to analyze and summarize the data. The output from the tool included a visual presentation of the results in the form of an evaluation diamond and a data quality diamond as shown on Figure 3. The evaluation diamond is a simple way of presenting the data in a summary manner.

The four points of the diamond represent the four sustainability-based evaluation aspects (technical, environmental, social, and economic). The outer edges of the diamonds represent the maximum score (100%) or data quality (1) a technology can obtain. The darker shaded area within each evaluation diamond represents the weighted average score of each technology.

In general, the shaded area that is farthest from the outer edge represents an area where improvement could be considered for a given technology (a gap in the technology). The data quality diamonds illustrate the level of confidence of the evaluators' score, the lack of suitable information, and/or the applicability of the indicators. Ultimately, the greater the area of the data quality diamond, the higher the confidence in the evaluation diamond. Figure 3 is an example of an evaluation diamond for a technology that shows gaps in its ability to meet the environmental criteria. The Data quality diamond illustrates 100% confidence in the technical and social aspect and slightly less confidence in the evaluation results for the environmental and economic aspects.

Table 1. Data Quality Description.

No Data / Lack Info	Scoring Carried Out	Data Quality Value
	✓	1
✓	✓	0.5
✓	-	0
	Indicator N/A	0

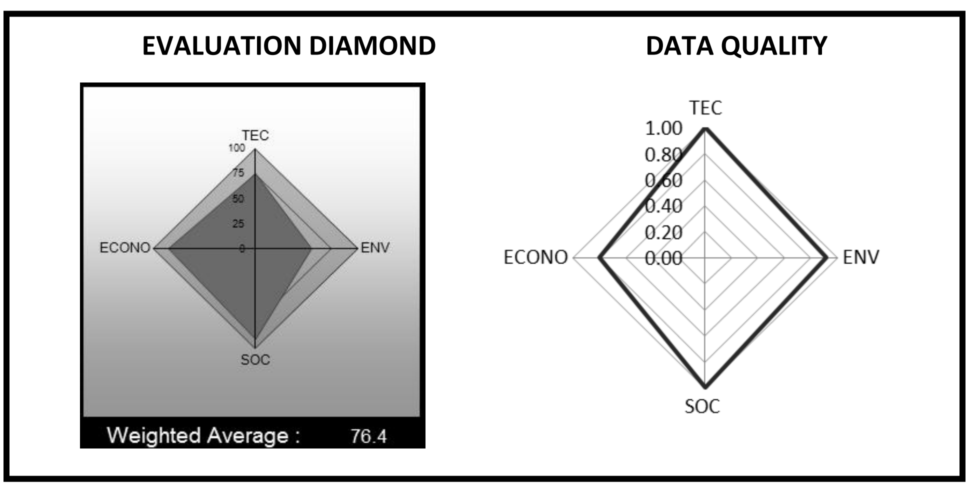

Figure 3. Example Evaluation Diamond.

CONCLUSIONS

The method used for the C3 evaluation provides traceability/accountability, allows for identification of priorities when making a decision, and helps to remove subjectivity from the decision making process. It has allowed for an informed decision to be made based on the factors considered to be the most important for tailings management and reclamation. The GoldSET tool can be used to further analyze the data for identification of gaps and opportunities. Sensitivity analyses can be run on the data to identify impacts of site-specific drivers on an ongoing basis.

THE OIL SANDS TAILINGS TECHNOLOGY ROADMAP PROJECT: THE IDENTIFICATION AND IMPROVEMENT OF TAILINGS TECHNOLOGY SUITES, AND PATHWAYS FOR TECHNOLOGY DEVELOPMENT

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ABSTRACT

In 2011, Alberta Innovates – Energy and Environment Solutions (AI-EES) awarded a contract to the Consortium of Tailings Management Consultants (CTMC) to complete a study to develop an Oil Sands Tailings Technology Deployment Roadmap to improve the current tailings management in the industry.

The study scope of work was divided into four main components: 1) to identify and describe all known tailings management technologies, throughout all stages of the mining life cycle, 2) to define important tailings reclamation objectives to which successful tailings technologies should contribute, 3) to evaluate the identified tailings technologies to determine their strengths and weaknesses, in light of these objectives, and 4) to identify technologies and/or suites of technologies which could improve the ability of tailings management practices to meet the previously defined objectives, and the pathways by which they could be brought through the research and development process to commercial implementation.

This paper describes the work completed during Component 4 of this study including: improvement of existing commercial technology suites and identification of new technology suites, development of roadmaps for furthering selected technologies and technology suites to commercial implementation, leveraging of synergies and collaboration in the development process, and defining the "Big Picture" context for the roadmap work.

The full text of the report is titled:

OIL SANDS TAILINGS
TECHNOLOGY DEPLOYMENT ROADMAP
Project Report – Volume 5
Component 4 Results

The full report is available at the following URL:

http://www.cosia.ca/media-resources/resource-library/

INTRODUCTION

Background & Outline of Component 4 Scope

The Component 4 team of the Oil Sands Tailings Technology Deployment Roadmap (TTDR) study was tasked with reviewing and assimilating the previous component reports in order to assess the current development state of identified technologies and technology suites. For the purpose of the project, a technology suite was defined as a combination of multiple technologies operating in tandem to fully realise their benefits. From literature reviews and input from Oil Sands industry experts, a technology development model was created, which was then applied to the individual tailings technologies and technology suites to produce development roadmaps to progress these technologies.

Component 4 focused on identifying technologies and/or technology suites that could improve the ability of tailings management practices and meet previously defined goals, by identifying the pathways by which they could be brought through the research and development process to commercial implementation. The Component 4 work process was broken down into a series of eight specific steps:

1. Review and assimilate information.
2. Select "highlighted" technologies that would improve existing commercial technology suites or contribute to the development of new technology suites.
3. Define the full Oil Sands tailings technology Research and Development (R&D) cycle.

4. Assess the benefits, risks and costs of the highlighted tailings technologies, and from this, select "priority" technologies for inclusion in the TTDR.
5. Develop detailed Roadmaps.
6. Identify synergies between R&D pathways.
7. Establish the "Big Picture" context for the Roadmaps.
8. Final Report.

Methodology

After an extensive review of the information provided from the Component 1, 2 and 3 reports, two internal workshops were held by the Component 4 team. The first workshop was held to evaluate existing technology suites; to identify improvement opportunities within these suites and to highlight the technologies with the potential to address problem areas. A second workshop was held in order to brainstorm new technology suites and to highlight those technologies with potential for achieving the Tailings Management and Reclamation (TMR) objectives. Technologies identified through these processes were highlighted for further evaluation.

A Generic Model of Oil Sands Tailings Technology Development was developed and used to provide a basis for the development of the technologies in the TTD roadmaps.

The benefits, risks and costs of the technologies highlighted in the two workshops were compared against one another to form a relative prioritization system. The prioritization system was used to identify a technology's potential to address current and future TMR objectives, allocating a high priority to technologies considered to show promise.

Roadmaps were developed for several technology suites. A structured development approach was used for all high priority technologies within each suite, as well as other technologies that were considered necessary for suite improvement. The format of the roadmaps allowed for synergistic aspects of technology suites to be identified, thus facilitating the identification of technologies with potential to contribute to multiple technology suites.

The final stage of the process was to outline the big picture development context of technology advancement in the Oil Sands industry.

THE ROADMAP PROCESS

Identification And Highlighting Of Technologies Through Analyzing Technology Suites

The purpose of the workshop process was to create a list of "highlighted" technologies that could improve existing tailings technology suites or contribute to new suites in order to focus further research and development efforts. The first workshop evaluated commercial technology suites, identified their improvement opportunities (i.e. performance in this area could be enhanced through the use of a technology), and identified technology gap fillers (a technology that addresses an identified improvement opportunity of a suite). Eight separate commercial tailings treatment suites identified by the Component 1 team were considered during this workshop:

1. Centrifuging MFT with conveyor/stacking.
2. Coke Capping.
3. Composite Tailings.
4. Conventional Tailings.
5. In-line thickening with accelerated dewatering.
6. In-line thickening with thin lift evaporative drying.
7. Thickening.
8. Water capped end pit lake.

It was determined in the first workshop that coke capping is not representative of an entire suite; due to the limited capacity to enhance the process through the addition of other technologies, therefore it was not evaluated further. Any technology identified as being a potential gap filler was included in the list of highlighted technologies for further evaluation.

A second workshop was held to review prospective lessons from other industries and apply them to the Oil Sands, as well as to brainstorm new and unique technology suites to improve current Oil Sands tailings management. The proposed new technology suites were theoretical and their processes were designed to meet desired outcomes (e.g. minimize operation footprint). The resultant suites were not considered to be viable or economic solutions for current practice, without further in-depth scrutiny. Proposed new technology suites included:

9. Improvement to Water Based Extraction.

10. Non-Aqueous Solvent Extraction.
11. Retort Based Extraction.
12. In-Pit Tailings Stream.
13. Parallel High/Low Fines Suite.

This exercise identified several technologies that would comprise the new suites; all were considered highlighted technologies for further evaluation.

The technologies identified through the workshop exercises formed a list of highlighted technologies. This list was subject to critical evaluation through cost-risk-benefit analysis (described below) to pare it down to a short list of technologies that warrant further investigation for research and development initiatives. The initial list was composed of all technologies that had been selected as gap fillers, or components of a new technology suite. In order to narrow the selection, all technologies considered to be in the commercial stage were not evaluated further due to industry wide understanding and acceptance of their capacity. Focus was to be put on technologies in either the research or development stages; any variations to current technologies were considered with respect to the stage of their applicability and not that of the parent technology.

The highlighted technologies were subjected to a cost-risk-benefit analysis. Each technology was evaluated with regard to a standard set of indicators provided from Component 3 of the study. The performance of a technology was used to gauge whether the indicator represented either a benefit or a risk for that particular technology. The benefit-risk performance was then compared with the expected technology cost rating in order to assign a prioritization. Technologies were then compared to others from the same group in the mining life cycle, i.e. all deposition technologies were compared against one another to show the data spread of the results. Individual data sheets were completed to present technology specific costs, benefits and risks.

The high priority technologies identified and evaluated as part of this process included (in no particular order of preference, and with their original numbers and project labels intact):

- T-024 Alberta Taciuk / T-548 Retort Based Extraction
- T-060 MFT Spiked Whole Tailings
- T-069 Solid Bowl Scroll Decanter Centrifuge
- T-085 Thermal Drying
- T-197 Super CT
- T-208 Paste Thickener
- T-267 Froth Treatment Tailings Thickening
- T-529 Oleophillic Sieve / Beads
- T-032 Accelerated Dewatering
- T-062 Co-mixing MFT & Overburden
- T-090 Vertical Drains
- T-099 Stacker Hydro-Cyclones
- T-188 Under-Drained Tailings
- T-510 Tailings Discharge Tremmie
- T-138 Water Capped MFT Lake
- T-550 Tailings Surface Sealants

Additional highlighted technologies evaluated due to their important contribution to technology suites improvement included:

- T-018 Hydrodynamic Cavitation
- T-020 FTT – Oil & Heavy Minerals Recovery/Thickening
- T-037 Thin Layer Freeze Thaw Cycling Dewatering
- T-039 Accelerated Evapotranspiration using Vegetation
- T-040 Thin Lift Drying Robinsky Cones
- T-065 Interlayered MFT/Sand Placement
- T-067 Cross Flow Tailings Filtration
- T-076 Pressure Filtration
- T-080 Vacuum Filtration
- T-088 Shock Densification
- T-185 Non Segregating Tailings from Cyclone Underflow, Thickener Underflow & MFT
- T-186 Solvent Extraction
- T-206 Non Segregating Tailings from sand and TT
- T-235 Poldering
- T-270 Conventional Hydraulic Fill
- T-438 Subaqueous Capping
- T-606 Mobile In-Pit Crusher
- T-600 Fine Sizing (Crushing)
- T-601 Waste Cooling
- T-602 Secondary (Deep Cone) Separation Vessel
- T-603 Control of Biogenic Gas during MFT Spiking
- T-604 High Temperature Heating for Solvent Removal
- T-605 Water Capped Lake (over TT)
- T-606 Mobile In Pit Crusher
- T-607 Mobile versions of conventional separation/extraction equipment
- T-608 Geotextile Drainage Layer

- T-609 Mobile Centrifuge
- T-610 High Density MFT Harvesting
- T-611 MFT Tank Thickening
- T-612 Vibrating Screens
- T-614 Shear Conditioning of Soft Tailings

A selection of appropriate supporting technologies was expanded upon in the roadmap section. It was important to consider that some technologies possess a very specific use while others could be utilised for a range of purposes.

Development of TDD Roadmap Model

Three different sources were considered in order to develop the Generic Oil Sands Tailings technology R&D model; a scan of published literature on the subject from other technology driven industries, interviews with key professionals in the Oil Sands industry and an internal work session to synthesize the results.

The resulting model (Figure 1) is constituted of four equally important quadrants: Formulation and Mobilization, Research, Development and Commercial Implementation. The model is comprised of 2 preliminary steps (within the Formulation and Mobilization phase) and 16 main steps. Potential pitfalls (issues that could result in fatal flaws to the development of a technology) and remedies (actions taken to avoid problems in development) were identified throughout the process. In addition, appropriate timeframe recommendations and a discussion on the development cycle were presented.

An overview of the Research & Development model for Tailings technologies follows, more information may be found in the project report: Volume 5.

Formulation and Mobilization

During this stage of development, it is important to establish the internal and external frameworks for technology development. The stage is comprised of two components:

1. Rumination
2. Team Mobilization

Research

The research stage is performed on a theoretical and laboratory scale in order to generate a solid understanding of the fundamental behavior of the proposed technology. The goal of the research stage is to establish if the technology can accomplish what it proposed to do. This stage is comprised of following components:

1. The idea
2. Conceptualization
3. Bench scale lab testing
4. Screening
5. Published database
6. Technology validation

Development

The development stage reflects increasing confidence in a technology's performance, whereby proceeding to each successive step increases the cost and operational complexity of a trial. The goal of this stage is to validate, and scale up a technology to full operational use. This stage is comprised of following components:

7. Pilot
8. Prototype
9. Techno-economic validation

Commercial

The commercial stage represents the ultimate stage for a technology and full scale incorporation into daily operations. During the commercial stage a technology must be evaluated against the requirements of both internal and external stakeholders. This stage is comprised of the following components:

10. Business case
11. Engineering
12. Stakeholder engagement and reclamation plan
13. Commissioning and start-up
14. Operation
15. Reclamation and closure
16. Reflection

Tailings technology development is seldom straightforward. There are a number of pitfalls that may be encountered during the process:

- Confusion within project teams regarding member roles and responsibilities.
- Making claims about a technology without being certain as to the validity of the claim.
- Allowing the overall view of the worthiness of a technology to be shaped by perceptions and preconceived ideas.

- Not dealing with fatal flaws in a timely manner.
- Not fully analyzing the data during and after the pilot trial, resulting in a lack of understanding of the results / process.
- Scaling up trials too early in the development stage and missing the early discovery of flaws at points where they can be altered before costs become prohibitive.
- Underestimating capital and lifecycle costs.
- Continuing the development of a flawed technology due to the level of corporate commitment.
- Lack of communication and understanding between operations, management and engineering teams.

In noting the potential pitfalls, it is important to note that they are not without remedies:

- Development of criteria for deciding when to proceed forward in the development cycle.
- Understanding by management and decision makers that the success of the team is not tied to the commercial success of the technology but rather to the quality of the investigation.
- Continuous researching of the fundamentals.
- Remaining open to scientific critique of the technology in question.
- Continual examination of the technology for fatal flaws, at every step of the development process.
- Building time into the schedule to properly analyze the data and understand its implications to the technology's performance.
- Involving operational staff during the planning and development of the technology prototype.

Timeframes

The time required for each step and each stage is highly variable. The time required is linked to the category of the item being developed: to introduce new instrumentation might take weeks to months; a new technology might take several years; and an entire technology suite could take upwards of five or more years. Even these times are subject to the complexity of the technology in development.

In order to shorten the development process it is essential to build upon the foundations of previous ideas, and to be open to adapting lessons learned from other industries. An observation by an expert in Oil Sands tailings development was that reducing organizational hesitation in decision making could reduce the timeframe for tailings technology development by as much as 50%.

It was found that there were potential consequences to rushing technology development; skipping steps may bypass the opportunity to identify a technology flaw at an early stage where it could easily be addressed without major setback or cost, as opposed to identifying the same issue at a point where it could entail very high cost, consume valuable resources or potentially derail future development.

Technology Roadmaps

Nine different roadmaps were developed to address both current and proposed technology suites:

Current Technology Suites

- Centrifuging MFT with conveyor/stacking
- Composite Tailings
- In-line thickening with accelerated dewatering
- In-line thickening with thin lift evaporative drying
- Thickening
- Water capped end pit lake

Proposed Technology Suites

- Improvement to water based extraction
- Non Water Based Extraction (an amalgamation of solvent extraction, retort extraction and the parallel high/low fines suites)
- In-Pit Tailings Stream

Each suite roadmap incorporated technology components for that suite, addressed areas with potential for improvement and suggested further development of technologies that would benefit the suite as a whole. Within the roadmaps, the generic R&D model was applied to individual technologies to reflect their respective stage(s) of development.

Each roadmap consisted of several components:

- **Suite introduction** – A brief description of the purpose of the suite and its process.

- **Improvement opportunities** – A table summarizing the improvement opportunities and respective gap filler technologies identified through the Component 4 work. This was only completed for existing technology suites.
- **Technology breakdown** – Segmentation of the identified gap filler technologies / suite components into commercial, research, development or new technologies. Only research and development technologies were singled out for analysis by technology deployment table.
- **Suite diagram –** Sketches showing the technological process for commercial suites, or flow charts displaying the proposed methodology of the new suites.
- **Suite specific R & D Model** – The generic model (Figure 1) was populated with the technologies specific to each suite; the location of the technology on the graphic reflected the technology's current stage of development.
- **Technology Deployment Table** – A detailed look at individual technologies and their capabilities.

Though the emphasis of the roadmap was placed on the technology suites, the substance of the roadmaps focused on the technologies that constituted the suites. A technology deployment table was completed for each technology identified as being at the research or development stages, or a variation of a current commercial technology. Each table presented information on the respective technologies in the following format:

- **Technology Name**
- **Technology No.** – As identified by the Component 1 Compendium of Technologies.
- **Parent Technology Suite** – Identification of the technology suite(s) of which the technology is a component.
- **Potential Technology Contribution** – Identification of the capacity of the technology either as a gap filler, as a component of a current technology suite, or as a promising technology.
- **Technology Priority Status** – Result of the internal prioritization process, yielding low, medium or high priority values.
- **Background information** – Cost-risk-benefit sheets produced by Component 4 addressed specific information for each technology.
- **Development status** – The current development status of the technology, supported by previous efforts conducted by academia and industry.
- **Expected performance** – The expected performance/outcome of the technology's use.
- **Steps to progress the technology** – General steps required to progress the technology's development, referenced from the R&D model as well as steps specific to the individual technology.
- **Expected timeline** – Expected timeframe to progress the technology to a certain stage of development.
- **The Barriers, roadblocks and pitfalls to achieving success** – Issues that may prevent successful development and implementation of the technology.
- **Remedies and measures to overcome these barriers** – Identification of potential solutions to address the roadblocks to further development.

As a result of this exercise, a number of potential synergies for development were identified:

- Technologies that showed promise across a large number of applications.
- Cross-connecting technologies: a technology which could be beneficial in more than one application.
- Avoidance of duplication of effort: communication and co-operation have increased with the formation of many recent initiatives, and should continue.
- Batch testing and comparative evaluation of chemicals, amendments and enhancements.
- Technologies with similar life cycles or development pathways.
- Performing R & D in the same place.

A strategic consideration of these synergies is likely to yield a number of benefits for the development of Oil Sands tailings technologies.

Big Picture Thinking

When planning for the future development of a technology, the big picture needs to be accounted for as this can minimize potential barriers.

Five primary aspects should be kept in mind to achieve continued success, though this is not an exhaustive list (Boswell, 2010):

- Operational
- Technical
- Public
- Environmental
- Legal

While having sound technical and operational practice is important to improve the tailings management in the Oil Sands industry, taking account of the public's capacity for influence is crucial for success; public perception of the Oil Sands is shaped through several sources. An optimal engineering solution takes full advantage of transparent and efficient stakeholder engagement. Examples of stakeholders in such projects include:

- Mining companies and investors
- First Nations communities
- Consumers
- Media
- Government Organizations (AEW, ERCB, Alberta Infrastructure, etc.)

The environmental landscape and concerns have continuously evolved over the life of Oil Sands projects; the continued progression of public and political interest has placed the industry under intense scrutiny regarding environmental performance. Several environmental areas require diligent consideration:

- Sustainability
- Environmental Impact Assessment (EIA)
- Ecology
- Water use
- Land use
- Cumulative impact
- Climate change

Developing technology is required to recognize that environmental regulations can and will change and that operators will need to work with all stakeholders to achieve the evolving regulatory and reclamation goals.

Effective communication between the Oil Sands industry and regulators is essential to ensure that the requirements of public and environmental stakeholders are met in a coordinated manner by contributing in the following ways:

- Anticipating future needs and changes in legislation.
- Proactively commenting on any new draft tailings legislation and suggesting alternative formulations.
- Understanding and working with the regulators to establish reclamation solutions and satisfy current regulations.

Technology development in Oil Sands tailings does well to consider every component of the big picture, from operational challenges to public engagement and regulatory requirements, in order to achieve the TMR objectives and ultimately those of sustainable development.

LESSONS LEARNED, CONCLUSIONS & RECOMMENDATIONS

There were many valuable lessons learned throughout this project, and they included:

- More input could have been used from other disciplines in order to balance the strong inputs from Tailings experts.
- There was room for more substantial contributions from tailings technology providers and vendors that would potentially improve the practice of Oil Sands tailings management.
- A number of technologies were not included in the study due to a lack of information; effort should be made to flesh information on these technologies out for evaluation down the road.
- Varying perspectives in evaluating technology made for challenges in arriving at a single assessment of a technology.
- There are three distinct avenues of application for tailings technology:
    - Legacy tailings,
    - Ongoing tailings generated from extraction plants already in production,
    - Tailings anticipated to be produced by future mines and extraction plants.

  The way forward for a technology or technology suite is highly dependent on the application.

Some of the conclusions of the project included:

- There is still no “silver bullet” tailings technology, i.e., a single technology or suite which will solve all of the Oil Sands tailings problems with a single effort.
- A significant number of technologies have been highlighted, and represent substantial opportunities for further development in the Oil Sands.
- The generic model graphic is a useful tool for visualizing technology development.
- Synergies exist in the development pathways for highlighted technologies.
- Nine detailed Tailings Technology Development Roadmaps were developed.
- The Big Picture context should be borne in mind as the Roadmaps are implemented.
- Final implementation of a tailings technology will also depend on many external factors to development, including:
    - Site/ lease conditions and environmental character.
    - Water regime.
    - Stage of the mine life cycle.
    - Existence of legacy fluid fine tailings.
    - Water and clay chemistry and geochemistry.
    - Relative importance of capital and operating cost.
    - Resource requirements, including human capital.
- Timelines for bringing potential tailings technologies to full commercial implementation are variable and lengthy.
- There is opportunity to increase performance and decrease the cost of existing commercial operations through several methods (not limited to):
    - Reduce segregation during beaching
    - Increase deposit strength & improve trafficability
    - Improve fines capture/storage
    - Address geotechnical risks
    - Reduce long-term settlement
    - Address long term environmental impacts
    - Potential for pre-treatment to remove bitumen
    - Separate fines rich from fines poor ore
    - Separate tailings streams

A number of recommendations were made during the creation of the roadmaps:

- Individual operators should assess the technology roadmaps and develop a screening program for their own leases.
- The threshold for vendors to make their technology available for testing and use should be reduced.
- A more rational and consistent R&D model / process needs to be implemented by most major Oil Sand operators
    - Anticipative research
    - Manage development timeframes
    - Improvements in corporate governance to streamline the process.
    - Reduce developmental hesitation to reduce the timeframe required.
- Consider lessons from other industries.
- Partnerships between industry operators and tailings technology providers should be encouraged in order to strengthen the bond and leverage development.

Progress does not rely solely upon planning, but also in taking action once the plan has been formulated. In order for the generic model and roadmaps to have an impact, they should be considered as tools for the Oil Sands industry to use for both current and future technology development.

Figure 1 portrays graphically the generic Model for Oil Sands Tailings Technology Development. Through use of this map with individual technologies a visualisation to the technology’s status and progress in development may be determined.

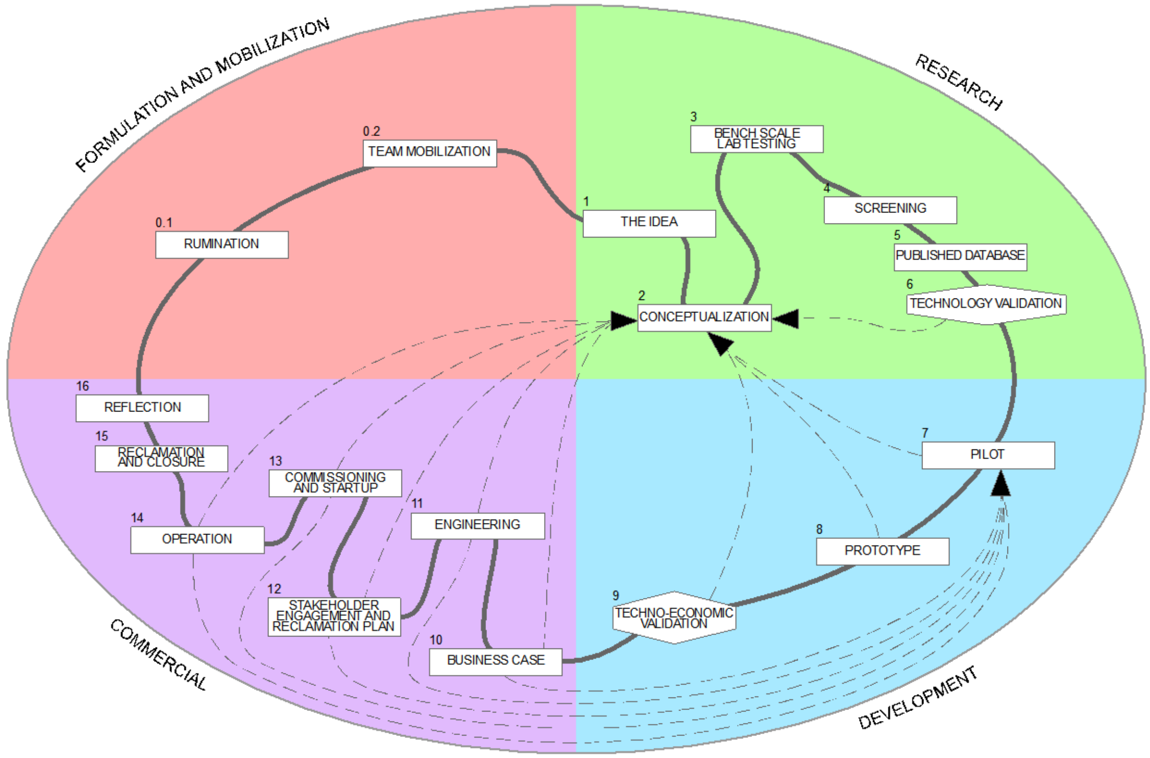

Figure 1. Generic Research & Development Model

ACKNOWLEDGEMENTS

The sentiments expressed by Dr. Sobkowicz in his keynote address at this conference are echoed by the authors of this paper.

There are many colleagues who worked with us across industry, academia and government and who gave tirelessly of their time to contribute to the project, to which our thanks are gratefully extended.

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EVALUATION OF OFF THE SHELF AND OFF THE WALL TAILINGS TREATMENT TECHNOLOGIES: A CAUTIONARY TALE

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ABSTRACT

Oil sands tailings management has struggled over the years with the perception that little or no progress has taken place towards the remediation of the tailings ponds. Most recently, the interest in oil sands tailings has been piqued with the introduction of directive 74, which mandates tailings remediation on an escalating scale, so that by 2013, 50% of the fines in the processed ore must be separated and treated to create a 5kPa deposit in one year. The steady (but slow) progress made by the industry in tailings management in the past 20 years is often lost in the din of “solutions” to the “problem” of oil sands tailings being proposed on all fronts. A simple, small scale, and easily interpreted approach to screening or rating the performance of various fluid fine tailings technologies and/or process aids is proposed and discussed in the context of some of the off the shelf and off the wall approaches evaluated over the past 20 years.

INTRODUCTION

The effect of water chemistry on oil sands tailings behavior has been a subject of study for many decades. Unfortunately, there are still some misconceptions that oil sands tailings management challenges are due to some unique property of the oil sands tailings. In fact, the only potentially unique property of oil sands tailings is the extent of hydrocarbon contamination. The role of bitumen contamination in defining tailings properties can sometimes be difficult to understand when conventions vary as to what constitutes “solids” in a tailings sample. Most often solids would be the mineral solids, with the bitumen and water content of the tailings sample defined separately. Some studies, however, use oven-dried solids that would include bitumen as part of the solids fraction. Many early studies on tailings did not specify other important parameters such as the fines content (less than 44 micron mineral) of the mineral components, the water chemistry of the pore fluid, or the bitumen content. Later, it became clear that defining the 44 micron fraction of the tailings was often inadequate and the clay size needed to be specified.

Recognition of the shortcomings in defining sample properties prompted the development of a set of characterization criteria for MFT samples under investigation. The white paper outlining these criteria (developed by D. Scott at the University of Alberta) was widely distributed, but unfortunately not widely adopted. Generally, characterization of important sample parameters did improve in the geotechnical community, but not necessarily elsewhere. Unfortunately outside of the research community there is little recognition of what is important in determining tailings behavior, and more importantly, what tailings properties are important in defining the tailings management challenges. This paper proposes to outline and hopefully dispel some of the common misconceptions about tailings behavior, using data from the Fine Tailings Fundamentals Consortium (FTFC 1989 to 1996 and elsewhere(1-3)).

In order to explain complex tailings behavior and the even more complex tailings management challenges to the general public, simplifications are made. For example, news stories in the popular press note the slow settling of oil sands tailings, and the “hundreds of years” that would be required to begin the reclamation process. Talk about the “new” (insert here consolidated/composite, TRO/AFD (4,5), or centrifuged) tailings process that reduces this time to months, days, or hours and one has the introduction to any number of news articles.

Many “solutions” to the fine tailings accumulation “problem” are based on the “discovery” that tailings settling rates are significantly increased by some type of chemical treatment, or by the removal of bitumen. The following discussion will demonstrate why changing the tailings settling rate via bitumen removal or chemical intervention does not substantively change the long term fluid fine tailings management challenges. This will include cautions on the recent interest in methods to densify tailings as well as the claims for novel bitumen extraction processes that purport to result in no accumulated fluid fine tailings.

RESULTS AND DISCUSSION

One of the most commonly quoted discussions of the role of bitumen in determining tailings behavior was by Scott et al (6). Figure 1 shows the mature fine tailings model where bitumen controls the behavior and prevents complete consolidation of the clays and minerals. Figure 2 shows a mineral only interpretation of the lack of MFT consolidation based on the extent of a water envelope associated with the various mineral types (adapted from Yong and Sethi (7)). Figure 3 shows an updated version of the mineral control model, accounting for the swelling character of the kaolin and illite, and the lack of significant amounts of amorphous iron oxides.

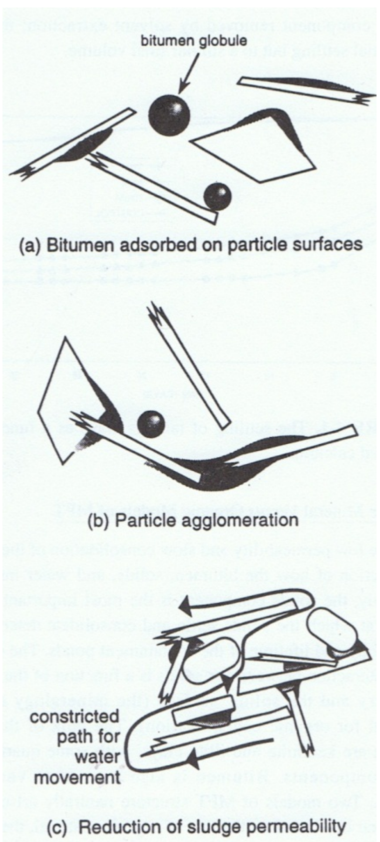

Figure 1. The bitumen controlled MFT model proposed by Scott et al (3).

Figures 4 and 5 show the settling rate and final volume for a bitumen free clay suspension compared to the settling of a middlings sample from a batch extraction test. The batch extraction test middlings are at 9% solids, 100% less than 44 micron and 50% less than 2 micron, with 1% residual bitumen content. The ideal clay suspension is kaolin from the Georgia clay sample bank and here the mineral is 100% less than 2 micron in size.

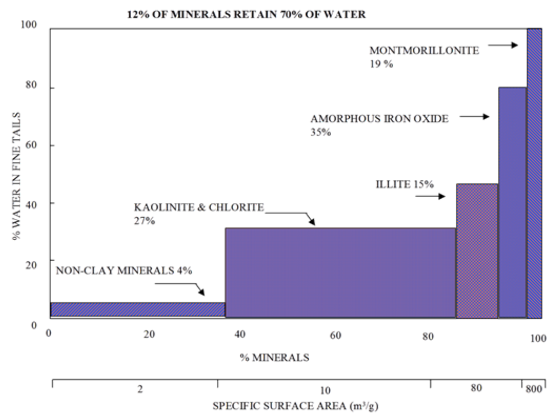

Figure 2. The mineral controlled MFT model proposed by Yong and Sethi (4).

The initial settling rate is slower for the mineral with the bitumen component, but the final settled volume is determined by the mineral content and mineral size. For the pure kaolin clay sample, the settled volume is approximately twice that of the mineral suspension from the extraction test. The batch extraction test is from the OSLO (other six lease operators) process, which at the time was simply a non-caustic extraction test using kerosene as a collector and MIBC (methyl isobutyl carbinol) as a frother for the flotation process.

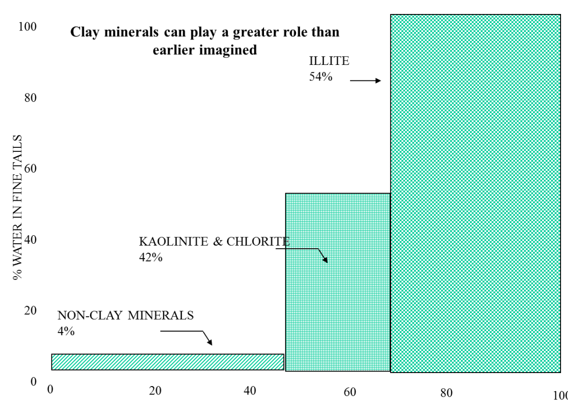

Figure 3. The mineral controlled MFT model modifications proposed in the FTFC (1).

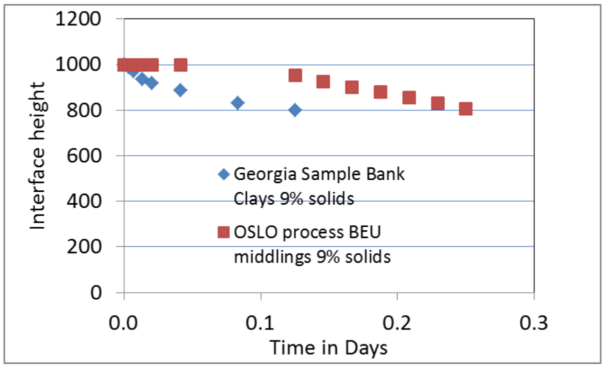

Figure 4. Increased short term settling rate with bitumen free clay minerals.

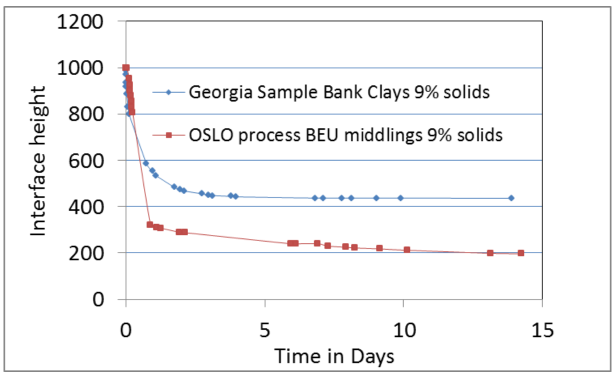

Figure 5. Settled volume differences related to total clay content for the samples from Figure 3.

The differences in settling rates for the OSLO and Clark hot water extraction process tailings were evaluated by the FTFC, with the conclusion that although the initial settling rate is determined by the water chemistry, the long term MFT accumulation is determined by the particle size.

Water Chemistry, Mechanical Energy, and Effective Particle Size

FTFC work showed that the effective particle size is influenced by water chemistry. This is highlighted in Figure 6; a hydrometer based particle size distribution. The OSLO or non-caustic fine tailings have no fraction behaving as particles less than 2 microns in size. In the higher bicarbonate caustic process, about 50% of the mass settles as though it was less than 2 microns in size. In addition, when the OSLO tailings have the water chemistry modified to include 1000ppm bicarbonate, they show the same 50% less than 2 micron fraction. Figure 7 shows FTFC comparisons of the tailings settling for the Clark and OSLO extraction processes. The initial settling rates are considerably faster for the non-caustic OSLO process compared to the Clark extraction water chemistry. The treated water example was the OSLO process with added calcium in the extraction water. The differences in the settled volume are very small for the different Suncor and Syncrude ores (8).

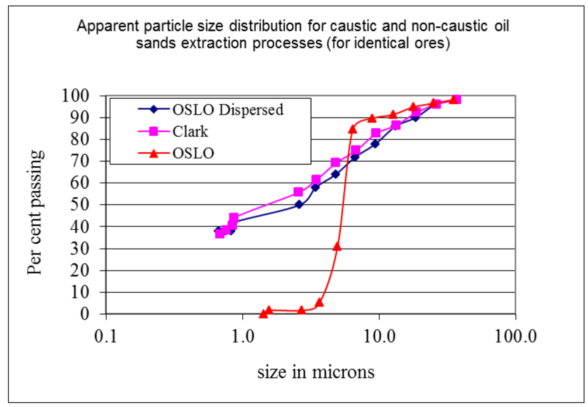

Figure 6. Effective particle size distribution for the OSLO (A: low bicarbonate) and Clark (B: high bicarbonate) extraction processes.

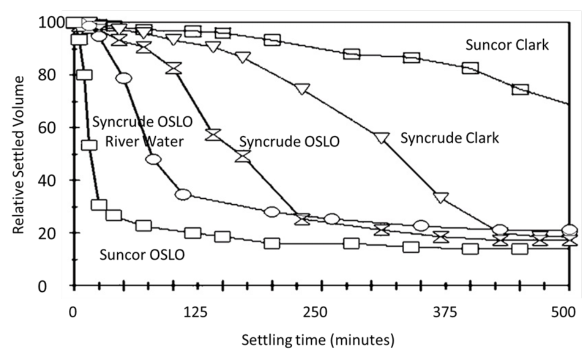

Figure 7. The settling rate for the OSLO (low bicarbonate) and Clark (high bicarbonate) extraction processes for two ores.

Figure 8 shows how MFT dispersion increases with increased mechanical energy. In the series of three MFT samples diluted by a factor of 4 in tap water, the samples were stirred for 10minutes, 5minutes, and 1minute respectively. In the left photo after one day, almost no evidence of settling is apparent in the most aggressively mixed sample. In the right photo, after three days, the two samples with the least amount of mixing energy have almost completely returned to the pre-mixing volumes. Understanding the tailings stream energy in lab and pilot programs is an important aspect of understanding tailings behavior.

Volume Changes Associated with Tailings Densification

The distinction between the actual tailings mineral size distribution and its effective size distribution as determined from something like a settling test is often misinterpreted. There is no end to the "new" extraction processes or tailings treatments that simply increase the short term settling behavior. Reference to the ternary diagram popularized by Scott et al clearly shows the transition from liquid to solid (for a 100% less than 44 micron tailings sample; see Figure 9) occurs at about 66% solids. This would correspond to a clay to water ratio of about 1.0 for a typical MFT with 50% clay on fines. Table 1 shows the impact of extreme manipulation of the tailings chemistry in order to determine the limit of densification with chemical manipulation. Although significant volume reductions are sometimes achieved, they are not nearly what would be required to have the solids content approach the liquid-solid boundary.

Figure 8. The effect of mechanical energy on tailings clay dispersion. From left to right, the 4x dilution MFT was stirred for ten, five, and one minutes respectively. The photo on the left is after 24h settling and the photo on the right is after 72h.

The simplification that the tailings accumulation in surface mined oil sands is due to the slow settling does not mean that quickly getting to 40% solids will solve the reclamation challenges. Figure 10 shows how significant the volume changes are as solids contents change. Moving from 30 to 50% solids results in an almost 50% volume change, yet Figure 11 shows that this is still a relatively weak material. In order to get to the 60 to 70% solids content that represents the liquid-solid boundary, another 50% volume reduction would be required. Note that the 10kPa yield strength required by directive 74 will not be achieved until the MFT has reached a solids content of about 80% (9). The change from 50 to 80% solids would require another factor of two reduction in volume.

Evaluating Tailings Densification Claims

Pilot scale evaluations of extraction technologies inevitably conclude that the resulting tailings behave in a manner that would significantly improve tailings management. The dispersion of fluid tailings with mechanical energy has been demonstrated in many examples, and the extent of this dispersion is impossible to duplicate in a pilot test with short tailings pipeline runs. As a result, all extraction processes with claims for improved tailings properties need to be evaluated with caution. One can wonder what role this factor might have played in the original mismatch between predicted and actual tailings behavior in the first commercial extraction developments.

Evaluating the benefits of a tailings densification process needs to be based on an understanding of the increase in solids content (fines over fines plus water for minus 44 micron tailings) or the clay to water ratio increase. Without an understanding of the particle size distribution of the tailings treated, along with the initial solids content, it is simply not possible to determine if claims have any merit

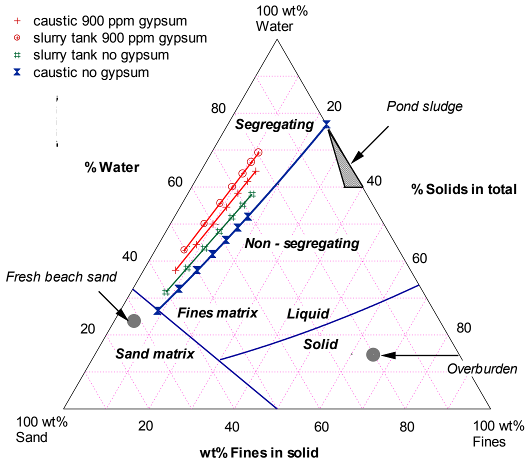

Figure 9. The ternary diagram showing the liquid-solid transition line.

The physics and chemistry of the tailings behavior is well established, and confirmation of performance claims requires data that would demonstrate a significant volume reduction and concomitant increase in the clay to water ratio. As mentioned earlier, there are no shortage of technologies that purport to solve the tailings accumulation by increasing the settling rate. Although these increases may seem impressive, they are nowhere near what is required to have an improvement over the business as usual case. To go from a typical MFT at 30% solids to the liquid-solid transition would require a 4x decrease in tailings volume.

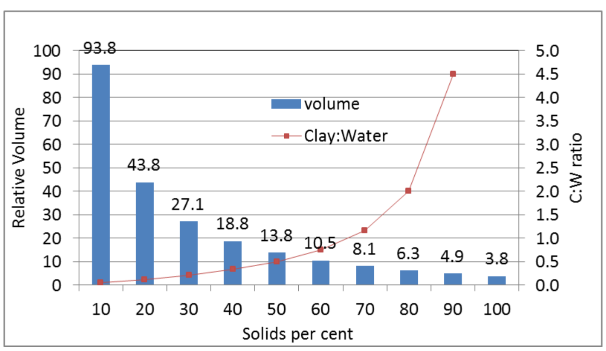

Figure 10. The relationship between solids content, tailings volume, and clay to water ratio.

SUMMARY

The FTFC established that removing bitumen from the tailings only increases the initial settling rate and does not increase the clay to water ratio. Water chemistry changes can positively influence the final settled volume, but not over a significant range considering the magnitude of the chemistry changes involved.

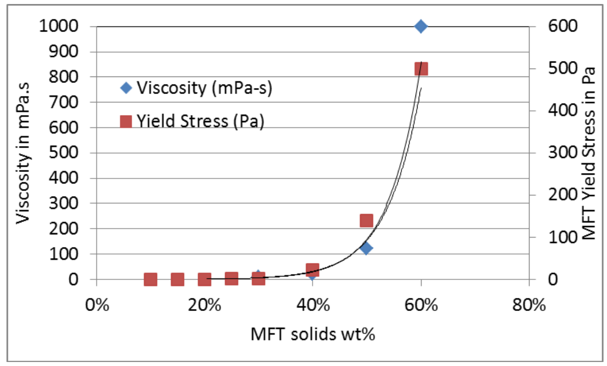

Figure 11. The relationship between viscosity and yield point as a function of solids content for a typical tailings.

New tailings process claims that accumulation of fluid fine tailings is unnecessary simply because of a minor increase in settling rate due to some process or another should be benchmarked against limits in behavior defined by work done by the Fine Tailings Fundamentals Consortium. In addition, the proposed mechanism for the improvement needs to be critically evaluated against the same performance limits.

In order to reach the liquid-solid boundary delineated in the ternary diagram, some energy input is going to be required in order to evaporate the excess water or to provide a significant loading on the deposit. The capillary forces holding water in the suspension as the clay to water ratio increases would preclude any other mechanism.

ACKNOWLEDGEMENTS

Thanks to Don Scott and Amar Sethi for useful discussions on tailings fundamentals going back to 1989.

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12. Energy Resources Conservation Board (2009). Directive 074: Tailings performance criteria and requirements for oil sands mining schemes, February 2009.

Table 1. Volume Reductions and Final Solids Content for Various Chemical Treatments.

<table>
<tr><td colspan="11">Initial Conditions: 33% MFT, 0.5 clay to fines, 48 days settling</td></tr>
<tr><td>Volume Reduction</td><td colspan="4">Ca mg/l</td><td>K mg/l</td><td>Mg mg/l</td><td>Na mg/l</td><td>C:W</td><td>SAR</td><td>Final Solids wt %</td></tr>
<tr><td>32%</td><td colspan="4">573</td><td>11</td><td>107</td><td>381</td><td>0.40</td><td>5</td><td>44</td></tr>
<tr><td>33%</td><td colspan="4">1269</td><td>28</td><td>146</td><td>419</td><td>0.40</td><td>4</td><td>45</td></tr>
<tr><td>31%</td><td colspan="4">2709</td><td>34</td><td>173</td><td>437</td><td>0.39</td><td>3</td><td>44</td></tr>
<tr><td>33%</td><td colspan="4">230</td><td>20</td><td>858</td><td>378</td><td>0.40</td><td>4</td><td>45</td></tr>
<tr><td>25%</td><td colspan="4">297</td><td>25</td><td>1902</td><td>393</td><td>0.35</td><td>3</td><td>41</td></tr>
<tr><td>13%</td><td colspan="4">47</td><td>8</td><td>31</td><td>1164</td><td>0.29</td><td>46</td><td>37</td></tr>
<tr><td>20%</td><td colspan="4">82</td><td>13</td><td>53</td><td>2098</td><td>0.32</td><td>63</td><td>39</td></tr>
<tr><td>6%</td><td colspan="4">489</td><td>85</td><td>244</td><td>76836</td><td>0.26</td><td>998</td><td>35</td></tr>
<tr><td>7%</td><td>9</td><td>5</td><td>5</td><td>271</td><td>0.27</td><td colspan="3">25</td><td colspan="2">35</td></tr>
</table>

$$SAR = \frac{[Na]}{\sqrt{1/2([Ca] + [Mg])}}$$

A REVIEW OF DESIGN AND OPERATIONS CONSIDERATIONS ASSOCIATED WITH THICKENING TECHNOLOGIES AND THEIR APPLICATION TO OIL SANDS TAILINGS

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ABSTRACT

With current accumulated volumes of MFT exceeding 800 M m^3, limited solids sedimentation of fluid fines tailings after decades, storage of large unusable volumes of water in tailings storage facilities, and stipulations for fines capture included in Directive 74, current and future operators in the Alberta oil sands are investigating a wide range of tailings management technologies including use of thickened tailings. Shell Canada's thickened tailings product (TT), and Suncor's implementation of TRO™ process demonstrate potential application of thickening and tailings dewatering technologies to enhance management of oil sands fluid fines tailings at full scale production rates. While several advantages are indicated through use of these technologies, such as achieving desired rates of fines sequestration, several key factors impact overall definition and achievement of success with these technologies both within and outside of oil sands industry. Factors include: physical and chemical properties of pre and post thickened tailings waste streams and associated thickened tailings product variability; configuration and preparation of the tailings disposal area; effective operation and monitoring of deposition systems; and post-deposition in-cell management and handling of thickened tailings. When incorporated into a comprehensive tailings management system, these factors combine to enable not only short term optimization of the storage capacity of dedicated disposal areas (DDAs) but have the potential to produce a stackable tailings product that both limits land disturbance required to store and reclaim thickened fine tailings and significantly reduces tailings management costs associated with polymer use, and tailings handling.

This paper reviews lessons learned from use of thickening technologies in the mining world outside of the oil sands where these technologies have a longer history of use. The paper also explores how these lessons relate to what is already known about application of these technologies within oil sands industry and how they may be integrated to develop the sort of robust fine tailings management system that is required in the oil sands.

INTRODUCTION

Background

In 2010 the Alberta Energy Resources Conservation Board (ERCB), primary provincial regulator for Alberta oil sands operations estimated that approximately 170 square kilometers of land was occupied by oil sands tailings ponds which contained approximately 840 M m^3 of fluid fine tailings (FFT) (Alberta Energy Resources Conservation Board, 2010). By 2020 this tailings volume is estimated to increase by 30% if significant measures are not taken to both reduce existing FFT inventories and to limit the rate of FFT production associated with existing and new oil sands projects approved for development (Grant et. al, 2008). In addition, the solids concentration of produced FFT, referred to in the industry as mature fine tailings (MFT), averages approximately 35% by weight, and has been practically unchanged in the last 35 years (Wells, 2011). Consequently, removing water from FFT and increasing fines capture has become the main goal of almost all tailings treatments strategies in the oil sands industry. However, it is the opinion of the authors that the ultimate goal of these operations should be production of trafficable, load-bearing surfaces suitable for inclusion in long-term closure landscapes that are both chemically and physically stable.

As a result of requirements primarily associated with the ERCB's Directive 74 (2009), oil sands industry has shown and continues to show increasing interest in tailings dewatering technologies including use of thickened and paste tailings technologies. This directive stipulates production of tailings with minimum undrained shear strength of 5 kPa within the first year of deposition; and deposition of tailings that are ready for reclamation within five years of commencing active deposition (tailings must have a trafficable surface layer with minimum undrained shear strength of 10 kPa).

Tailings dewatering strategies implemented in the oil sands should:

- Optimize reuse and recycling of process water thereby further reducing fresh water demand;
- Reducing the environmental, social, and potential economic risk profile associated with tailings storage facilities storing predominantly FFT;
- Facilitate better optimization of storage capacity within tailings storage facilities;
- Reduce and limit inventories of FFT;
- Minimum demand for dam embankment construction material and account for the natural topography, climate, and watershed profile of Northern Alberta;
- Limit footprint of areas disturbed associated with tailings storage;
- Be integrated into an overall tailings and water management system; and
- Optimize costs associated with tailings management and site reclamation.

THICKENED TAILINGS OUTSIDE OF OIL SANDS INDUSTRY

General Overview

Initial development of conventional thickened tailings technology has been attributed to the work of Dr. Eli Robinsky and his application of this technology at the Kidd Creek mine in northern Ontario in 1973 to the surface disposal of copper-zinc tailings. That it would take another 22 years of technology upgrades and operations modifications to fully realize Dr. Robinsky's objectives for implementing this technology is anything but trivial consideration (Robinsky, 2000).

The technology as applied at Kidd Creek was initially developed to facilitate central discharge of a thickened tailings product to create a self-supporting hill or ridge thereby minimizing use of erodible and expensive dyke construction material in flat to gently sloping terrain (Robinsky, 1999). To this end, the concept of a centrally deposited thickened tailings product has been widely applied throughout the metal and non-metal mining world, and especially in Australia where evaporation rates are high and available sources of fresh water are limited.

In a simplified description of this process, tailings dewatering is achieved by using high-density thickeners in which larger tailings solids (underflow) are allowed to settle to the bottom of a thickening tank prior to discharge into the designated tailings storage area. The overflow, typically clear water, is returned to the plant for continued use. The degree of thickening, pH and temperature impact the slope of discharged tailings which typically ranges between 1 and 3 percent. Based on climate, these gentle slopes can reduce occurrence of erosion while maintaining enough drainage for vegetation.

However, thickeners are not the sole means of enhancing tailings dewatering rates. As such, dewatered tailings are produced throughout the wider mining world using a variety of methods including cycloning, centrifugation, filtration, belt presses, and flocculation either individually or in combination.

Production and storage of dewatered tailings limits the size (and in some cases even existence) of a supernatant pond. This avoids potential compounding of tailings and water management issues. In other words, dewatering of tailings at the earliest opportunity allows for optimal return of process to a separate water storage facility or for direct return to the plant. This limits the amount of water to be managed within the tailings facility. It also allows tailings facilities to be managed as tailings facilities and not require design and construction of water dams which must satisfy higher design standards and require use of large volumes of costly construction material. Presence of large water ponds upstream of conventional tailings embankments, especially in impoundments with poor operational controls, can represent a significant liability for operators, as the volume of flowable/unconsolidated tailings that could be mobilized in the event of a dam failure is increased. Additionally, production of dewatered tailings has been cited as a consideration to limit production of undesirable seepage of noxious chemicals into environments downstream of tailings facilities and as a mitigation measure to limit potential development of acid rock drainage.

Thickened and dewatered tailings are produced at various sites for a wide variety of reasons. To achieve desired objectives, these systems must be integrated into an overall operational and tailings management process flow that accounts for site-specific realities and constraints.

Moreover, the challenge of managing tailings streams with a significant (>50%) fines fraction is not unique to the Alberta oil sands and is one of the instances in which thickening technologies have been applied in the wider mining world. Fine tailings challenges exist in the management of Kimberlite, uranium, phosphate, and bauxite/alumina refinery tailings. As such, the case studies cited in this paper have been selected to provide examples not only of where these technologies have been implemented but to describe some of the key learning from each application.

Reviewed Case Studies

In 2012 the authors completed a comprehensive review of the use of thickened tailings technologies in the mining world outside the Alberta oil sands. As a result, the learning presented here come from case studies from the diamond, coal, uranium and alumina industries thought to be most instructive for continued application of similar technologies within the Alberta oil sands. The case studies represent sites where management of fine tailings are also a key factor in the decision to implement use of thickened tailings technologies. Table 1 (at end of paper) summarizes the key motivations and learning associated with each case study reviewed.

Summary of Key Lessons Learned

Successful implementation of thickened tailings technologies must account for site process, operational, logistic, topographic, environmental, climatic, hydrologic, and management realities. While the cited end products of thickening technologies present many advantages, achievement of these end products is not assured by mere selection or installation of thickening systems at a particular site. Moreover, within the same commodity, successful implementation of one mode of tailings dewatering at a specific site does not guarantee successful implementation of the same technology at a different site.

Case studies from the Kimberley and Ekati diamond mines describe management of Kimberlite tailings and indicate that a significant learning and implementation curve can exist as it relates to start up and continued production of a consistent tailings product. Ensuring consistent deposition and subsequent in-situ handling of deposited materials can also present significant challenges.

The Lone Mountain coal mine case study indicated that significant cost savings could be realized once fine tailings flocculation had been optimized. To achieve these results, significant testing and process refinement were required. Said differently, the definition of success did not end at the end of the spigot but ultimately accounted for the performance of deposited tailings and how their long-term behavior was affected by flocculation rates applied.

The example from the Cluff Lake uranium mine illustrated the challenges associated with mining operations in the Canadian north. However, the case study indicated that transition from conventional tailings storage to producing a thickened tailings product is entirely possible once the objectives of the technology shift are clearly understood by each component of the beneficiation, waste management, and water management teams. A clear understanding of the scope and implications associated with implementing the technology from the process engineers through to the tailings operators and reclamation planners were key components to overall project success.

The case study from the Jonquiere alumina refinery indicated that performance of deposited thickened tailings was enhanced through use of mudfarming technology and freeze thaw. Said differently, optimization of deposited tailings performance and geotechnical behavior may be achieved by methods that result in additional removal of water from the deposited tailings solids.

Corser & Strachan (2011) and Robinsky (1999) also indicate that in addition to improving storage density and tailings strength, use of thickened tailings can significantly enhance water recovery and increase final solids content. In hard rock mining the amount of water discharged by a thickened tailings storage facility is generally one-fifth to one-third of a conventional tailings system. Progressive reclamation is also possible at some operations. However, until site realities including process, tailings production rate, available real-estate for tailings deposition, and post deposition tailings and water management are fully accounted for in all stages of planning, implementation and process refinement, cited advantages of these technologies should be considered to be theoretical. Said differently, tailings management challenges and site realities exist in real time, at full scale in the field and not on the pages of white papers or lab journals.

REVIEW OF USE OF THICKENED TAILINGS TECHNOLOGIES IN OIL SANDS INDUSTRY

General Overview

The treatment and dewatering of oil sands tailings has been formally studied by numerous researchers since the 1970's. However, it should be noted that many of the challenges and observations associated with tailings produced through hot water extraction processes were noted by Dr. Karl Clark in the late 1920s during his experiments (Sheppard, 1989).

Treatment methods applied to oil sands tailings may generally be categorized as follows (BGC, 2010):

- Physical/mechanical treatments e.g. filtration and centrifugation;
- Natural processes e.g. sedimentation, self-weight consolidation and freeze-thaw;
- Chemical/biological treatments e.g. using thickeners or the in-line thickening methods;
- Mixtures/co-disposal methods e.g. consolidated tailings (CT), and mixing thickened tailings with sand; and
- Permanent sub-aqueous storage e.g. use of water caps above deposited MFT.

While other methods exist and are currently being investigated (see BGC, 2010 for additional info), this paper focuses on case studies where the previously indicated technologies were applied.

Early Research

Since the late 1970's numerous laboratory-scale research projects have been conducted looking at dewatering of oil sands tailings. One such study was the work published by Rao in 1980.

Rao (1980) studied flocculation of two different oil sands tailings slurries using four different flocculants after pre-treating the tailings with the optimum concentration of Mg^{2+} and Ca^{2+} ions (coagulants). Rao also used several methods to mix the flocculent with the tailings slurry. This process recovered 60-70% water from the tailings slurry with 20-25% water loss by evaporation. In the study, Rao's process required 125 grams of flocculent per 1000 liter of tailings and based on this laboratory experiment, he estimated that it would be possible to dewater 1000 litres of tailings slurry at a cost of roughly one dollar (1980 CDN dollars).

While lab-scale studies continue to the present time, the authors of this paper will focus on research and trials at bench and pilot scale given the significant scale at which trialed technologies must be able to deliver in order to be both effective and economic for FFT management throughout the mine life of the various oil sands operations where various forms of the hot water extraction process are used.

Conventional Thickening with Flocculant

In 2005, Syncrude performed a pilot thickener test (190 mm $\times$ 4.5 m thickener) on tailings from the Aurora mine. Flocculent dosage of 84 to 90 g/t was applied to slurries containing 12 % solids by weight and 49 to 52.5 fines by weight. The thickening process produced underflow with solids contents ranging between 58.5 and 60 percent solids by weight. The produced material also had an unsheared yield stress of 317-409 Pa and a remolded/double sheared yield stress of 100-115 Pa. The achieved yield stresses were considered advantageous for pumping underflow for subsequent discharge in a containment facility (Yuan and Lahaie, 2009).

To determine the consolidation rate and deposition slope angle that would be achieved a flume test was carried out on these thickened tailings which were pumped to the flume using a centrifugal pump. An initial deposition slope of 4.3% was achieved which reduced to 3.6% after 4.5 months. During the same time the yield stress of deposited material increased from an initial value of 120 Pa to 1910 Pa. However, the measured yield stress at the toe of the deposited was estimated to be only 300 Pa due to the accumulation of connate water at the toe (Yuan and Lahaie, 2009).

When the average density of the deposit reached 70 wt.% at the end of the flume test on March 1, 2006, it was subjected to natural desiccation at 20°C under laboratory conditions (no exposure to sunlight or wind). After 29 days under these conditions, the solids content of the deposit reached 80 wt% and up to 100 wt % at the toe of the deposit. The rapid consolidation of these flocculent treated tailings compared to composite tailings made using gypsum (CT) was believed to be attributed to the higher permeability of the flocculated tailings at the same void ratio. Although this study did not include a thin-lift desiccation test in the field, the desiccation rate was expected to be higher in the field as a result of exposure to wind and sunshine, and installation of lateral drains in the field plot. The material used in the flume test hat a liquid limit of 25.8%, a plastic limit of 16%, and a plasticity index of approximately 9.9% (Yuan and Lahaie, 2009).

Yuan and Lahaie (2009) also reported completion of the following field trials:

- 2001 - 2003: 10-m conventional high-rate thickener
- 2003 – 2004: 1.5-m deep cone paste thickener and a 4-m high compression thickener; and
- 2006-2008: 4-m high compression Aurora Thickener Prototype (ATP).

Each trial investigated important parameters of the thickening process such: sensitivity to feed diversity, challenges associated with cold weather operations, effect of residual bitumen in the thickener feed, rheology of thickened tailings, and, challenges associated with pumping. Based on these field trials, Yuan and Lahaie (2009) postulated that thickened (paste) tailings technology was implementable in the oil sands industry from the perspective of being able to develop a thickened product. However, they also reported completion of preliminary trials to assess the effects of pumping and piping thickened tailings.

Consolidated Tailings

Consolidated tailings research is one of the few thickened tailings research avenues that has made it to the level of industrial scale production. Mikula et al. (1998; 2008a) spent years developing the so-called “consolidated tailings (CT)” process at the Devon labs of Natural Resources Canada. CT are tailings in which sands (solids larger than 44 μm in diameter and the majority of cyclone underflow tailings) do not segregate from fines (solids smaller than 44 μm) as a result of introducing a chemical reagent (e.g. gypsum). Sand to fines ratio is kept at 5.5:1 for effective fines capture or to larger than 3:1 for rapid consolidation (Chalaturnyk et al., 2002; Powter, 2010). CT which is also referred to as composite tailings and as non-segregating tailings (NST) in the oil sands literature, created an alternative to wet closure landscapes by producing dry stackable tailings (Mikula et al., 1998; 2008a).

The CT process established by Mikula et al. (1989; 1996) was commercialized for the first time in 1994 by Suncor (Mikula et al., 2008a). The following benefits associated with implementation of this process were reported (Lawrence & Ali, 2010):

- Increase in recycled water return and decrease in fresh water consumption; and
- Increase in tailings storage capacity due to decrease in the volume of deposited (CT) tailings.

CT is still considered to be a primary tailings treatment method in the oil sands industry. In spite of the enumerated benefits, CT was considered a significant risk to Suncor’s long-term tailings management strategy due to limited availability of dyke construction material and the high costs associated with technology implementation. CT relies on tailings sands for fines capture and makes construction of dyke structures inevitable (Lawrence & Ali, 2010). This motivated Suncor to more towards their Tailings Reduction Operations (TRO) that are currently being piloted at an operational field scale (Suncor, 2009). Further discussion of TRO will be provided in the section describing use of in-line thickening technologies.

In addition to gypsum (used by Suncor), other process aids such as carbon dioxide, lime, acid and lime, and polymers are used in other CT processes. However, one of the primary challenges with CT is concentration of additives in the connate water which cannot be discharged and therefore continue to accumulate in tailings storage facilities (Mikula et al., 2008).

Thickening technology was also utilized in several large scale field pilot projects in a government-industry collaboration known as the “2001 thickened tailings field trial.”

Syncrude also piloted a CT process to produce a tailings product with 68-70 % solids by weight. The addition of coagulant(s) and the subsequent effect on the properties of the clay minerals contained in the tailings stream was a critical component of this process. Addition of organic and inorganic coagulants during Syncrude’s CT process and a description of the process is provided by MacKinnon et al. (2001). Addition of coagulants (e.g. lime, gypsum, CO_2, Alum, etc.) changes the nature of the particle size distribution of the tailings from a gap-graded to a more uniformly graded mixture. The gap-graded particle size distribution of the tailings causes the segregation of fines from the slurry while uniform particle size distribution significantly reduces or eliminates the segregation during discharge and deposition (Matthews et al., 2002).

Figure 1 shows how addition of different coagulants changes the segregation boundary for the CT product. In addition to how effective a coagulant is in making the tailings a non-segregating mixture, cost, influences on water chemistry, and depositional and geotechnical performance are critical parameters in selecting the best coagulant for the CT process.

An important challenge faced by Syncrude during implementation of the gypsum based CT process at the industrial scale was the ineffectiveness of recommended dosage based on laboratory experiments. In the 1995 Field Demonstration, Syncrude ended up using 1400 g/m^3 of gypsum instead of 900 which was recommended by laboratory experiments (Matthews et al., 2002).

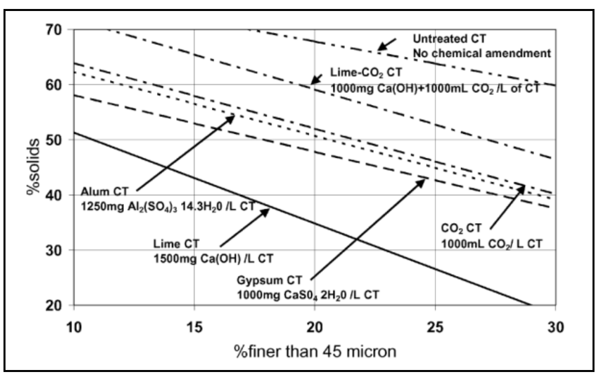

Figure 1. Effect of Coagulant Use on Segregation in Syncrude's 2001 Field Trial (after Matthews et al. 2002)

Based on these trials Syncrude considered their CT process, also known as the Aurora Thickener Pilot (ATP), to be commercially viable technology that would be implemented at Syncrude's Aurora mine site (Fair, 2008).

These trials demonstrated CT's ability to reduce tailings consolidation consequently reducing the size of tailings ponds (limiting land disturbance), construction costs, increasing the volume of water available for recycling, improving the quality of recycled water, and increasing the thermal efficiency of the extraction process as well as reducing greenhouse gas emission by using warm recycled water.

However, use of the technology impacted process on water quality and highlighted a need for continuing research into coagulant type and dosage rates. Research in this vein was completed by Apex Engineering Inc. (Chalaturnyk et al., 2002) in which calcium hydroxide was used as the flocculant and combined with carbon dioxide in the thickening process to mitigate ion accumulation in the recovered water.

Non-Segregating Tailings

Shell Canada used paste tailings technology as a step during large scale production of at the Muskeg River Mine tailings test facility. Cyclones, addition of coagulant and flocculent, and high density paste thickeners were used to produce a final tailings product with 75-78 % solids by weight.

Matthews and Masala (2009) claimed that Shell Canada's NST process maximized storage of sand, fines and clays; produced homogenous tailings with high fines capture efficiency; rapidly consolidated to produce trafficable surfaces; reduced the FFT inventory; and was more flexible and faster to reclaim.

To produce their version of NST, Canadian Natural Resources Limited (CNRL) used a CO_2 amended process in combination with cyclones and thickeners at the Horizon Project (Chu et al., 2008). Tailings were thickened prior to mixing and the pH adjusted to produce NST. Polymers were added in multiple stages of this thickening process. At the Horizon Mine, coarse tailings (underflow) are also segregated using hydrocyclones. These coarse tailings were subsequently blended with fine thickened tailings (slurries thickened in gravity thickeners) and pumped to the tailings storage area. CO_2 was added in the pipeline transporting NST to produce the following advantages (as described by Chu et al., 2008):

- Decrease required tailings pond capacity thereby reducing cost and reducing the environmental footprint;
- Recycling process water at relatively high temperatures thereby reducing use of natural gas for heating process water

(thereby reducing greenhouse gas emission);
- Use of bitumen recovered by the NST process; and
- Faster reclamation.

Similar to Syncrude's CT process, very specific conditions (e.g. fines concentration and sand to fines ratio) are required for the Horizon's NST process to produce non-segregating tailings. After testing hundreds of samples, the NST process has been uniquely designed for the Horizon Mine so that the majority of ore types at the mine have the suitable characteristics for this process.

In-line Thickened Tailings

In the oil sands industry, segregation of tailings is usually defined as a considerable change in fines content with depth in the deposit. In the early 1970's, both concepts, filtration and thickening, were ignored due to the high costs and instability of the slopes constructed from chemically treated tailings. However, this research resulted in important findings associated tailings segregation that would be precluded when sulphuric acid and/or lime was added (Caughil et al., 1993). This research also claimed that polymers with high molecular weights – commonly used as flocculants in thickening processes - do not significantly improve the self-weight consolidation behavior (e.g. consolidation rate, consolidation time etc.) of FFT. At the time, other considerations such as the effect of lime or acid addition to tailings and on the resultant release-water chemistry (e.g. metals and bicarbonates concentrations) were identified as requiring additional study prior to implementing any of these processes (Caughil et al., 1993).

Xu and Hamza (2003) successfully used both bench and pilot-scale thickeners (2 t/h) for thickening tailings samples collected from commercial oil sands extraction plants. They produced non-segregating paste-like tailings with solids contents of about 60% by weight. They also investigated use of 30 different polymers in the thickening process. Their results showed that high molecular-weight, medium-charged anionic polymers were most effective for thickening FFT. However, as noted earlier, applicability of bench and pilot-scale studies to the commercial oil sands operation scale is still being assessed.

Jeeravipoolvarn et al. (2008b) reported the successful use of in-line thickening at the pilot-scale. Their field and laboratory studies indicated that the in-line thickening process allowed tailings to compress from an initial void ratio of 65.6 (solids content of 3.7%) to 5.2 (solids content of 32.6%) in only 10 days. This sedimentation stage was then followed by consolidation of the tailings to 50% solids content within the top 1 m of the tailings pond in 4 months.

Suncor's TRO process also utilizes in-line thickening technology. The process involves mixing FFT with a polymer flocculent but does not involve use of a conventional thickener. Suncor's TRO has enabled this oil sands operator to mitigate the liability of increasing MFT inventories (Lawrence & Ali, 2010). However, it should be noted that this solution has been developed for Suncor's specific site realities and requires access to significant drying area on which polymer amended tailings are spread and exposed to additional drying methods.

Centrifugation

Mikula et al. (2008b; 2009a,b) showed that centrifuges can be used to produce a cake which can be dry stacked. This process reduces the required water for the oil sands operations by a factor of 2 (Mikula et al., 2009a). A thickener was used by Mikula et al. to prepare the centrifuge feed to improve efficiency of polymer mixing.

Mikula et al.'s research on use of centrifuges for dewatering MFT was tested at a pilot scale by Syncrude in 2008 (Fair, 2008; Mikula et al., 2009b). Tailings from this study were deposited with initial yield strength of about 2kPa and solids content of about 55% solids by weight. Although promising, several challenges such as reliability and performance of centrifuges, flocculant type, dosage and mixing optimization, and high operating costs (including energy costs and net greenhouse gas emissions) require careful additional review if this technology is to be successfully implemented at the operational scale (Mikula et al., 2009b).

EFFECTIVE APPLICATION OF THICKENED TAILINGS TECHNOLOGIES IN THE OIL SANDS

Industry-Specific Considerations

Key considerations associated with use of thickening technologies in the oil sands are associated with development of an overall water and waste management plan that accounts for site process and logistical realities. Tailings and water management solutions must fit within the overall mass and water balance for each site and do not exist or operate in isolation. Failure to integrate process success metrics and post-deposition tailings management activities will result in failure to meet ultimate objectives of net cost effective management, closure, and reclamation of FFT. A review of the numerous publications documenting the various uses of thickening technology suggest that identification of optimized solutions is an iterative process requiring a close feedback loop between field results and research findings. Moreover, the combined cost of polymer and the significant volume of FFT to be treated suggests that measures to identify means of optimizing flocculant dosage are critical.

Based on the experience of this paper's authors, polymer dose optimization should be assessed from the perspective of both short term (initial and near term water release, and quality) and longer term (overall strength development, true bearing capacity, and material performance) perspectives.

Lessons From the Wider Mining World

A review of case studies from within and outside of oil sands reveals the consistent experience that implementation of tailings thickening and dewatering technologies requires careful accounting for specific site realities. There truly is no silver-bullet in any tailings management scenario at any site. This means that successful tailings management on each site will require complete integration of the entire team charged with production and handling of tailings.

The case studies taken from outside oil sands industry are consistent with findings from tailings management practice throughout the wider mining world and reveal that where embankment construction materials are limited or dear, production of a consistent thickened tailings product may go a long way to realizing significant capital savings. Moreover, production of a thickened product that can be deposited once and does not require subsequent handling further optimizes tailings handling and management costs.

Once the containment facility is defined, deposition of a consistent thickened tailings product also enables better utilization of the facility's storage capacity. Furthermore, this technology can be applied in low sloping terrain but measures must be taken to ensure adequate preparation of the tailings storage facility and maintenance of proper drainage profiles as ponded water can adversely affect the material's physical properties and strength profile.

The case study review also indicated that while each operation faces specific challenges in implementing any new technology, key difficulties exist in producing a consistent tailings product. Within the context of the oil sands, it is the observation of this paper's authors that variability is the norm within a site and from site to site. Therefore, it is recommended that a tailings management plan be developed that is responsive to the variability that naturally exists. Management of a consistently variable product requires both accurate monitoring of real-time conditions and the ability to make adjustments as needed. Moreover, lessons and observations learned at field scale should be used to refine and enhance current process to ensure that overall, comprehensive success metrics are achieved by all involved operational and management departments.

Lessons learned encompass operation of pumping and piping systems (flow, control, measurement and deposition), effective FFT dosing, tailings deposition, and post-discharge tailings management.

CONCLUSIONS

Use of thickened tailings is neither a panacea nor a one size fits all solution in mining. Far from it, lessons from the wider mining world clearly indicate that the devil does indeed lie in the details. Furthermore, while experience with thickened tailings and other tailings dewatering exists within the oil sands, effective use of these technologies requires understanding of gel formation, secondary dewatering, and strength development. Existence of competing metrics for success within a single operational team often leads to disaster. Within the context of the oil sands this could be evident where acceptance criteria defined by the process team directly impacts the ability to develop ultimate material strength or result in achieving specific shear strength criteria while allowing true material performance (i.e. bearing capacity and net material handling costs) to be overlooked.

In the area of oil sands tailings management, significant lessons can be learned about process optimization from sites where their ability to operate is directly impacted by their ability to limit costs associated with tailings management while achieving operational and end land use/closure objectives.

While the scale of oil sands operations is large, each site represents a system of complex yet intrinsically related parts. As such, overall success must be influenced and defined through attainment of multiple objectives that are framed by a common vision – reduction of FFT inventories to produce closure landscapes that are truly chemically and physically stable and do not pose a threat to human or animal life or receiving environments.

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Table 1. Summary of Non-Oil Sands Case Studies

Mine	Climate/Precipitatio	Process	Time Line	Process details	Motivations to consider TT	Key Points/Learning	References
Ekati Diamond Mine *BHP Billiton Group* •Production (2009): 4,000 carats •Tailings production rate: 4400 tpd	•Humid subarctic •333 mm/yr	Conventional thickening	Conventional thickening in 1998	•Pumping Method: Centrifugal pumps •TSF Type: Surface/Sub-aerial disposal •Deposit Slope : 1%	Strict environmental regulations (Heightened scrutiny)	•Enhanced water recovery (90%) •Thickened tailings deposited below the lake surface •Piping challenges: -Maintaining operation of 8 km pipeline at 8◦C (in the cold winter) •Optimizing polymer types and dosage key to cost savings •Water management must still be accounted for: -Self regulating and continuous cone dewatering system required	•Oxenford and Lord (2006) •The Weather Network (2012) •InfoMine (2009) •Williams et al. (2008) •Rylatt (1999)
Lone Mountain Coal Mine *Arch Coal Inc.* •Processing rate (2011): 28,800 tpd •Production (2011): 2.4 Mtpa •Thickening rate (2009): 2,640 tpd	•Humid continental •1,342 mm/yr (1,295 rain)	Paste product	•Belt press (filter cake): 1997-2007 •Paste thickener: 2007-present	•Pumping Method: Gravity discharge and centrifugal pumps •TSF Type: DVD/Side hill •Beach slope: 2%	•Reduce the costs of the old technology (belt press) while maintaining regulatory limitations (paste characteristic of tailings): -Operating, maintenance, transportation, capital and chemical cost (saving ~2.5 $/t of refuse (> $1,000,000 operational annual saving) •Increasing the mine life (increasing the capacity of paste production)	Use of gravity for discharging the paste thickener placed on hill side near the deposition pond. •Components of cost reduction: -Optimization of the flocculent system -Optimization of the gravity discharge	•Gupta & Johnson (2007) •Sperling's Best Places (2010) •Arch Coal (2012) •Patil (2008) •Slottee & Johnson (2008)
Cluff Lake Uranium *Mine ESO Uranium Corp.* •Uranium Production: 1500 tpa (4 million lb U308) •Thickening rate: 877 tpd	•Continental, semi-arid •452 mm/yr	High density thickener	• Conventional tailings ponds: 1981-1995 •Conventional thickener: 1995-2002	•Pumping Method: PD pumps •TSF Type: DVD •Slope deposit: 3%	Extending the mine life for several years by increasing the storage efficiency	•Due to very high value of the mine products, only extending the mine life was enough to implement TT technology. •Good example of the successful retrofit application of TT to an existing conventional tailings operation. •Pump wear problems due to the angular nature of the residue •1.7 km tailings pipeline in extreme cold weather •Thickener exposed to the severe Canadian winter environment •No recycling of water at all	•Lord (2003) •Oxenford and Lord (2006) •Wagner et al. (2006) •El Dorado Weather (2012) •Williams et al. (2008) •Schnell & Corpus (2000)
Jonquiere (Vaudreuil) Alumina Refinery *Rio Tinto (Alcan)* •Production (2006): 1.4 Mtpa •Thickening rate (2008): 1,233 tpd	•Humid continental •997mm/yr (683 rain)	Deep Cone thickener	•Tailings ponds: 1942-1987 •Mud stacking (thickener): 1987-present	•Pumping Method: 3-stage PD pumps •TSF Type: Central thickened discharge (CTD) •Slope deposit: 3-4%	Energy & chemical cost saving Increase storage efficiency in the tailings management area	Highly consolidated tailings (75%) achieved by combining: -TT -"Mud farming" (mechanical rework of the dried mud) and; -Freeze-thaw	•Fourie & Jewell (2006) •Oxenford & Lord (2006) •The Weather Network (2012) •Williams et al. (2008) •Munro & Smirk (2012) •Smirk & Jackson (2010)

Session 7

Tailings Dewatering Technologies

A NOVEL METHOD TO IMPROVE TAILINGS DEWATERING

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ABSTRACT

There are three challenges facing scientists and engineers in mature fine tailings (MFT) treatment: fast solids/water separation, maximum water recovery, and consolidation of the final sediment. To date, polymeric flocculants have been widely used in the oil sands industry, alone or in combination with electrolyte coagulants, to the described challenges.

As a mixture of solids and liquid, tailings naturally stratify during settling according to Stokes' Law, with coarse particles concentrated at the bottom of a sediment bed and fines at the top. According to Stokes' Law, MFT viscosity is one of several factors affecting the particle settling velocity. In this paper, a new method using surfactant to reduce MFT viscosity, therefore, to accelerate tailings settling velocity and to improve tailings dewatering, will be presented. Experimental results indicate that with addition of a solid surfactant at 4,000 ppm and moderate shaking for 30 minutes, tailings viscosity was reduced from 6,300 cPs to 2,200 cPs at shear rate of 2.2 sec^{-1}. The treated MFT sample was centrifuged at 1,000 G-force for 5 minutes. After that, the top layer can be easily poured out of the centrifuge tube. The solid content in the top layer is only 6.9%, but is as high as 80.2% in the bottom layer. In this way, the MFT was readily separated without addition of extra water.

INTRODUCTION

Effects of different factors on settling velocity

The ultimate goal of our tailings project is to accelerate particles settling rate and to improve separation of fine particles from the tailings mud. To achieve this goal, several different approaches have been utilized. One of the approaches is to use various coagulants and flocculants to increasing particle size, and therefore to increase particle settling velocity.[1] According to Stokes' Law, there are several factors affecting the velocity, e.g. particle size, fluid density and viscosity. Thus, one of the new approaches is to reduce the viscosity of the tailings and therefore, to increase the particle settling rate.

The effect of different factors on a particle's settling velocity can be quantitatively analyzed and compared by use of Stokes' Law below:

$$v_s = \frac{gd^2(\rho' - \rho)}{18\mu} \tag{1}$$

where, v_s: settling velocity; d: diameter of particle; μ: viscosity of liquid phase; ρ': density of solid particles; ρ: density of the liquid phase; g: acceleration of gravity, 9.8 m/sec^2.

Particle settling velocity under different conditions was calculated based on Stokes' Law equation (1) and listed in Table 1 below.

Table 1. Theoretical Settling Velocity under Different Conditions Calculated by Use of Stokes' Law Equation.

Case	d (μm)	μ (cP)	ρ' (g/cm^3)	ρ (g/cm^3)	v_s (cm/hour)
#1	50	1904	2.6	1.02	0.407
#2	100	1904	2.2	1.02	1.215
#3	50	1048	2.6	1.01	0.743
#4	50	792	2.6	1.01	0.984
#5	50	121	2.6	1.00	6.479
#6	100	121	2.2	1.00	19.438

In the calculations above, the density of the liquid phase is 1.02 g/cm^3 and the density of the untreated solid particles is 2.6 g/cm^3. If a number of particles with the density agglomerate by flocculation process, the overall size may increase by a factor of 2, and the volume will increase by a factor of 2^3=8. The overall mass, however, will not increase eightfold because the aggregated particles will have a loose structure. Therefore, the density is assumed to decrease from 2.6 to 2.2 g/cm^3 after the flocculation.

In current technologies for tailings dewatering, coagulants and flocculants are dissolved in process water and then added to tailings, which has the side-effect of diluting on the tailings.[2] To evaluate the effect of dilution on the tailings' viscosity, the viscosities of the tailings diluted with process water at different ratios were measured. It has been found such dilution has a significant effect on the tailings viscosity. By comparing Case #1 and Case #5 listed in Table 1, if the viscosity has been reduced from its original value of 1904 cPs to 121 cPs while simultaneously not increasing the particle size, the theoretical settling velocity can be increased from 0.407 cm/hour to 6.479 cm/hour – a remarkable increase.

In practice, some surfactants have been used as viscosity reducers and are called "thinners" in contrast to "thickeners" that aggregate solids. The thinning mechanism is complicated and depends on the nature of the system. For example, for emulsion systems of two immiscible liquids, reduction of the viscosity may be caused by demulsification or reversion of W/O emulsion to O/W one because the latter usually has a lower viscosity and higher conductivity;[3] for a dispersion system of a solid and a liquid, reduction of the viscosity may be caused by wettability alteration of the solid surface from oil-wet to water-wet. For a waterborne system, once the wettability has been changed to a more water-wet condition, it is favorable for a faster sedimentation of the solids and a lower viscosity of the liquid phase. [4]

EXPERIMENTAL

1. Dilution of Original MFT with Process Water:

To evaluate the effect of dilution on tailings viscosity, the original MFT and the process water were mixed at different ratios to obtain different solids concentrations. In the original MFT, the solids content was 38.7%. The viscosity of each diluted tailings slurry was measured in order to find out the relation between the solids content and rheological properties.

2. Calibration of the Viscometer:

To get accurate viscosity results, a #25 spindle was purchased from the Brookfield Engineering. The Brookfield DV-III viscometer was calibrated by use of two standard fluids, (1) a viscosity standard of 602 cPs at 25 °C; (2) a viscosity standard of 1999 cPs at 25 °C. Brookfield spindles #25 and #31 were used for the calibration. Results indicated that calibration with spindle #31 has excellent agreement with the standard values; but spindle #25 had about 15% experimental error.

3. Surfactant Effect on Tailings Viscosity:

Selection of surfactants: The purpose of this experiment was to screen surfactant candidates that can reduce tailings viscosity without the addition of water. Criteria for selection of surfactants included 1) good solubility in water (HLB>10); 2) high purity (>95%); 3) low or moderate toxicity and readily bio-degradable. The experimental procedure was as follows:

a) 0.1000 g of surfactant was added to 24.90 g of original tailings and the total mass with bottle was measured for each sample.
b) The bottles were placed on a reciprocal shaker to mix the surfactant and MFT for two hours at low shear and no foam as shown in Figure 1.
c) The total mass for each bottle was re-weighed to check if any leakage or evaporation occurred. 5 blank tests were conducted at the same time for comparison.
d) Sample viscosity at different shear rates was measured using a Brookfield viscometer DV-III at 25 °C.

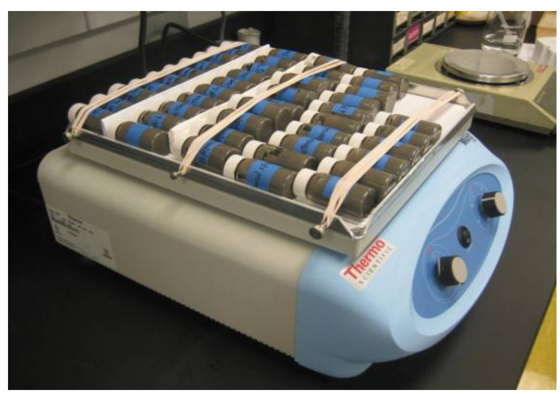

Figure 1. Mixing surfactant and tailings by a reciprocal shaker to minimize shearing effects on tailing samples.

4. MFT Dewatering with Assistance of Surfactant:

a) 199.20 g of tailings was added to a glass bottle, followed by 0.800 g of a solid surfactant. The bottle was placed on a reciprocal shaker to mix surfactant and MFT for 30 min, as shown in

Figure 2. For comparison, a blank test (containing no surfactant) was conducted at the same time.

b) After 30 minutes, 30.0 g of the tailings were weighed into centrifuge tubes for centrifuged at 1000 G for 5 minutes, as shown in Figure 3.

c) The upper liquid phase was decanted, and the lower solid sample was placed in an oven at 90 °C to measure the dry solids percentage.

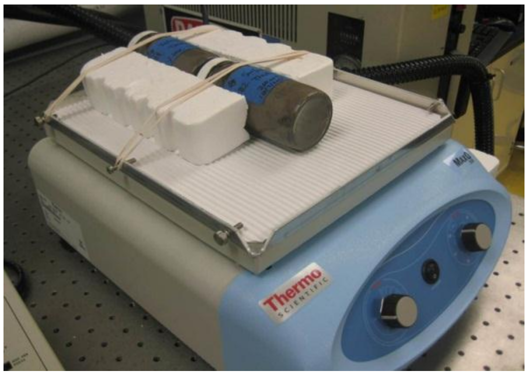

Figure 2. Tailings sample treated with surfactant and shaken on a reciprocal shaker.

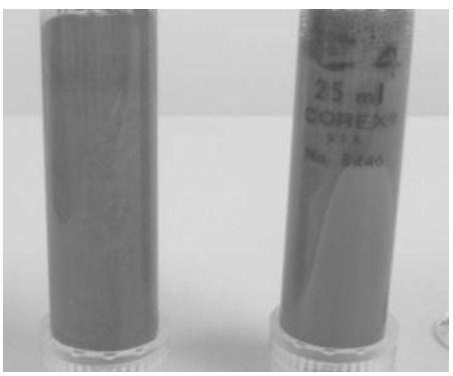

Figure 3. Tailings samples were centrifuged at 1,000 G-force for 5 minutes. On the left, sample without treatment shows no separation; on the right, tailings shows solids separated on the bottom.

5. Contact Angle Measurement:

a) The original tailings were treated with different surfactants and then diluted to solids concentrations of 10.0 wt%, 1.0 wt% and 0.1 wt%; A clean glass square was immersed into the diluted tailings to form a thin coating of tailings particles as shown in Figure 4.

b) The coated glass squares were placed in an oven to dry at 100 °C for 2 hours.

c) At room temperature, the contact angle of distilled water on the glass squares was measured using a ramé-hart Advanced Goniometer 500-F4 as shown in Figure 5.

d) The contact angle for each water droplet was measured every 15 seconds for 10 minutes to reach equilibrium. Because of heterogeneity and the roughness of the coated surface, each sample needed to be measured 5 times at different positions on the glass surface and the average value was used as shown in Figure 6.

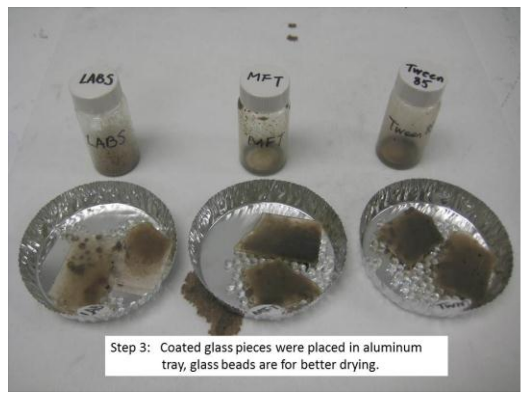

Figure 4. One clean glass square was coatedwith tailings treated with surfactants.

Figure 5. Contact angle of distilled water on the coated glass squares was measured by an advanced goniometer.

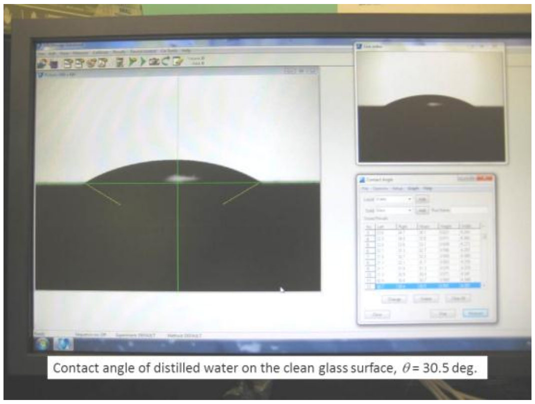

Figure 6. Contact angle for each water droplet was measure every 15 seconds for 10 minutes until an equilibrium was reached.

RESULTS AND DISCUSSION

1. Calibration of Brookfield Viscometer

The Brookfield viscometer was calibrated first with two standard fluids and two spindles, #25 and #31, at 25.0 °C. The calibration curves are shown in Figures 7 and 8. From the curves, it can be found that the viscosity data obtained by spindle #31 shows a high consistency with the standard values, particularly when the torque is between 60% and 70%. For the #25 spindle, however, calibration data were higher than the standard value at all RPM and torque percentages. The average experimental error was about 15%. Because the standard fluids are Newtonian fluids, their viscosity is constant at different RPM and does not change with the shear rate at a given temperature.

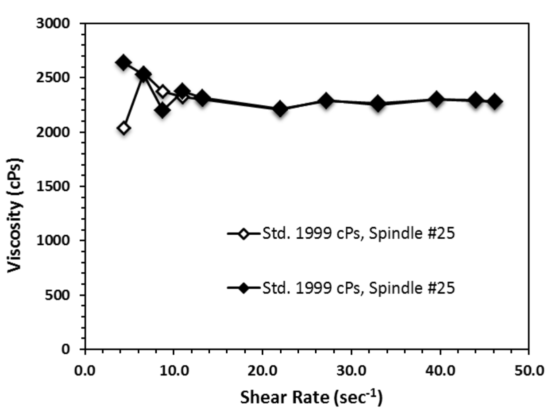

Figure 7. BROOKFIELD viscometer calibration curve for spindle #31.

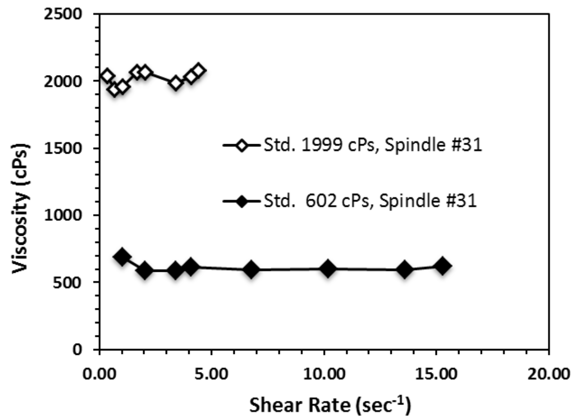

Figure 8. BROOKFIELD viscometer calibration curve for spindle #25.

2. Dilution of Original MFT with Process Water:

As described previously, the original tailings sample was diluted with the process water at different ratios. After dilution, the viscosity of each diluted tailings sample was measured and viscosity curves of the samples at various shear rates are shown in Figure 9. From the curves, it can be found that tailings viscosity changes dramatically with the RPM and shear rate, indicating the tailings slurries are shear thinning non-Newtonian fluids.

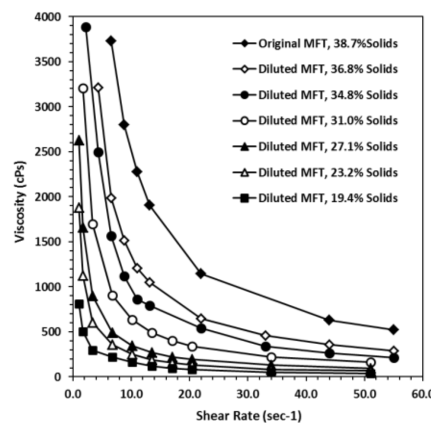

Figure 9. Viscosity of original tailings and diluted tailings at different shear rate.

Because two different spindles were used in the measurement, they give different shear rate even at the same RPM. To find out the effect of dilution on the tailings viscosity, the viscosity results need to be compared at the same shear rate (sec^{-1}) for different samples. Viscosity data at the shear rate of 13.6 sec^{-1} are experimentally available for different samples, so viscosities at this shear rate for different samples are listed in Table 2 and the relationship between viscosity and solids contents is plotted in Figure 10.

Table 2. Viscosity Results of Original Tailings and Diluted Tailings at Different Dilution Ratio (measured at 25 °C and a shear rate of 13.6 sec^{-1}).

Original Tailings	Process Water	Dilute Ratio	Solids (wt.%)	η (cps)
50.0 g	0.0 g	n/a	38.7%	1904
47.5 g	2.5 g	19:1	36.8%	1048
45.0 g	5.0 g	9:1	34.8%	792
40.0 g	10.0 g	4:1	31.0%	490
35.0 g	15.0 g	7:3	27.1%	268
30.0 g	20.0 g	3:2	23.2%	189
25.0 g	25.0 g	1:1	19.4%	121

From Table 2 and Figure 10, one can find that viscosity of the original tailings at 25 °C is 1904 cPs. After dilution with a small amount of process water, e.g. 2.5 g of the water was added to 47.5 g of the tailings, the solids decreases slightly to 36.8% from the original 38.7%, but the viscosity decreases by a dramatic 45% from its original 1904 cPs to 1048 cPs. When the tailings were diluted with more process water, e.g. in a 1:1 mass ratio, its viscosity decreased to 121 cPs, which was only about 6% of the original viscosity.

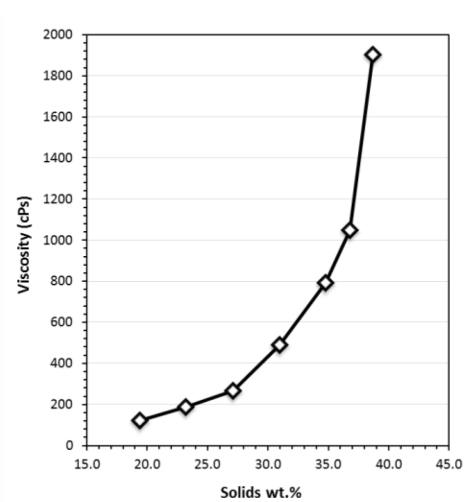

Figure 10. Viscosity of original tailings and diluted tailings at 25 °C and shear rate of 13.6 sec^{-1}.

Therefore, it can be concluded that dilution with process water shows significant effect on tailings viscosity. Viscosity reduction is an effective method to accelerate particle settling rate and improve tailings dewatering process.

3. Effect of Surfactant on Tailings Viscosity

In order to investigate surfactant effect on tailings viscosity, 19 non-ionic and anionic surfactants were selected for testing in MFT. The test results are shown in Figures 11-14.

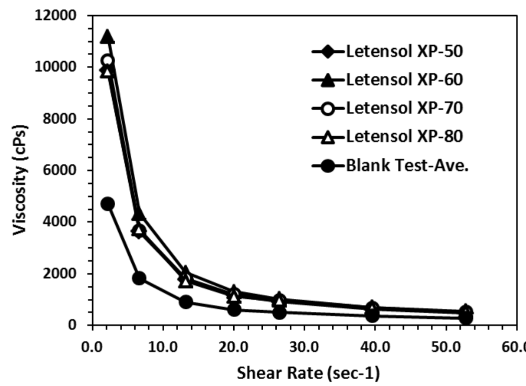

Figure 11. Effect of Letensol® surfactants on tailings viscosity.

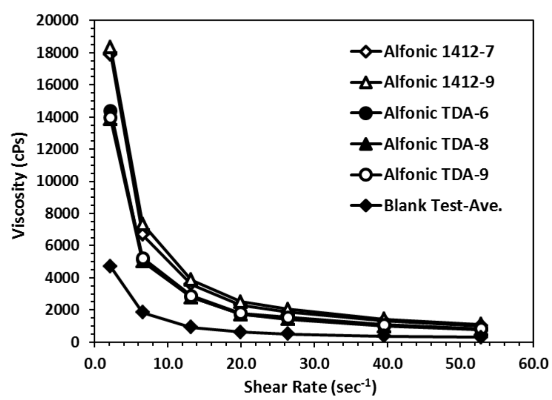

Figure 12. Effect of Alfonic® surfactants on tailings viscosity.

From Figures 11-14, one can see that surfactants Lutensol®, Alfonic®, Genapol® and Plurafac® do not reduce tailings viscosity, but instead increase the viscosity substantially. In Figure 14, it can be found that Bio-Soft® 101, Bio-Soft® 126 surfactants (Stepan) and Natsurf ® 265 surfactant (Croda) can reduce the original tailings viscosity from 4750 cPs to 3300 cPs at shear rate of 2.2 sec^{-1}, and from 1000 cPs to 600 cPs at a shear rate of 13.2 sec^{-1}. Bio-Soft® 101 and 126 (Stepan) are linear alkylbenzene sulfonic acids, while Natsurf® 265 (Croda) is an ethoxylated natural primary alcohol with 8 EO groups. It is therefore concluded that linear alkylbenzene sulfonic acids may be more effective to reduce viscosity for original tailings.

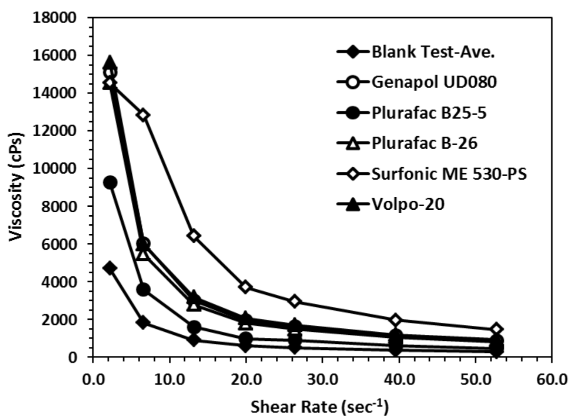

Figure 13. Effect of some surfactants on tailings viscosity.

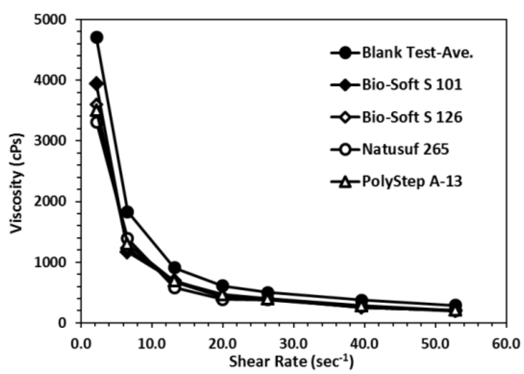

Figure 14. Effect of Bio-Soft surfactants on tailings viscosity.

4. Dewatering with Assistance of Surfactant

As described in the previous part of this paper, the original tailings were treated with a surfactant at 0.4 wt.% (4000 ppm) concentration. After being shaken for 30 minutes, 30.0 grams of the treated tailings were put into a centrifuge tube and centrifuged at 1000 G-force for 5 minutes. After the separation, 17.4 grams of the liquid phase was poured into a pre-weighed aluminum tray. The rest of the solid phase in the bottom layer was 12.6 grams. Both the liquid and solid phases were dried for 2 hours in an oven at 110 °C to measure the dry solid contents in each phase. For comparison, a blank test of the tailings without any surfactant treatment was also conducted using the same procedures at the same time. The experimental results are listed in Table 3.

The original tailings sample used in this experiment contains 38.3 wt% solids, which is much lower than solids content of 50 wt% required by ERCB Directive 074. In Table 3, it can be found that by surfactant treatment the solids content in the bottom phase was increased to 80.2 wt% after a brief centrifuge separation at 1000 G-force for 5 minutes; meanwhile, the solids content in the liquid phase (supernatant layer) was reduced to 6.9%, which will make it much easier to recover water using common flocculant treatment methods.

By comparison, solids content in the solid phase of the blank was only increased slightly to 42.4%, which is still below the solids content required by Directive 074. Meanwhile, the solids content in the blank's liquid phase was 34.8%, which is still close to the solids content of the original tailings.

5. Measurement of Water Contact Angle

Tailings consist of a mixture of water, clay, sand and residual hydrocarbons produced during the bitumen extraction process. The bitumen exists in a tight emulsion of oil and water that is stabilized by solid particles, so the wettability of the solid particle surfaces may have an impact on the emulsion system and tailings viscosity.[5]

In previous studies it has been reported that the residual hydrocarbons and insoluble organic solvents are associated with the particulate minerals and most of this insoluble organic compounds are strongly associated with the particles with sizes less than 45 μm. Such association between the organic residues and clay particles can change the surfaces from originally water-wet to strongly oil-wet. [6]

Surfactants are able to remove the organic residues adsorbed on a particle surface and change the wettability to less oil-wet or back to its original water-wet condition. [7] To evaluate the performance of surfactants in the wettability alteration and to investigate the relationship between the particle's wettability and tailings viscosity, contact angles of distilled water on glass squares coated with original tailings and the tailings treated with different surfactants were measured.

In this experiment, two surfactants were selected for contact angle study. One was LAS, a linear alkylbenzene sulfonate, which reduces tailings viscosity, while the other was Tween® 85, a polyoxyethylene sorbitan tri-oleate, which increases tailings viscosity. The results are graphically shown in Figure 15.

Table 3. Separation Results of Tailings Sample with and without Treatment of Surfactant.

Test Procedures	Tailings Treated with Surfactant	Blank Test
Original tailings (g)	29.88	30.00
Surfactant Added (g)	0.12	0.00
Centrifuged at 1000 G-force for 5 minutes	Yes	Yes
Liquid Phase (Supernatant, g)	17.4	17.5
Solid Phase (Settled Bed, g)	12.6	12.5
Dry Solids in Liquid Phase (g)	1.2 (6.9%)	6.1 (34.8%)
Dry Solids in Solid Phase (g)	10.1 (80.2%)	5.3 (42.4%)
Total Dry Solids (g)	11.3 (37.8%)	11.4 (38.0%)

In the figure, one can see contact angle of distilled water on the glass square cleaned by methanol is between 26.8° and 32.4°. In general, a water droplet contact angle of less than 30° is considered to be an indication that a solid surface is water-wet,[8] so the glass surface can be considered water-wet. Water contact angles on the glass coated with original tailings are between 34.4° and 47.4°, which is higher than that on the clean glass surface and indicates that the tailings particle surfaces are oil-wet. Water contact angles on the glass coated by tailings treated with LAS are between 27.9° and 36.5°, which is an indication of a less oil-wet or more water-wet condition caused by the surfactant treatment. This means that LAS can make the tailings surfaces more hydrophilic. The reason for this wettability alteration may be the removal or desorption of organic residues from the particle surfaces by the linear alkylbenzene sulfonate.

Water contact angle on the glass surface coated by tailings treated with Tween 85® is between 36.9° and 48.7°, which is even higher than that of glass coated with original tailings. This means this surfactant can increase the tailings' hydrophobicity. The reason for the wettability alteration to more oil-wet or less water-wet may be the adsorption of polyoxyethylene sorbitan tri-oleate on the particle surface. The three oleate chains could make the particle surface wettability even more oil-wet.

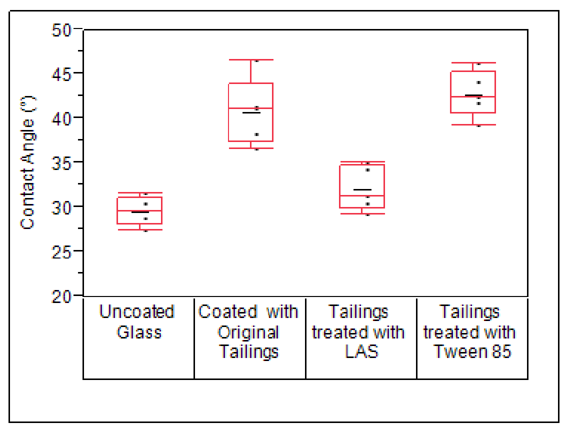

Figure 15. Contact angle of water on the glass coated with tailings treated by different surfactants.

In Figure 15, one can see that the standard deviation of contact angle measurements is relatively high, particularly for measurements on the coated glass. This is because of the surface roughness caused by the tailings coating on the glass slides. Therefore, the technique and procedures for coating glass surface by tailings need to be further modified and improved for better reproducibility and higher accuracy.

CONCLUSIONS & RECOMMENDATION

1) Dilution of original tailings with process water has a significant effect on tailings viscosity, even at low mass ratio like 1:19 or 5 wt.%.

2) Tailings viscosity is one of the most important factors affecting settling velocity of fine particles. Lower viscosity will results in a higher settling velocity.

3) Some surfactants can reduce tailings viscosity without addition of a single drop of extra water. However, some surfactants will increase tailings viscosity, which make tailings dewatering more difficult.

4) With treatment of proper surfactants at 0.4 wt.%, tailings dewatering can be improved substantially. After the treated tailings was centrifuged at 1000 G-force for 5 minutes, solid content in the solid phase (bottom layer) can be increased to 80.2% and the solid content in liquid phase can be decreased to 7%, which can make subsequent water recovery processes much more effective.

5) Using surfactants that can reduce tailings viscosity to accelerate particle settling velocity is a novel technology to improve tailings dewatering and maximize water recovery.

6) In addition, this new technology is also helpful recover hydrocarbon residual in the tailings.

ACKNOWLEDGEMENTS

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THE MITD PROCESS FOR DEALING WITH MFT

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ABSTRACT

The paper discusses a technique for dealing with the MFT (Mature Fine Tailings) problem from the Canadian Oil Sands. It describes a process called MITD (MFT to Immediately Trafficable Deposits). This process uses direct drying to convert both existing and future MFT into immediately trafficable deposits which can be used in land reclamation. While there may be multiple technologies that seem to have the capability to produce trafficable deposits, the other technologies currently offered require further treatment of the processed material. This additional treatment generally involves drying in open pits for an extended period of time. When considering the cost of any of these processes, it should be recognized that anything that is done to eliminate the MFT problem will add cost to the production of oil from the Canadian Oil Sands.

The Bepex MITD process provides significant advantages and is estimated to fall between the processing costs of the leading technologies which are currently being tested on a relatively small commercial scale. This new process provides a guaranteed means to provide immediately trafficable deposits, utilizes a minimal amount of land, and increases the amount of water that is recycled thereby reducing the water requirements from the Athabasca River. While thermal processing will result in additional energy consumption, the net energy consumption can be reduced significantly by heat recovery in the system. The MITD Process provides the following advantages compared to any current process in the advanced stage of development:

- The MFT deposits are immediately trafficable allowing oil sands operators to meet objectives set forth in the ERCB Directive 074
- With the addition of a simple recycle system, it is believed that a significant portion of the VOCs and Naphthenic acids will be destroyed in the process resulting in less air pollution.
- The amount of water available for recycle will be increased which will significantly reduce the amount of water that will be pulled from the Athabasca River.
- The MITD process will be less subject to atmospheric conditions such as freezing weather and heavy rainfall.

This paper describes the MITD direct drying process and includes an economic comparison against two of the leading competing processes. The economic study is based on factors that were correct at the time the study was developed and the author's knowledge of the different processes.

INTRODUCTION

While this paper does not propose to present a detailed expert analysis of the MFT problems and solutions or a complete historical context of the problem, some basic understanding of the situation is necessary to appreciate the value of the proposed MITD process. The Canadian Oil Sands provide a huge source of oil for North America. However, the oil must be released from the clay/sand deposits where it resides. Mining techniques are used to recover the rock and transport it to the process site where the Clark hot water process is used to recover the viscous oil which is referred to as bitumen. A schematic diagram of the hot water extraction process is shown in Figure 1 below.

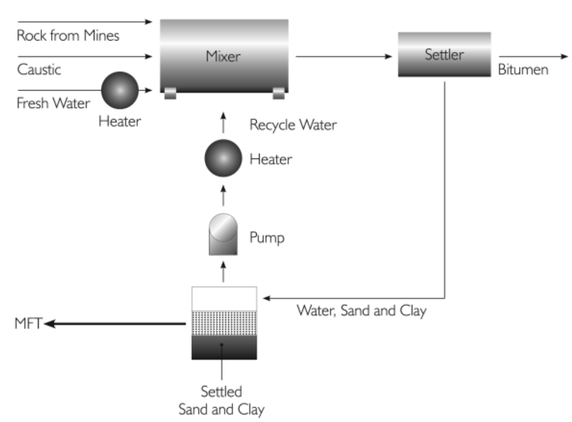


Figure 1. Hot Water Extraction.

As shown in this figure, the products and by-products from this process consist of bitumen, water that is recycled, clay/sand deposits that are allowed to accumulate in the settling pond and a mixture of water and clay that settles out in a third layer located in between the clay/sand and the water layers. This interfacial material will not settle into water and clay components except over such a long period of time as to make unaided gravity settling impractical. This interfacial material consisting of 30 weight % clay is pumped to an accumulation pond. Since there is no pragmatic disposition of this material, it has built up over the years and will continue to build up. A recent source estimated the amount of material at 750 million cubic meters. While this description of the Clark hot water process is highly simplified, it illustrates the complex nature of the MFT mitigation problem.

In addition to the process issues, unlike many capital expenditures, mitigation of the MFT problem has only minimal savings associated with a remediation project. Because of this and the relatively low cost of stockpiling the MFT, projects have been delayed in hopes that a low cost mitigation technology could be developed. As of this date no low cost technologies have been developed that seem applicable on a commercial scale. Thus any project to eliminate the MFT will increase the cost of producing oil from the Alberta oil sands.

The Alberta Energy Resources Conservation Board issued Directive 074 in February of 2009. This directive specifies both the timing of converting the MFT into reclaimed land and the final specification of the reclaimed land. They do not specify what process must be used, but specify that the final product must meet the following specifications:

- Within one year of creation of a dedicated MFT disposal area, the undrained shear strength must exceed 5 kilopascals.
- It must rise to an undrained shear strength that exceeds 10 kilopascals and be trafficable and ready for reclamation after 5 years.

The directive also specifies the rate that the existing reservoirs must be converted to the reclaimed land. However, this timing is such that it will not be possible to achieve.

There are multiple techniques that have been proposed to allow achieving these target shear strengths in the specified time period. This paper focuses on 3 of these technologies (Thin Film Drying, the Centrifuge/Mixing process and the MITD process).The authors of this paper do not claim expertise in Thin Film Drying or the Centrifuge/Mixing process. Most of their knowledge has come from reviews of the literature and attendance at the 2010 IOSTC. These three techniques involve the removal of water to produce trafficable deposits. Laboratory results based on Kaolin Clay have shown that if the percent solids in the MFT is increased to a concentration of 70 to 75 wt. %, the shear strength will exceed 10 kilopascals. These laboratory tests are summarized in Figure 2.

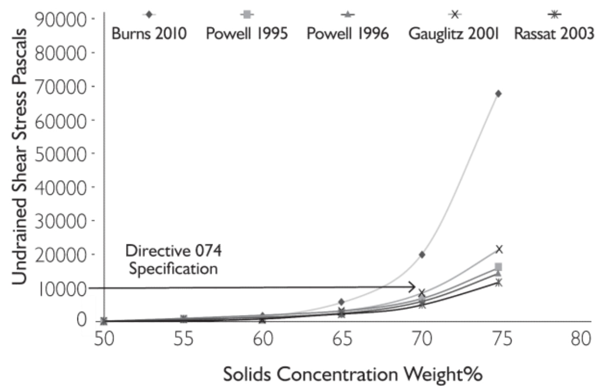


Figure 2. Undrained Shear Stress Pascals vs Kaolin Clay Solids Content Weight %.

All of the potentially commercial processes involve at least two steps. However, only the MITD direct drying process produces a trafficable deposit (undrained shear strength greater than 10 kilopascals) immediately. The other processes require a significant holding period before the deposits become trafficable in addition to requiring large areas of land.

While thermal processing has been considered in the past, it has not received detailed consideration because of economic and Green House Gas considerations. This paper presents an assessment of both of these aspects by comparing the two leading technologies and the emerging MITD technology.

PROCESS DESCRIPTIONS

A description of each of these processes is presented in the following paragraphs. As indicated earlier, the Thin Film Drying and Centrifuge/Mixing processes are based on literature and conference knowledge only. However, it is believed that this knowledge is adequate for a comparison of the three technologies.

Thin Film Drying – In this operation, the MFT are contacted with a flocculating agent prior to being spread out in a thin layer (10 to 50 cm thick) in a sloped pit. The flocculating agent causes the very small fines to agglomerate which reduces the surface area and frees the water to drain into the bottom of the pit. In addition to the water that drains into the pit, water is also evaporated. Approximately 50% of the water removal occurs by evaporation from the pit. A schematic flow diagram of this process is shown in Figure 3, below.

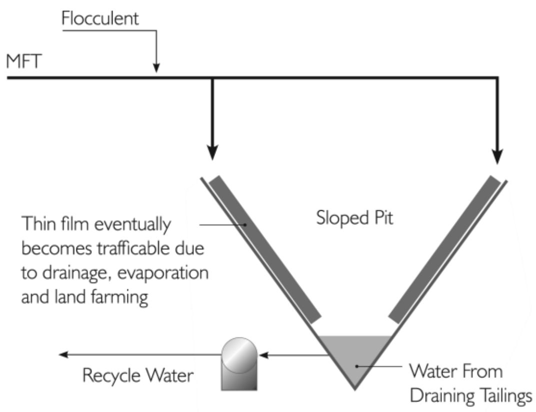


Figure 3. Thin Film Drying.

The negative aspects of this process are:

- The operation can only be conducted in non-freezing weather. Even in non-freezing weather, the rate of evaporation is a strong function of the atmospheric conditions. During the summer months, the average relative humidity is between 50 and 60%. This is relatively high for an atmospheric drying application. In addition, potential drying rates in June are typically 3 to 4 times those in October.
- The drained material in the open pits may adsorb water from any rain.
- The surface of the thin film dries by both drainage and evaporation faster than the subsurface layers. This causes the top layer to become a barrier to future evaporation and for drainage from subsequent depositions. Thus while the surface of the initial layer is very dry, other levels and subsequent depositions are not dried adequately. To compensate for this uneven drying, land farming techniques are used to provide fresh MFT surfaces. These land farming techniques require many pieces of heavy equipment and are very labor intensive. The economics assume that 40 people are required year around and an additional 200 people are required during the 9 month land farming season.
- The water that is evaporated into the atmosphere is not available for recycle.
- Any Naphthenic acids or other volatile organic materials (VOCs) present in the MFT are at least partially evaporated into the atmosphere.
- The cost of flocculent at the target treat ratio is very high ($2/Barrel of Bitumen).

Centrifuge/Mixing Process – In this operation the MFT are fed to a centrifuge. The cake from the centrifuge will be approximately 55 to 60 wt% solids. At this concentration, the undrained shear strength will be approximately 0.5 to 1.0 kilopascals. In order to provide the additional strength, two additional steps are required. The centrifuged MFT are compounded with coarse sand at a ratio of about 5lbs of coarse sand/lb of solid MFT and the material is allowed to compact and further dewater in a pit. A schematic of this process is shown in Figure 4. The disadvantages of the pit operations are similar to those of the Thin Film Drying Process described earlier. Flocculent is still required to obtain the high solids concentration from the centrifuge.

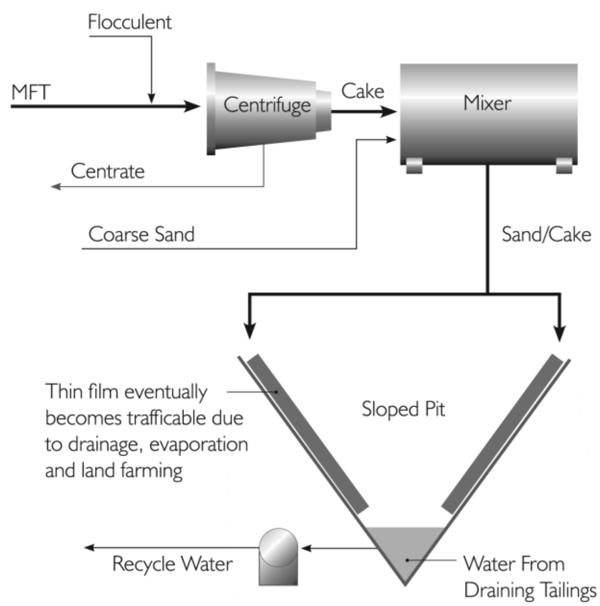
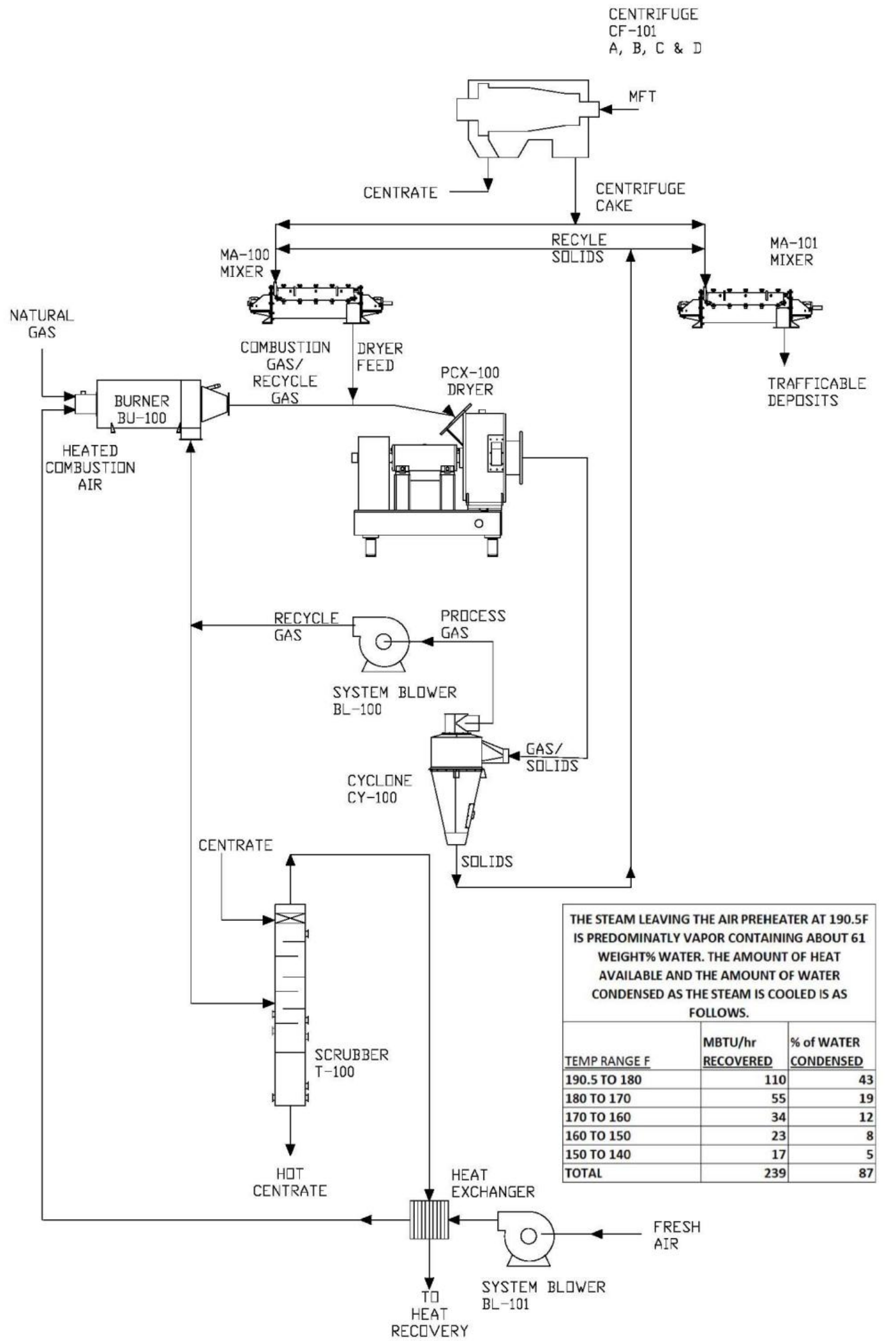


Figure 4. Centrifuge Process.



THE STEAM LEAVING THE AIR PREHEATER AT 190.5F IS PREDOMINATLY VAPOR CONTAINING ABOUT 61 WEIGHT% WATER. THE AMOUNT OF HEAT AVAILABLE AND THE AMOUNT OF WATER CONDENSED AS THE STEAM IS COOLED IS AS FOLLOWS.

TEMP RANGE F	MBTU/hr RECOVERED	% of WATER CONDENSED
190.5 TO 180	110	43
180 TO 170	55	19
170 TO 160	34	12
160 TO 150	23	8
150 TO 140	17	5
TOTAL	239	87

Figure 5. The BEPEX MITD Direct Drying Process.

As with the centrifuge process, the MFT are mechanically dewatered to 55-60% solids. Although flocculent is required in this process as well, the required level is less when compared to the other processes. The dewatered MFT are treated thermally in a direct drying system. The system is controlled to remove the water to the optimum moisture level for both product characteristics and heat input. The process economics are based on an energy balance that assumes 80wt% solids in the final product. As shown in Figure 2, this is conservative since all of the correlations predict a shear stress above 10 kilopascals at this solids concentration. The product can be controlled to any level between a wet cake and a dry powder.

While thermal processing will result in additional energy consumption, the vapor from the system is close to saturation which facilitates the collection of water for recycle and heat recovery. This heat recovery significantly reduces the overall energy consumption of the MITD process.

The Bepex MITD process provides the following advantages compared to any current process in the advanced stage of development:

- The MFT deposits are immediately trafficable. The 80 wt % solids will have an undrained shear stress in excess of 10 kilopascals. Thus there will be no further treatment necessary.
- Because the deposits are immediately trafficable, atmospheric impacts to the process (freezing weather and heavy rainfall) are eliminated.
- Incorporating a relatively simple recycle stream into the process will allow a portion of the VOCs and Naphthenic acids to be destroyed in the process resulting in less air pollution.
- The amount of water available for recycle will be increased since the process does not utilize atmospheric evaporation. This will reduce the amount of water that will be pulled from the Athabasca River.

Detailed Description of MITD Process – The MITD process uses a combination of centrifuges and a direct heated dispersion type dryer to remove the water from the MFT to a level that meets directive 074 immediately. As illustrated in the process flow diagram - Figure 5, the MFT from the settling pond flow to the four centrifuges (CF-101 A,B,C and D). The centrifuge cake from CF-101 A, B& C is routed to a mixer (MA-100) where it is mixed with very dry solids recycled from the drying system before being fed into the dryer (PCX-100). Prior to entering the dryer, the dryer feed is mixed with combustion gas and recycle gas from the natural gas fired burner (BU-100). PCX-100 is a dispersion type dryer that combines the characteristics of a direct heated flash dryer and a mechanically aided dryer. PCX-100 contains unheated rotating discs that break up agglomerates formed in the mixer (MA-100). The heat input for the dryer consists of the burner (BU-100) which is fed with preheated combustion air, a hydrocarbon source (likely natural gas), and a recycled gas stream from the drying system. The solids are dried in the short contact time in the dispersion dryer. The gas and solids leaving the dryer flow into cyclone CY-100. The material is split into 3 streams consisting of a gas stream that flows to the system blower (BL-100), a recycle solids stream that flows to mixer MA-100, and a solids stream that flows to mixer MA-101. The centrifuge cake leaving CF101D flows to mixer MA101 and is mixed with the solid stream from the cyclone (CY-100) to produce the immediately trafficable product. This product will have a shear stress of greater than 10 kilopascals which will meet the requirements of Directive 074. Flocculent is still required for the centrifuge step. However, it is believed that only about 50% of the treat rate will be required since the cake dryness requirements are not as severe as with the centrifuge process.

The gas stream from cyclone (CY-100) passes through blower BL-100 where the pressure is raised to make up for the pressure drop in the system. The stream then splits into a recycle stream that flows back to the burner, and a gas purge stream that allows for purging of the excess gas: CO2, nitrogen, and water generated in the system. The purge gas flows to a scrubber (T-100) where trace amounts of solids are removed, but only a minimal amount of water is condensed. The water required for the scrubber is obtained from the water stream leaving the centrifuge and flows back to this stream after flowing through the scrubber. A recycle pump (not shown) will be provided to maintain a sufficiently high circulation through the scrubber.

Figure 5 also shows the potential amounts of heat that can be recovered by condensing the vent stream or injecting it as direct heating. In order to determine how much of this heat recovery is practical, the needs of the specific location must be determined. For example if there is a need for heat all the way down to 140F, it will be possible to recover 87% of the heat used to dry the MFT. If these streams are condensed, the hot condensate from this heat recovery process can be used in a variety of ways. It can be:

- Recycled to the Clark Hot Water process
- Used as Cooling Tower makeup
- Used as boiler feed water

In the MITD process, water recovery is estimated to be as high as 90% of the water fed to the extraction section. Thus the water consumption of a hot water process using the MITD technology to handle the MFT will be 49 gallons/ barrel of bitumen. This is compared to the current estimated consumption of 224 gallons/barrel.

MITD Process Development Status – As indicated earlier, the MITD process is in the pilot plant development stage. Bench scale drying tests have been conducted on actual MFT material that indicated that the critical moisture level when drying the MFT was well below 5%. The critical moisture level is the point where the drying process moves from rapid drying in the constant rate region to a declining drying rate in the falling rate region. Since large amounts of MFT were not readily available, the pilot plant operations have been conducted using a mixture of kaolin clay and asphalt sealant to simulate centrifuged MFT. Pilot plant wet feed rates exceeding 1000 lbs/hr at 60% solids have been demonstrated. The rate limitation was associated with the auxiliary equipment rather than the dispersion dryer. Work is in progress to upgrade the auxiliary pilot equipment in order to determine the limitations of the pilot plant dispersion dryer. It would be desirable to obtain a larger sample of actual MFT to confirm the sizing criteria, and determine how the components would be scaled up to a larger pilot plant located at the site of the MFT. This will have the advantages of processing the true MFT rather than simulated ones, development of information on scale up parameters, testing the shear strength of actual dried MFT and providing operating experience with larger equipment. Data from this larger pilot plant would be used to confirm the full size design that would handle the entire MFT output from a Bitumen plant producing 125,000 to 150,000 Bbls/Day.

ECONOMIC COMPARISONS

The cost of meeting Directive 074 consists of operating cost and cost of capital. The operating cost includes items such as manning, utilities, and flocculent. The cost of capital is generally expressed as a function of investment using the equation described below:

$$C = f^*I/P \qquad [1]$$

Where:

C = Capital cost in $/Bbl of Bitumen
f = Factor to include maintenance, depreciation, taxes and return on investment. A typical value might be 35% or 0.35 in the equation above.
I = Investment in dollars
P = Bitumen production in Bbls/year

Other factors are as follows:

- Land and water costs were not included. While these costs are real, they are not significant in the overall cost.
- Penalty cost of CO_2 production was based on $15/MT of CO_2.
- Heat recovery down to 140°F was included.

A summary of the cost of meeting Directive 074 using the 3 processes discussed in this paper is as follows:

Technology	$/Bbl*
Thin Film Drying	4.3
Bepex MITD	5.0
Centrifuge	7.4

* $/Bbl based on the cost of capital at 35%

The details of this study are too voluminous for this paper. However a more detailed summary is shown in Table 1 and the cost buildups are available at the end of this document.

ECONOMIC SENSITIVITIES

One of the considerations in any economic comparison is the conversion of the investment to an annual operating cost basis. This is usually done by multiplying the investment by a factor that is meant to include such items as maintenance, depreciation, taxes and return on investment. Typically this factor is about 35% of investment for projects which produce a new product or larger amounts of an existing product. However, for utility type of projects, the factor is lower because the depreciation rate is lower and the expected return on investment is lower. If this factor is reduced to

10% instead of 35%, the cost of the Bepex MITD and Thin Film Drying become essentially equal at about $2.70/Bbl.

The cost of fuel is a significant factor in the economic comparison. As shown in Table 1, the study assumes a cost of natural gas fuel of $5/MBTU. This represents an approximate well head price over the last few years before the discovery and production of significant quantities of shale gas. The $5/MBTU is significantly above the current well head price of approximately $2-3/MBTU. If $3/MBTU is used as the cost of gas it reduces the cost of meeting Directive 074 using the MITD process by about 14 cents/Bbl of bitumen. This change in economic basis will have essentially no impact on the other technologies.

In evaluating the MITD process economics, the authors took a conservative approach and set the product target moistures at 80% solids. Current laboratory data presented in Figure 2 shows that 75% solids product will meet an undrained shear stress sufficient to meet the Directive 074 specification in all of the studies. If the economics were re-evaluated based on 75% solids product, the resulting cost of meeting Directive 074 ($/Bbl) would be reduced.

GREEN HOUSE GASES (GHGs)

Green House Gases (GHGs) generally are considered to be water and carbon dioxide (CO_2) with an emphasis on CO_2. The alleged impact of these gases is well documented. They are a concern associated with recovering bitumen from the Canadian Oil Sands. The current process (surface mining followed by hot water extraction) required to recover the bitumen from the sands is more energy intensive than recovering oil from traditional sources. In addition, the more viscous bitumen requires more intensive processing in the refineries. The sum of these impacts means that processing the Canadian Oil Sands to the final products requires 10 to 15% more energy than the equivalent amount of conventional oil. However, the combustion process of converting petroleum products to energy generates 70 to 80% of the CO_2 produced in the total process. This total process of recovering, processing, and converting the products to energy is referred to as “Well to Wheel”. As shown in Table 2 the amount of CO_2 produced by the processing of the MFT regardless of which technology is used is well below the amount produced by either measure of current production of CO_2 (Well to Wheel or Bitumen Production).

While CO_2 is generally the focus for much of the attention of GHG discussions, water is also considered a GHG. The water vented into the atmosphere from all three technologies is also shown in Table 2. The wide discrepancy in these values is associated with the different processes The MITD process tends to contain much more of the water vapor. The other two technologies make use of the open pits which evolve about 50% of the water removed from the MFT to the atmosphere. The water being evolved from the pits is well in excess of the water formed by combustion. Table 2 is available at the end of this document.

VOCs AND NAPHTHENIC ACIDS

Both Volatile Organic Compounds (VOCs) and Naphthenic Acids are mentioned in literature associated with MFT. These compounds range from toxic materials to simple hydrocarbons that are released to the atmosphere as the MFT are processed in open pit drying. If these materials have sufficient vapor pressures, they will be released to the atmosphere as some of the water associated with the MFT evolves from the pit. If they are not evolved, they will stay with the MFT and become part of the reclaimed soil. They could vaporize from the reclaimed soil over time or simply remain and contaminate the soil. It is believed that the MITD process can be modified to allow a significant amount of these contaminants to be destroyed in the high temperature combustion.

CONCLUSION: THE MITD PROCESS FOR DEALING WITH MFT

This paper introduced a superior technique for *dealing with the MFT (Mature Fine Tailings) from the Canadian Oil Sands,* named MFT to Immediately Trafficable Deposits (MITD).

This process uses direct drying to convert both existing and future MFT into trafficable deposits which can be used for land reclamation. While there may be multiple technologies that seem to have the capability to produce trafficable deposits, these other technologies require further treatment of the processed material. This additional treatment generally involves drying in open pits for an extended period of time.

In addition to being cost competitive with the two leading publicized technologies, the MITD process provides a guaranteed means to produce immediately trafficable deposits that are not subject to atmospheric conditions such as freezing weather and heavy rainfall. The process requires a minimal amount of land, and increases the amount of water that is recycled which reduces the water requirements from the Athabasca River. With the addition of a simple recycle system, it is believed that a significant portion of the VOCs and Naphthenic acids will be destroyed in the process resulting in less air pollution.

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Table 1. Economic Comparison Bitumen Capacity = 124,000 Barrels/Day.

OPERATING COSTS:
Tons MFT/Bbl =0.36 Metric Tons of dry solids/Bbl of bitumen
Investment Related Operating Cost =35% of Total Erected Cost
Assumed Heat Recovery Temperature = 140°F

		Bepex MITD		Thin Film Drying		Centrifuge	
Component (Units)	**Unit Cost**	**Consumption**	**$/Bbl**	**Consumption**	**$/Bbl**	**Consumption**	**$/Bbl**
Electricity							
Centrifuge	$0.06/kwh	6,000 kwh				13,500 kwh	
Other	$0.06/kwh	46,500 kwh		1,000 kwh		1,000 kwh	
Total	$0.06/kwh	52,500 kwh	0.61	1,000 kwh	0.01	14,500 kwh	0.17
Gas	$5/MBTU	2,850 MBTU	2.76				
Flocculent							
Treat (on MFT)	$1.60/lb	600 ppm	0.76	1,300 ppm	1.64	1,300 ppm	1.64
Land Farming					0.42		
Heat Recovery			-2.42				
Investment Related			3.19		2.10		4.58
Other							0.99
CO_2 Penalty			0.1		0.05		0.01
Total			5.0		4.22		7.39
Round Off to two significant figures			**5.0**		**4.2**		**7.4**

Table 2. Green House Gas (Ghg) Impact Associated With Various Technologies.

Basis

- Consideration was only given to increased GHGs associated with producing trafficable deposits.
- Incremental energy assumed to come from natural gas and/or liquid hydrocarbons with following efficiencies
  + Internal Combustion Engine at 30%
  +Electrical Generation at 80%
- Water includes water from vaporization in pits
- Water assumes that final solids product contains 20% water
- Operating hours per year
  +Bepex Process = 8000
  +Others involving land farming = 6000
- Annual Cost evaluated at $15/MT of Carbon Dioxide

Summary of additional GHGs produced:

	Technology				
GHG	**Bepex MITD**	**Thin Film Drying**	**Centrifuge**	**Well to Wheel[(1)]**	**Bitumen Production**
CO_2, lbs/hour	81,000	56,000	10,700		
Lbs/Bbl	16	8.1	1.5	1,100	250 to 350
Water, lbs/hour	410,000	3,810,000	2,040,000		
Cost					
CO_2 $k/year	4,340	2,250	430	304,000	69,000 to 97,000
CO_2 $/Bbl	0.10	0.05	0.01	7.37	1.67 to 2.34

(1) Well to Wheel is terminology that covers total CO_2 production from the oil well through the combustion of the fuel.

FREEZE-THAW ENHANCEMENT OF ALBIAN MATURE FINE TAILINGS (MFT)

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ABSTRACT

Natural freeze-thaw significantly alters the drainage properties that enhance dewatering of the thawed tailings. In this research, Albian MFT was characterized. A series of freezing tests were conducted using different temperature boundaries. Slower freezing rate induced higher solids content and higher undrained shear strength once thawed. In addition, finite strain consolidation tests were carried out on both as-received and frozen/thawed MFT. Freeze-thaw decreased the compressibility by about half compared to as-received MFT and increased the permeability 6 times at the same void ratio. Both compressibility and permeability curves converged at higher effective stress (σ'=100 kPa). These results can be used to evaluate the potential field behaviours and assist with optimizing freeze-thaw dewatering of the Albian MFT.

INTRODUCTION

Oil sands in Canada are mostly found in Northern Alberta where temperatures stay below zero for five months in a year (Environment Canada 2011). Freezing forms polygonal fissures within fine grained soils and tailings are expected to release a large amount of water upon thawing. Johnson et al. (1993) and Proskin (1998) reported that MFT releases over 50% pore water by volume upon freeze-thaw for both Syncrude and Suncor MFT. Dawson (1994) reports significant dewatering of Suncor, Syncurde, and OSLO MFT when thawed. It was known that the location of oil sands mines and extraction methods affect the mineralogy and chemistry component of MFT. An optimum MFT management plan requires knowledge of the behaviour of a site MFT. Since no data has been published on the freeze-thaw dewatering behaviour of Albian MFT, characterization and freezing tests on as-received, and finite strain consolidation tests on both as-received and thawed Albian MFT were performed to investigate the freeze-thaw enhancement.

CHARACTERISTICS OF ALBAIN MFT

The Albian MFT was sampled at a depth of 7.5 m in Albain main pond during June 2008. The properties of MFT, including water content, solids content, void ratio, bulk density, Atterberg limits, specific gravity, bitumen content, fines content, and shear strength were summarized in Table 1. The particle size distribution curve of dispersed Albian MFT was given in Figure 1.

Table 1. Characteristics of Albian MFT.

ω (%)	SC (%)	e	ρ_b (g/cm^3)	PL (%)	LL (%)
174	36.6	4.35	1.29	27	54
LI	**G_s**	**b (%)**	**f (%)**	**S_u (Pa)**	**S_{ur} (Pa)**
5.4	2.53	1.29	99.8	36.5	32.1

Notes: ω = water content, SC = solids content, e = void ratio, ρ_b = bulk density, PL = plastic limit, LL = liquid limit, LI = liquid index, G_s = specific gravity, b = bitumen content, f = fines content, S_u = peak undrained shear strength, and S_{ur} = residual undrained shear strength.

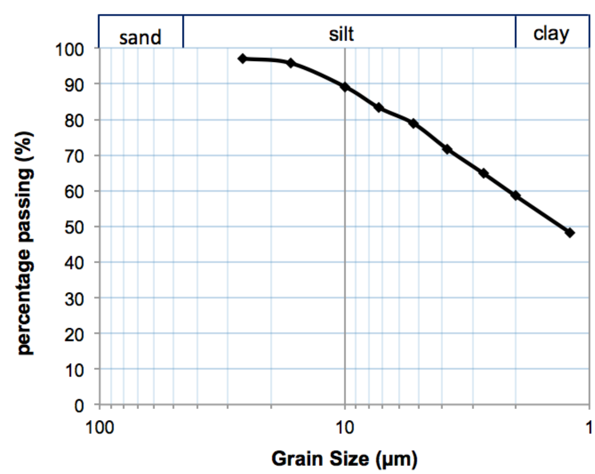


Figure 1. Size distribution of dispersed Albian MFT at 7.5 m.

The water content was determined by oven-drying at 105°C for 24 hours (ASTM 2010a). The solids content was defined as the ratio of solids weight to total weight. The bulk density was determined by measuring the weight of MFT filled a 1 L cylinder.

The Atterberg limits tests were performed as per ASTM (2010b). The specific gravity was determined in accordance to ASTM (2010c). The bitumen content was defined as bitumen weight over solids weight. Fines content was defined as the ratio of weight of solid particles passing sieve # 325 (D=45 μm) to weight of total solids. Wet sieving was used to determine the fines content (ASTM 2007). Hydrometer (152 H) test was adopted to determine the size distribution of dispersed MFT. Undrained shear strength of MFT was measured using the BROOKFIELD DV-II+ programmable viscometer with spindle #73 at shear rate of 0.2 RPM (ASTM 2005).

APPARATUS AND TESTING PROCEDURES

Freezing tests

A series of freezing tests were conducted while applying different temperature boundary conditions to investigate the effect of freezing rate on the post thaw dewatering of Albian MFT. The freeze/thaw cells shown in Figure 2 were used to accommodate samples with the height of 13.6 cm and diameter of 10 cm. The top cap and base pedestal were connected to temperature controlling systems (BATHs) to apply controlled constant temperatures. The freeze/thaw cells were wrapped with insulations to minimize radial heat flow (Figure 3). The tests were performed in a walk-in cooler maintained at temperatures between 0 and 1°C.

Three MFT samples were frozen over 72 hours with top temperature of -20, -10, and -5°C, respectively. All samples were then thawed for 96 hours with top temperature of +5°C. The bottom temperature stayed at 0°C throughout all stages of the tests. Sufficient time (more than 10 days) was allowed for post-thaw self-weight consolidation. At the end of the tests, post-thaw solid content and undrained shear strength profiles were measured.

Two groups of finite strain consolidation tests were carried out to study the effect of freeze-thaw on the consolidation behaviors of Albian MFT. The as-received and frozen/thawed MFT consolidometers (Figure 4) are capable of measuring the deformation and pore pressure during consolidation, and permeability testing at the end of each consolidation stage. Both consolidometers can accommodate initial sample height of 13 cm and diameter of 10 cm.

Figure 2. Freeze/thaw cells.

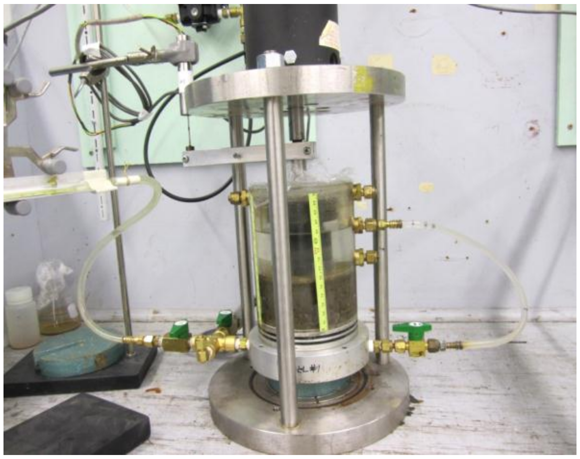
Figure 3. Freezing test setup during freezing all three samples.

Finite strain consolidation tests

Three identical finite strain consolidation tests were conducted on as-received Albian MFT under ambient temperature (+20°C). The samples were consolidated under incremental stresses of self-weight, 0.23, 0.5, 2, 4, 10, 20, 50, and 100 kPa, respectively. Stresses lower than 10 kPa were applied using dead weights, while stresses higher than 10 kPa were provided by a pressurized bellofram system.

Two finite strain consolidation tests were performed on frozen/thawed Albian MFT in a walk-in freezer. The samples were first frozen for 72 hours in the consolidometers with top temperature of -15°C and bottom temperature of -5°C. Then they were thawed for 96 hours with top temperature of +5°C and bottom temperature of 0°C. Both samples were consolidated under self-weight, 1.3, 4, 10, 20, 50, and 100 kPa, respectively. The buoyant weight of the top freezing plate was used to apply 1.3 kPa, and additional stresses were added by virtue of air pressure through an internal loading system.

The consolidation behaviors of both MFT were determined in terms of effective stress versus void ratio, and permeability versus void ratio.



a) As-received MFT consolidometer

b) Frozen/thawed MFT consolidometer

Figure 4. Finite strain consolidometers.

RESULTS AND DISCUSSIONS

Freezing tests

MFT samples were separated into a clear aqueous layer on top of a dense soil layer following freeze-thaw and post-thaw consolidation. Figure 5 presents the solids content profiles of the as-received and thawed MFT. Freeze-thaw enhanced the solids content of all three samples by different magnitudes. Sample one (top freezing temperature -20°C) and sample two (top freezing temperature -10°C) had similar solids content profiles. Solids content first increased linearly till 4.8 cm, then remained nearly constant to 8 cm, and finally increased significantly near the bottom. Solids content of sample three (top freezing temperature -5°C) followed the same trend but had higher solids content than the other two samples. The concave part of the curve between 3.7 and 6.7 cm was due to backfill of surface water and did not reflect the actual thawed solids content.

Figure 6 shows the undrained shear strength profiles of as-received and thawed MFT. Freeze-thaw increased the shear strength of all three samples, and higher freezing rate resulted in lower shear strength. More specifically, the profiles were close to 3.3 cm depth and then differed. Shear strength of sample one (top freezing temperature -20°C) increased gradually with depth. Shear strength of sample three (top freezing temperature -5°C) increased most sharply and reached 1.2 kPa at the bottom.

Freezing induces fractures and over-consolidated soil peds that form between the ice-filled fractures. The soil peds settle under self-weight upon thaw and the fractures behave as flow channels to drain water. Released water gathered on top of the MFT. The total volume of surface water and final MFT sample equalled the initial sample volume. Therefore, freeze-thaw is capable of enhancing the solids content and shear strength of MFT (Zhang 2012).

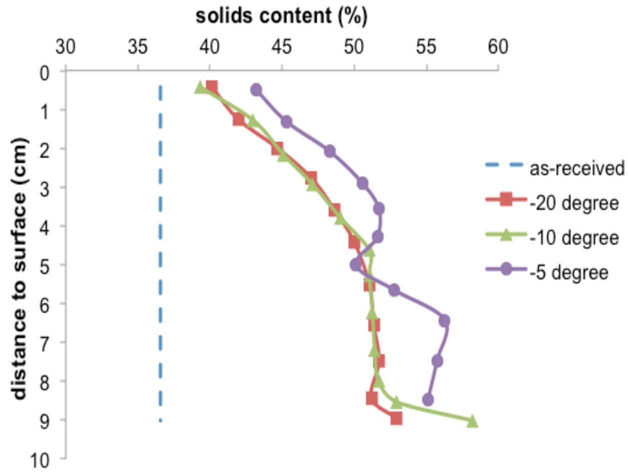


Figure 5. Post-thaw solid content profiles.

A higher freezing rate froze most pore water in place and ice crystals retarded water from flowing to the freezing front, while lower freezing rate allowed water flow to the freezing front resulting in additional consolidation of the MFT. Consequently, lower freezing rate resulted in higher post thaw solids content and shear strength.

MFT was placed in the walk-in freezer for a few days before testing. Its temperature was maintained constant and close to 0°C. A period of time was required to achieve a linear temperature distribution in MFT samples after applying the freezing temperatures. As a result, the upper part of MFT was frozen at much higher freezing gradient. Solids content and shear strength of thawed MFT were lower and the profiles were closer at the upper part as shown in Figure 5 and Figure 6.

Figure 7 shows the comparison between undrained shear strength and solids content for both as-received and thawed MFT. Freeze-thaw enhanced the shear strength comparing with those of as-received MFT. The curves for thawed MFT follow the same trend. The shear strength increased gradually until solids content reached 50%; then the rate of increase became much higher. Suction generates at the freezing front and further consolidates MFT without adding additional overburden pressure. MFT also has high thixotropic ratio and gains strength with time (Suthaker 1995). Both pre-consolidation pressure and thixotropic behavior are responsible for the higher shear strength of thawed MFT at the same solids content in Figure 7.

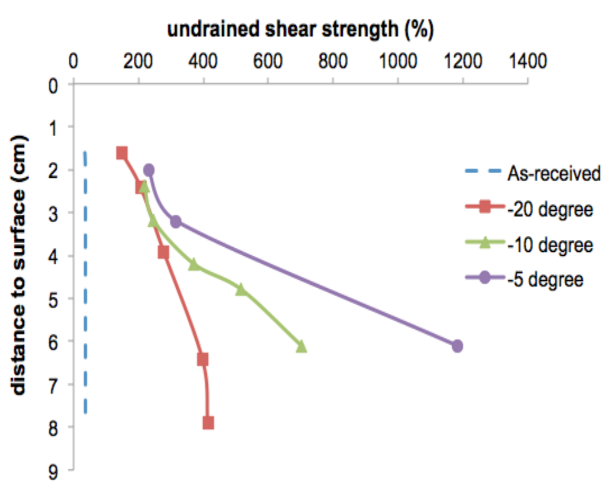


Figure 6. Post-thaw undrained shear strength profiles.

The effect of over-consolidation pressure generated during freezing is more significant in soil than in a water-dominated slurry. Thus, shear strength increased much faster when MFT approaches higher solids contents and becomes more soil like.

Finite Strain Consolidation Tests

Figure 8 presents the compressibility curves for both as-received and frozen/thawed MFT. One cycle of freeze-thaw released 45% of initial pore water by volume and reduced the void ratio from 4.35 to 2.42. The as-received MFT was more compressible comparing with the once thawed MFT. The compression index (Cc) of as-received MFT was about 0.18 and 0.10 for thawed MFT. All compressibility curves converged at effective stress of 100 kPa where the void ratio approached 1.0.

As discussed in the previous section, freeze-thaw can release significant amount of pore water and change the compressibility. Variation in freezing gradient during temperature equalization and distance to freezing front induce differences in suction/pre-consolidation pressure within MFT. During the post-thaw consolidation, the compressibility of thawed MFT gradually approaches that of as-received MFT and became the same at effective stress approaching 100 kPa.

Figure 9 presents the permeability behaviors of both as-received and thawed MFT. Freeze-thaw increased the permeability by 6 times with the same void ratio. Permeability of as-received MFT varied between 3.3×10^{-7} and 5.2×10^{-9} cm/s during the tests. The permeability of thawed MFT was more sensitive to void ratio and in the range of 5.4×10^{-7} cm/s to 1.0×10^{-8} cm/s. All permeability curves converged as the void ratio approach 1.0.

The weights of soil peds and small overburden stress do not fully close the flow channels created during freezing. Although soil peds are less permeable, excess pore water would escape via the short path of least resistance. The overall permeability is higher than that of as-received MFT at the same void ratio and it decreases significantly with decreasing void ratio. Permeability of thawed sample would finally approaches the as-received sample once the effect of freeze-thaw eliminates.

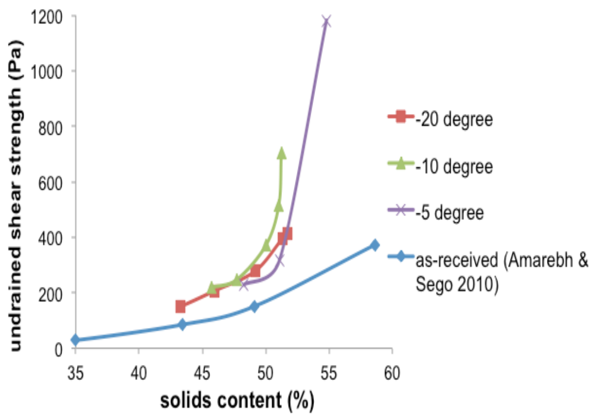


Figure 7. The relationship between undrained shear strength and solids content.

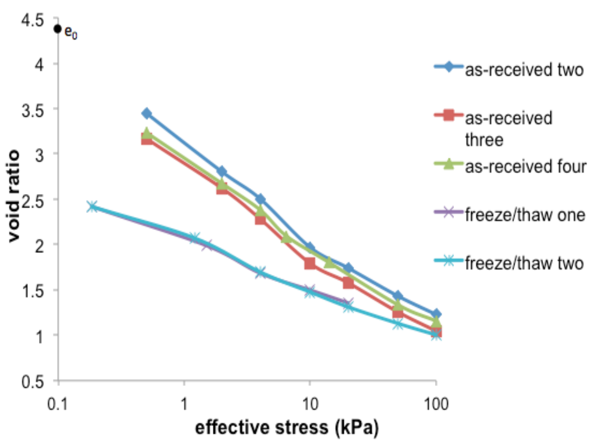


Figure 8. Compressibility behaviors of both as-received and frozen/thawed MFT.

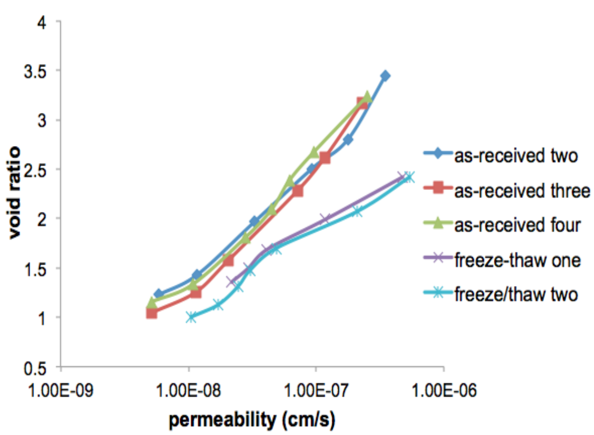


Figure 9. Permeability of both as-received and frozen/thawed MFT.

The weights of soil peds and small overburden stress do not fully close the flow channels created during freezing. Although soil peds are less permeable, excess pore water would escape via the short path of least resistance. The overall permeability is higher than that of as-received MFT at the same void ratio and it decreases significantly with decreasing void ratio. Permeability of thawed sample would finally approaches the as-received sample once the effect of freeze-thaw eliminates.

CONCLUSIONS

The freezing tests revealed that freeze-thaw can increase the solids content and shear strength of Albian MFT. Lower freezing rate induces higher solids content and shear strength. In addition, undrained shear strength increases with solids content and the rate of increase becomes larger at solids content higher than 50%.

The finite strain consolidation tests on both as-received and frozen/thawed MFT demonstrated that freeze-thaw reduces the compressibility to about its half in terms of compression index, and increases the permeability by six times of as-received MFT with the same void ratio. However, all compressibility and permeability curves converge when effective stress approaches 100 kPa.

Freeze-thaw has the potential to be one of the options for dewatering of Albian MFT.

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HIGH SPEED DEWATERING OF OIL SANDS TAILINGS

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ABSTRACT

Slow settling rates of tailings ponds lead to a low rate of water recovery and inefficient, unsustainable handling and disposal of tailings. The industry is looking for a technology that solves these problems in real time, and that is sustainable and future proof for meeting higher environmental standards.

Genesis, a US company, holds patents on a dewatering system that can be land or barge mounted and that works with portable hydraulic dredges to instantly dewater slurry from tailings ponds, or it could potentially take a fine tailings flow upstream of the tailings pond. The slurry is pumped to the Genesis Rapid Dewatering System (RDS), which instantly separates solids, including sand, ultra-fine clays, silts, organics, and rock dust down to 0.0007 micron in size, while simultaneously returning clear water (<20 ppm total suspended solids). This highly automated system with no moving parts has been used in commercial waterway dredging projects with hydrofillic clays, in circuits at aggregate plants, and on paper pulp sludge tailings.

A two or three stage process instantly recovers the free water phase from slurries, yielding stackable, truckable, or conveyable solids for disposal or reuse. Coarse sand, if present, is removed, and the fine grained material is flocculated and pumped to the Genesis AquaScreen. While other methods require waiting for flocculated solids to fall slowly through the water column in a thickener or clarifier, the AquaScreen instantly drains water away from the flocculated mass. If stackable cake is desired, the dewatered cake from the AquaScreen undergoes secondary dewatering in the Genesis TerraCore units. Each TerraCore has a unique internal structure designed to release capillary water at high speed through the specially designed apparatus. As secondary water is released, suspended solids, dissolved solids and residual polymer are attracted to the multitude of sites within the fine grain lift of solids. This water is ready for discharge or further treatment, if needed.

INTRODUCTION

Development of High Speed Dewatering Technology

Dewatering slurries at high speeds and volumes has met with the historical stumbling block of ultra-fine grained material, such as clays, silts, organics, and rock dust. In industries ranging from waterway dredging to various types of mining and oils sands tailings, this has led to widespread use of confined disposal facilities (CDFs) and designated disposal areas (DDAs), or tailings ponds. Dewatering time for fine grained material through natural settling in these ponds can be as long as seven years.

Attempts to mechanize and speed up the process of dewatering ultra-fine solids through the use of clarifiers, centrifuges, and belt presses can be costly, susceptible to downtime, and unable to process at a 1:1 ratio with high volume inflows of thousands of liters per minute.

Hydraulic dredging combined with a relatively new type of high speed dewatering equipment for handling ultra-fine solids offers potential as an alternate approach to the clean-up of tailings ponds, whether for environmental remediation or to extend their capacity. The patented Genesis Rapid Dewatering System (RDS) has been in commercial use in the US waterway dredging sector since 2010. The purpose of the Genesis RDS is to achieve a 1:1 ratio of dredging (inflow) to dewatering of ultra-fine solids as small as 0.0007 micron at high speeds and volumes. Other objectives include continuous operation, minimal energy and maintenance, immediate recovery of clear water discharge, and production of stackable, truckable solids.

Process Flow for Rapid Dewatering

To apply this technology for cleaning out a tailings pond, a hydraulic dredge would feed the RDS, which has a 2-3 step process, involving key equipment components of the system. First, a desander removes and stockpiles coarse sand, if present. This can be accomplished with linear motion shakers over hydrocyclones. See Figure 1.

Next, the slurry of mature fine tailings is flocculated with polymer. Lab testing with the screen component of the equipment system determines the optimum polymer and dosage to flocculate the material. Density and flow meters adjust to changes in the dredge flow, so the fully automated polymer unit injects precise quantities into the slurry, which is then pumped to the Genesis AquaScreen. See Figure 2.

Figure 1. Desanding Units.

Figure 2. Polymer Injection System.

In this step of the process, the AquaScreen instantly strips the free water phase from the flocculated slurry as it passes over a stationary screen surface. This patented equipment immediately separates the fine grained solids, such as clays, silts, organics, and rock dust from water, as the slurry flows over the screens. Clear water passes through small openings in the screens in high volumes and can be returned to the waterway or recycled.

The AquaScreen removes the majority of the water from the dredged tailings slurry at high speed. The technology overcomes the channeling properties of the slurry with a patented manifold system that evenly distributes the slurry column. This enables continuous operation at high throughputs. See Figure 3.

The material typically comes off the AquaScreen in the 50-60% solids range for hydrophobic material. The flocculated tailings slide off the screens and are conveyed by gravity through chutes into TerraCores, the final stage of the technology.

In this last step of the system, a bank of TerraCores produces transportable cake. Each TerraCore resembles a large, open top box of between 7-10 meters in length, processing approximately 50 tons per hour. See Figure 4.

The TerraCore has patented, overlapping, concentric, dewatering zones built into the interior design structure. The 2.5 meter lift of flocculated mass typically releases secondary water in 3-24 hours, depending on the material. Hydrophobic materials generally give up water faster than hydrophillic slurries. The resulting solids are excavated from the TerraCores and can be stacked or transported. It may be possible to automate removal of the solids without need for excavation.

Figure 3. Aqua Screen.

Figure 4. TerraPods in Front of an Aqua Screen.

Operating Parameters and Production

The AquaScreen scales to any rate of inflow by adding units. Each unit dewaters up to 9,500 liters/min, or an estimated 200 tons/hour of tailings slurry in continuous operation. The technology can process slurries with a typical range of between 2-40% solids by weight. The TerraCores can also be scaled up by adding additional units.

The solids production off the AquaScreen is normally in the 50-60% range for hydrophobic materials and lower for hydrophillic substrates. The clear water discharge from the AquaScreens is typically at <20mg/l TSS and <30 NTU.

If used, the TerraCores will yield stackable, transportable solids. The percentage of solids varies, depending on the material, but can be as high as 60-70%. The clear water discharge from the TerraCore is typically at <10mg/l TSS and <30 NTU. See Figures 5 and 6.

The technology has potential for relieving future tailings operations from the free water phase with the AquaScreen alone. The instant recovery of large volumes of clear, reusable water puts the operation on the path to environmental sustainability. The AquaScreen and the TerraCore have no moving parts and little or no power requirement. If high speed production of a transportable solids or assets is desired, the TerraCores provide the final step in capillary moisture release that is sufficient enough to enable immediate trucking or conveyance of tailings without the use of presses or vacuum.

The footprint for one AquaScreen unit, one polymer system and 16 accompanying TerraCore units is approximately 30x45 meters. When scaled up, efficiencies can be gained with polymer injection, so the footprint does not necessarily increase at the same rate as the system size. Equipment can have different configurations, depending on space available, for example, a U-shape (see Figure 7), linear, or L-shaped.

The equipment was designed for mobility and ease of set up, which can be completed in eight days for a standard system. Larger scale operations may require 2-4 weeks, depending on size of the system. The AquaScreen and TerraCore are over-the-road compatible without special permits (see Figures 8 and 9).

Figure 5. Solids Being Excavated from a TerraCore.

Figure 6. Clear Water Discharge from an AquaScreen and TerraCores.

Footprint and Mobility

Figure 7. Equipment Set Up On Soil Washing Project.

Figure 8. An AquaScreen Arriving at Site for Set Up.

Figure 9. Two TerraCores Arriving for Set Up

CONCLUSION

Rapid dewatering of tailings dredged from DDAs, or upstream tailings dewatering could mark a new approach to environmentally sustainable tailings handling. Pilot testing will determine the size of the system needed, and provide more accurate and precise data on production of water and solids for the specific tailings involved in the application. The ultimate goal of this new methodology is minimizing the hydraulic loading of DDAs, and thereby minimizing the land required to dispose of tailings. Without the free water phase, more options arise for material handling at lower costs.

OIL SANDS MATURE FINE TAILINGS AND BITUMEN EXTRACTION

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ABSTRACT

In Alberta over 1,300,000 bbl/d of bitumen is produced from surface mineable oil sands using the Clark Hot Water Bitumen Extraction process. This process uses *NaOH* or sodium salts of weak acids as extraction process aids which provides acceptable bitumen recovery efficiency; however, it produces a toxic mature fine tailings and increases salinity of the process water. Current mature fines tailings inventory exceed 800,000,000 cubic meters and this volume is steadily growing as the present mines are expanded and new ones opened. To manage the environmental impact of oil sands tailings the Energy Resources Conservation Board of Alberta now requires that oil sands mining companies develop measures to reduce the amount of fluid fine tailings being produced by 50% by 2013.

Oil sands industry previously implemented non-caustic extraction processes to produce tailings with improved settling and consolidation characteristics; however, these processes were largely unsuccessful and later modified to some version of the Clark Hot Water Bitumen Extraction process. Also, the Composite (or Consolidated) Tailings process was implemented to reduce the inventory of the mature fine tailings. In this process the whole tailings was passed through cyclones and the cyclone underflow was blended with mature fine tailings and treated with $CaSO_4$ to prevent segregation of the fines. Unfortunately, cyclone overflow effluent produces additional mature fine tailings. Furthermore, the $CaSO_4$ additive harms the release water chemistry by increasing Ca^{2+} concentration and results in H_2S emission by the biological reduction of SO_4^{2-}.

To reduce the environmental impacts of oil sands plants we focused on two fundamental process concepts: (i) reduce clay dispersion in the extraction process by using CaO, O_3 and Biodiesel as extraction process additives; and (ii) dispose tailings as a nonsegregating material by blending Cyclone Underflow, Thickener Underflow and existing mature fine tailings by using CaO or CaO and CO_2 additives to prevent segregation. Implementation of both bitumen extraction and NST production processes would reduce the environmental impacts of oil sands plants. Use of CaO as an extraction process aid would reduce clay dispersion, eliminate increase in process water salinity, specifically Na^+ concentration and reduce mature fine tailings production without harming bitumen extraction efficiency or fuel quality of bitumen. Furthermore, it would potentially reduce clay content in bitumen froth and thickener overflow and reduce water soluble naphthenic acids salts content.

Key words: Bitumen extraction, CaO, O_3 and biodiesel additives, clay dispersion, release water chemistry, nonsegregating tailings using CaO and CO_2.

INTRODUCTION

Background

In Alberta, surface mineable oil sands are commercially utilized for bitumen production by oil sands ore-water slurry based extraction processes. Oil sands plants are composed of: (i) ore-water slurry preparation and extraction; (ii) froth treatment; and (iii) tailings disposal unit operations. A froth composed of about 60 % bitumen, 30 % water and 10 % solids is produced in the extraction plant. In the froth treatment plant bitumen is recovered using naphthenic or paraffinic solvents depending on the downstream processing of the bitumen (Long et al. 2002; Romanova et al. 2003; Shelfantook, 2004). Bitumen produced using naphthenic solvents has a higher asphaltenes content (about 18 %) in comparison to those produced using paraffinic solvents (about 8 %). Bitumen with higher asphaltenes content is suitable for coking, producing coker gas oil which is upgraded to synthetic crude oil by hydrotreating processes. Bitumen produced using paraffinic (i.e. $i\text{-}C_5$, C_5 and C_6) solvents in froth treatment is of lower asphaltenes content and of lower viscosity; which is suitable for pipeline transportation and more acceptable for further upgrading to refined products. Because bitumen is directly fed to the refineries, contamination of bitumen with organometallic (organic bound multivalent metals, such as the alkaline earths Ca^{2+} and Mg^{2+}) complexes is undesirable. Tailings produced at the froth treatment plant are about 10% of the total tailings; with a composition which depends on the type of solvent used in the froth treatment plant (about 10 % hydrocarbons by paraffinic and <4% hydrocarbons by naphthenic solvents). Regardless of the type of extraction process and the solvent used for froth treatment, bitumen with minimum clay, organometallic species and water contents is desired for further upgrading.

WATER RELATED ENVIRONMENTAL IMPACTS OF OIL SAND PLANTS

Oil Sands Ore and Bitumen Extraction

Oil sands contain minor amounts of clay particles in the water layer surrounding the quartz sand grains. The great majority of clay minerals in the tailings come from the indurated clay-shale discontinuous seams and layers in the oil sands ore bodies. These dense but weak clay-shale materials are broken up during the mining process and the larger more indurated pieces are screened out as reject materials. In the bitumen extraction process, the clay-shale pieces are further broken down into clay aggregates and lumps and some clay lumps are dispersed into small clay booklets and flakes. The extent of clay dispersion in the extraction process depends on the additives used to promote bitumen extraction efficiency. These process additives effectively disperse much of the clay aggregates into clay flakes smaller than 2 μm which have large active clay surfaces.

Bitumen extraction in oil sands ore-water slurry based systems involves physical, chemical and hydrodynamics processes. Bitumen viscosity (bitumen mobility), bitumen liberation from the sand matrix, bitumen coalescence and aeration are the major factors controlling the extraction efficiency, as drafted in Figure 1. Decades of research and commercial experience confirmed that bitumen viscosity, therefore slurry temperature, surface and interfacial tensions and hydrodynamics of the slurry are the major factors influencing the efficiency of bitumen recovery (Masliyah, 2004; Kasperski, 2001).

Clark Hot Water Extraction (CHWE) Process

CHWE process uses *NaOH* or Na^+ salts of weak acids (such as *Na-Citrate*) as additives to promote bitumen extraction process. These additives increase the *pH* of the ore-water slurry to about 8.5-10.5. At elevated *pH*, asphaltic acids, which are partly aromatic, contain oxygen functional groups such as phenols, carboxylic acids and sulphonic acids that dissociate into ions. These anionic functional groups are water soluble which allows them to act as surfactants, reducing the surface and interfacial tensions and promoting the efficiency of the bitumen extraction process (Moschopedis et al, 1977 and 1980; Speight and Moschopedis, 1977; Clark and Pasternack, 1932; Clark 1939).

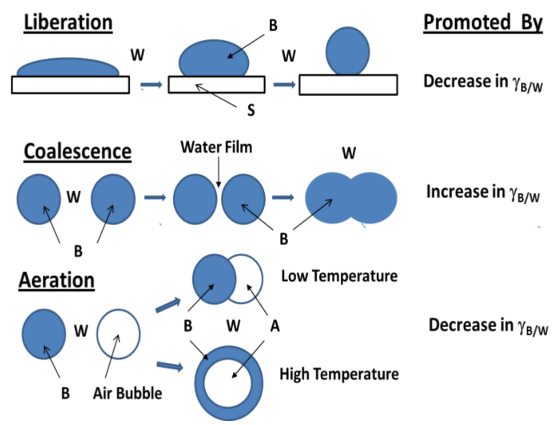

Figure 1. Bitumen extraction mechanisms.

In Alberta, Suncor Energy Inc.'s oil sands plant became operational in 1967; it was the first oil sands plant producing bitumen using the CHWE process. After Suncor's pioneering commercial experience all of the oil sands plants commissioned since have been designed to use the same extraction process with minor modifications. Design decisions on the newer plants were made probably because of the high capital investment associated with the development and implementation of novel extraction processes. This policy obviously reduced investment risks; however, amplified the hazards associated with the short fallings of the CHWE process.

Advantages and Disadvantages of CHWE Process

CHWE process or its slightly modified versions are used at all oil sands plants in Alberta, producing bitumen over 1,300,000 bbl/d capacity. CHWE processes produce about 1.2 m^3 to 1.5 m^3 tailings per barrel of bitumen. Tailings produced at the extraction plant are composed of sand, silt, clay, water and residual bitumen. Tailings discharged from the extraction plant segregate during deposition because of low solids contents; the sand particles segregate and settle out to form dykes and beaches, while the fine particles (<45 μm size) are carried with the run-off water into the ponds. Over a two year period the fine particles settle and form Mature Fine Tailings (MFT), which is a stable suspension of fluid fine tailings that has undergone settlement and compression to about 33% solids content by mass. Further densification of the MFT is a very slow process; as a result of which all tailings produced and discharged into tailings ponds without any treatment contribute to the accumulation of MFT.

CHWE process maintains acceptably high bitumen extraction efficiency; however, it causes clay dispersion in the ore-water slurry and produces tailings with poor settling and consolidation properties which results in the production of toxic MFT. Seriousness of the dispersion of the silt-clay size fraction of the oil sands ore for its disposal was firstly highlighted by Dr. Clark (Clark, 1939). What has been addressed in 1930s became the reality of 2000s.

Bitumen extraction plants use water at about 7 to 9 times of the volume of produced bitumen. Because of the zero discharge policy implemented on these plants the majority of the water demand is supplied by recycle water and only about 3.1 volumes of fresh water per volume of bitumen produced is imported from the Athabasca River. As a result, *NaOH* additive used in CHWE process causes steady increase in the salinity, specifically Na^+ concentration, of the process water over the years (Allen, 2008a, 2008b). Increase in water salinity detrimentally effects performance of bitumen extraction and tailings disposal processes.

An overall assessment of oil sands plants, as sketched in Figure 2, would conclude that extraction and tailings disposal processes are coupled. An additive used in extraction process promoting dispersion of the silt-clay particles and process water salinity should be assessed as the source of tailings disposal problem. Dispersion of the silt-clay size particles and increase in water salinity, specifically increase in Na^+ concentration, are the roots of the problems called "*the environmental impacts of oil sands plants*".

Since the water is the principle component added to the slurry being treated in the extraction and in the tailings disposal processes, the main cause of the environmental impacts of oil sands plants is the water chemistry. Water chemistry is eventually being altered by the chemical additives used in both extraction and tailings disposal processes. Therefore, water chemistry should be the focus of attention in the implementation of novel extraction and/or tailings disposal processes to reduce the environmental impacts of oil sands plants.

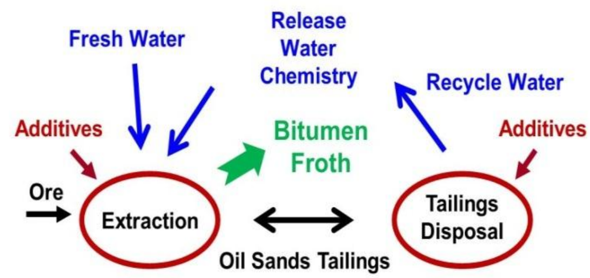

Figure 2. Schematic of an oil sands plant.

ERCB Directive 074 and Oil Sands Industry's Tailings Management Plans

Existing MFT inventory is exceeding 800,000,000 cubic meters and its volume is steadily growing as the present mines are expanded and new ones opened. Accumulation of MFT is considered an environmental liability; both the oil sands industry and government regulatory agencies have committed to reduce the production of MFT as outlined in the ERCB Directive 074 (ERCB, 2009). The technologies proposed by the oil sands players have appeared as novel and ambitious processes for the management of their MFT inventory; cost effectiveness and practicality of which appear to be foremost constraints.

All oil sands operators envision the use of end pit lakes at mine closure. Any fluid tailings remaining at this time will be returned to the mined pits and capped with water. It is planned that these lakes will, with time, become viable ecosystems that will sustain plant and aquatic life. Environmental concerns exist with this concept and end pit lakes have not yet been approved by the ERCB.

Syncrude has been conducting long-term field tests to develop and prove out this technology.

Previous Effort to Reduce MFT Production

Oil industry is aware of the fact that production of MFT is caused by *NaOH* additive used in CHWE process. To reduce or eliminate problems associated with the CHWE process tailings five decades of research efforts have been devoted for the development of alternative processes for bitumen recovery; however, none of these processes could be used commercially (Ozum and Scott, 2012).

Another remarkable effort made to reduce MFT production was the production of CT (Composite or Consolidated Tailings). This process was developed at the University of Alberta, based on the use of cyclones and treating the Cyclone Underflow tailings with existing MFT and treating the blend with CaSO4 as an additive to prevent segregation of the fines. CT process was implemented at the Suncor Energy Inc.'s plant and became a standard tailings disposal practice of the oil sands industry (Caughill et al, 1993; Liu et al, 1995). CT production has short fallings too; it produces additional MFT from the Cyclone Overflow, increases Ca^{2+} concentration in process water and emits H_2S by biological reduction of SO_4^{2-} in the tailings. CT production would possibly become impossible because of the steady increase in process water Na^+ content (Ozum and Scott, 2009).

Our Research Effort to Reduce Environmental Impacts of Oil Sands Plants

Our oil sands research policy is shaped by accepting two fundamental objectives:

(i) reduce the extent of clay dispersion in extraction process without harming bitumen extraction efficiency and fuel quality of bitumen; and,

(ii) dispose the tailings as a nonsegregating tailings (NST) so that the fines would not segregate from the sand.

While satisfying these fundamental objectives our research also focus on fulfilling the constraint that bitumen extraction and tailings disposal processes are coupled because of the process water being recycled. This makes the process water chemistry the key factor for the long term sustainability of the oil sands plants. Our research effort is focused on how to reduce water salinity caused by the additives used in bitumen extraction and tailings disposal processes. In fact, froth quality may also be influenced by the chemistry of the water used in extraction process. Froth quality may be harmed by the high pH resulting from the CHWE. High pH increases the activation of organic anions, such as carboxylates. These anions are necessary precursors for the generation of bitumen soluble organometallic complexes, such as calcium napthenates, that can damage refinery catalysts.

We have been developing novel bitumen extraction process additives such as *CaO*, O_3 (ozone) and *BD* (biodiesel, fatty acids methyl ester) which matches the above presented objectives (Ozum and Scott, 2010a; Ozum and Scott, 2010b; Babadagli et al, 2008). Our research findings made us believe that the use of *CaO* as a *pH* adjusting additive replacing *NaOH* in the CHWE process is the first step to be taken for long term sustainability of the oil sands plants.

***CaO* as an Extraction Process Additive**

When *CaO* is introduced into oil sands-water slurry system it forms $Ca(OH)_2$ by hydration with water and is simultaneously involved with the following reactions:

$$CaO + H_2O \leftrightarrow Ca(OH)_2 \qquad [1]$$

$$2Clay - Na + Ca(OH)_2 \leftrightarrow (Clay)_2Ca + 2NaOH \qquad [2]$$

$$2Naph - Na + Ca(OH)_2 \leftrightarrow Ca(Naph)_2 + 2NaOH \qquad [3]$$

$$Ca(HCO_3)_2 + Ca(OH)_2 \leftrightarrow 2CaCO_3 + 2H_2O \qquad [4]$$

by which *pH* of the ore-water slurry increases (Equation 1) similar to the CHWE process. Increase in *pH* would be lesser than to that of the CHWE process additive *NaOH* because of the nature of the dissociation constant of the reaction: $Ca(OH)_2 \leftrightarrow Ca^{2+} + 2OH^-$. Also, the water swelled *Na-Clay* flocculates (Equation 2) and bicarbonates turn to carbonates (Equation 3) simultaneously. There are many other chemical reactions take place; however, the reactions expressed in Equation 1 to 4 are the stoichiometrically independent set of reactions that we considered in our bitumen extraction and NST production processes. The *pH* and the temperature of the extraction slurry are probably the most important parameters to be measured because of their relations to the thermodynamic state of the complex reactive system. We also assume that hydrodynamics of the extraction and tailings disposal processes are sufficient to reduce mass and energy transfer resistances to a minimum and the equilibrium state is solely controlled by the thermodynamics of the system.

In our extraction process research, because of the complex composition of the oil sands ore and process water, optimum operating conditions were determined experimentally. To determine the effect of process parameters such as temperature, additives and additive dosages, extraction tests were performed using a Denver D-12 Flotation cell as a standard test apparatus. The speed of the rotor was set 800 r.p.m. to minimize the energy input into the slurry system so that the effect of process variables on extraction efficiency could become detectable.

Experimental

A set of extraction tests using 150 mg-*CaO* as a process aid was performed on an ore sample received from Shell Canada Ltd. which was of 7.7% bitumen, 9.7 % moisture and 82.4 % solids, all measured by the Dean-Stark extraction. The fines fraction (<45μm) of the solids was 27.7 %; because of which the ore was consider as a “high fines ore”. This study showed that the use of *CaO* as an extraction process additive promotes bitumen recovery efficiency without harming release water chemistry; therefore, it offers a great opportunity for the long term sustainability of the oil sands plants (Apex Engineering Inc., 2009; Ozum and Scott, 2009).

In most of the previous extraction tests *CaO* additive was used under 150 mg/kg-ore dosages and its effect on clay dispersion in the extraction slurry was not investigated. Because of the established misconception on the use of *CaO* as an extraction process additive, its usage higher than 150 mg/kg-ore dosages were not systematically investigated either. Our recent studies were focused on the use of *CaO* at higher dosages and its effect on primary and secondary extraction froths and clay dispersion. The optimal *CaO* dosage would most likely be depended on bitumen composition (such as the oxygen content), fines content and clay type in the fines; which requires further research.

Further Extraction Tests with *CaO* Additive

A new set of extraction experiments were performed to compare the performance of *NaOH* and *CaO* as extraction additives used at 300 mg/kg-ore dosage at 50 ^{o}C temperature. Extraction tests were performed using a Denver D-12 Flotation cell apparatus, rotor speed was set at 800 r.p.m. A normal grade ore provided by Syncrude Canada Ltd which was of 9.8% bitumen, 84.8% solids, 3.7 % water (all by Dean-Stark extraction); 9.5 % of the solids was fines (<45 μm). Artificial process water was used in these tests; chemistry of which is presented in Table 1.

Table 1. Artificial Process Water Chemistry.

	Total Alkalinity		Cations			Anions				
	(mg CaCO3/L)		(mg/L)			(mg/L)				
pH	Alkal	Hard	Na	Ca	Mg	Cl	HCO3	SO4	CO3	OH
8.4	324	130	332	30	14	214	375	255	10	0

Photographic images of the oven dried +320 mesh (>45 μm) fraction of the extraction tailings, extraction tailings and decant water samples recovered nine days after the extraction tests are presented in Figures 3, 4 and 5 respectively. Visual inspection of these representative samples show that *CaO* as an extraction process additive provides equivalent or higher extraction efficiency than *NaOH* when these additives are used at 300 mg/kg-ore dosages. Also, *CaO* additive reduces dispersion of the silt-clay size particles in comparison to *NaOH* additive. Photographic images of the recovered tailings after 13 days of settling are shown in Figure 5, which also shows that *CaO* as an extraction additive causes lesser dispersion of the silt-clay size particles compared to that of *NaOH*. All these visual observations show that use of *CaO* as an extraction process additive provides equivalent bitumen recovery efficiency and produces tailings with improved settling characteristics in comparison to *NaOH* additive. Similar results were also obtained in a previous study performed on Shell Canada oil sands ore samples.

It has been reported that excessive reduction in bitumen-water interfacial tension, when *NaOH* is used as an extraction additive, could promote wetting of bitumen droplets with water and detrimentally effects bitumen recovery efficiency (Schramm and Smith, 1987). The extent of bitumen-water interfacial tension reduction could be the most optimal when *CaO* is used as an extraction process additive.

Figure 3. Images of oven dried tailings.

The primary, secondary and total bitumen recovery efficiencies and solid-to-bitumen ratio of extraction froths are presented in Figures 6 and 7 respectively. The primary recovery favors *CaO* whereas the secondary recovery favors *NaOH*; however, neither difference was statistically significant and their combined recoveries were comparable or slightly higher for *CaO* additive. Bitumen recovery without an additive was erratic and generally lower.

Figure 4. Filtered (<45 µm) fraction of non-additive and *CaO* and *NaOH* additive extraction tailings.

The S/B (solids-to-bitumen) ratios in the primary and secondary froths were determined to be similar in non-additive and *NaOH* and *CaO* additive extraction froths. Extraction efficiencies and froth qualities using *NaOH* and *CaO* process additives are comparable, indicating that *CaO* is as an effective, if not a superior, extraction process additive as *NaOH*.

Figure 5. Extraction tailing, *CaO* and *NaOH* additives after 13 days.

Hydrometer Tests

Dispersive (ASTM D422-63) and non-dispersive (double, ASTM D4221-99) hydrometer tests were performed on the tailings produced by non-additive (blank) and *NaOH* and *CaO* additive extraction tests; results of which are presented in Figures 8 and 9. The particle size distributions measured by dispersive hydrometer tests presented in Figure 8 shows that fines fractions of the tailings produced by non-additive and *NaOH* and *CaO* additives extraction tests are identical. That is the indication that the oil sands ore samples used in these tests were almost identical.

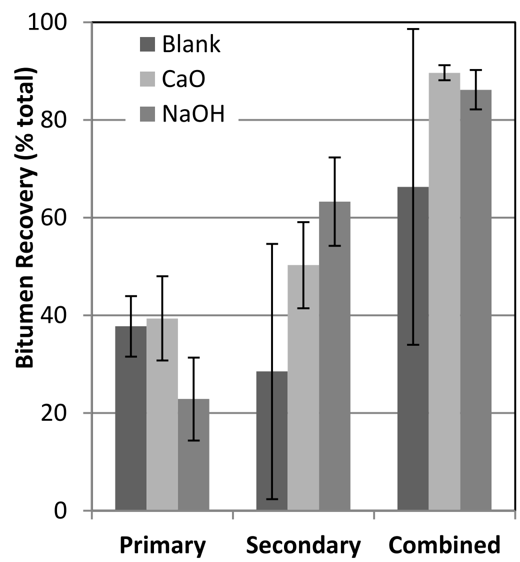

Figure 6. Effect of extraction additives on bitumen extraction efficiency.

It has been known that the influence of extraction process additives on tailings properties could be recognized by the non-dispersive hydrometer tests (Miller et al, 2010). Dispersive and non-dispersive hydrometer tests results for the fines fractions of the tailings show that the non-dispersive particle size distributions had lower percentage of fines than that of the dispersive tests. The difference between the dispersive and non-dispersive particle size distributions depends on the type of chemical additive used in the extraction process. Data presented in Figure 9 show that *CaO* as an extraction process additive causes lesser dispersion of silt-clay size particles in comparison to that of *NaOH* additive (300 mg/kg-ore dosages).

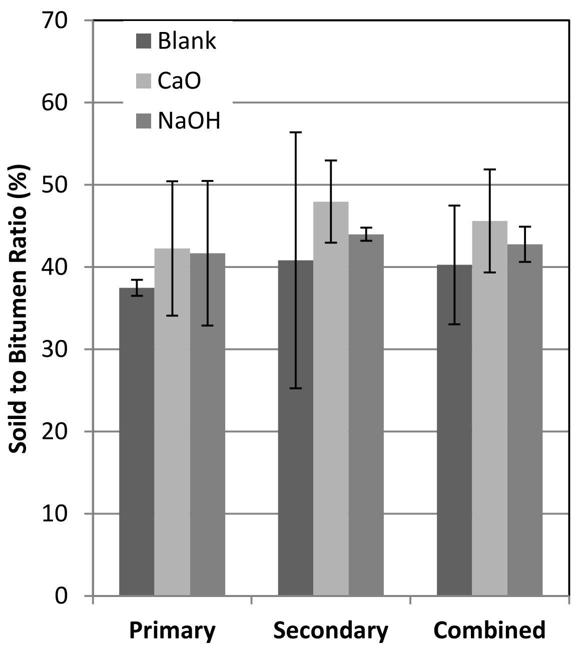

Figure 7. Effect of extraction additives on extraction froth quality.

Non-dispersed hydrometer test results are highlighted in Table 2. The dispersion at a particular particle size is calculated by dividing the percentage of fines in non-dispersed tests to the percentage fines in dispersed hydrometer tests. Data presented in Table 2 show that dispersion of fines greater than 10 μm grain size is insensitive to extraction process additive. However, fines under 10 μm grain size are significantly less dispersed for no additive or *CaO* additive (300 mg/kg-ore dosage) extraction tests.

Table 2. Non-Dispersed Hydrometer Test Results.

Extraction Additive	Particle Size (μm)				
	10	7	5	2	1
	Degree of Dispersion (% Dispersed Tailings)				
Blank	100	57	14	11	0
300 ppm *CaO*	100	75	14	11	10
300 ppm *NaOH*	100	100	100	81	28

Settling Characteristics

Tailings produced at a plant using CHWE process is a segregation tailings. Upon the deposition of the tailings to disposal pits the sand particles immediately settle while majority of the fines are carried to the tailings pond as thin slurry. The settling rate of these fines has impacts on the fluid fine tailings production and on the release water recovery. The rate of fine tailings sedimentation depends on the additives used in the extraction process.

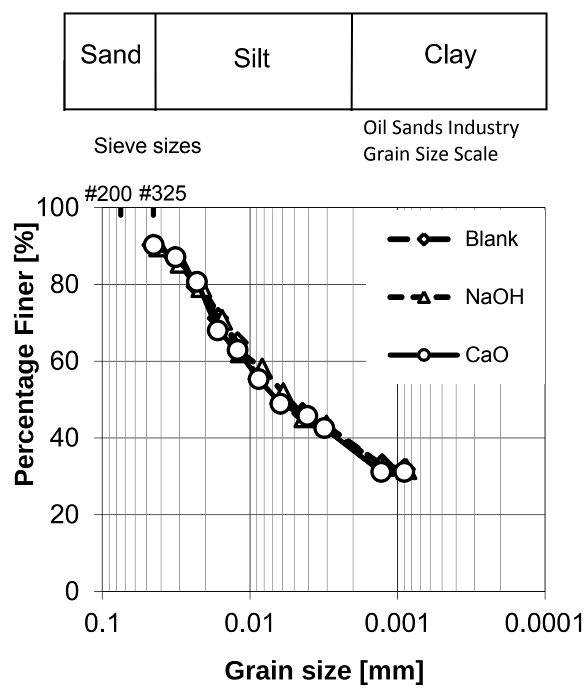

Figure 8. Dispersive hydrometer test results.

The solids content of fine tailings produced by *CaO* and *NaOH* extraction additives at different elapsed times are presented in Figure 10. Data presented in Figure 10 show that the fine tailings produced by using *CaO* additive (300 mg/kg-ore dosage) yields a clear water suspension after one day of deposition while that of with *NaOH* additive (300 mg/kg-ore dosage) yields a turbid suspension. The turbidity of the suspension is likely due the presence of higher percentage of dispersed particles size of less than 2 μm, which was also observed in non-dispersive hydrometer tests.

Segregation Boundary

Settling column tests were performed with 4 fines-sand mixture tailings at different solids, but identical (10%) fines, content; to determine the segregation boundary point of the extraction tailings. The four samples were filled into settling columns of 5.0 cm internal diameter cylinders to a height of 7.5 cm and left to sediment and consolidate for one week. After a week, the released water was siphoned-off, the sediment sliced into a number of layers and their solids contents were measured to calculate fines capture.

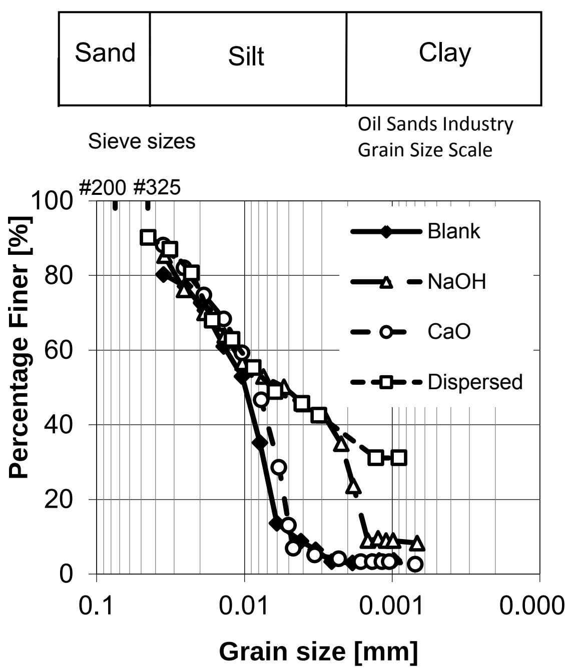

Figure 9. Comparison of dispersive and non-dispersive hydrometer test on fines produced by different additives.

Measures solids content profiles of the tailings produced by using *CaO* and *NaOH* as extraction additives at 300 ppm dosages are presented in Figure 11 and 12 respectively. The segregation boundary points of the fine-sand mixtures are derived from these plots as the solids contents corresponding to 95% fines capture. Data presented in these figures show that these tailings samples become nonsegregated at about 80% solids content.

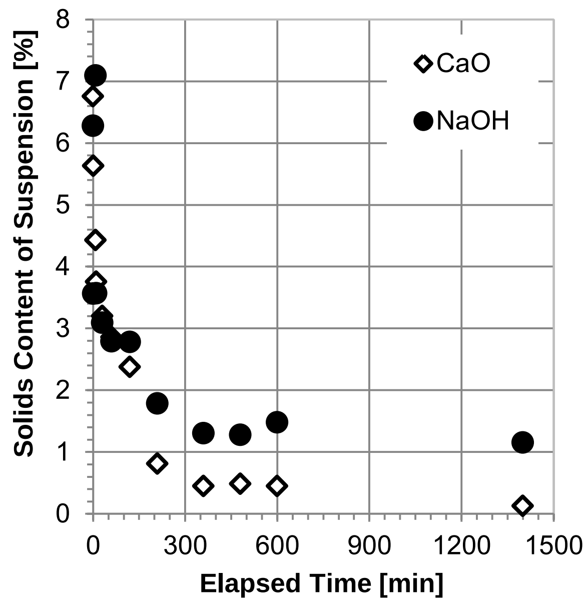

Figure 10. Solids content of fine tailings as a function of elapsed time.

Solids content profiles presented in Figures 11 and 12 also indicate that *NaOH* behaves as a better extraction additive to prevent segregation as it produces more fines since it causes lesser segregation than that of *CaO*. This observation also supported by the non-dispersed hydrometer test results presented in Figure 9 that use of *NaOH* as an extraction process additive produces more fines.

Data presented in Figures 9, 11 and 12 suggest that if *CaO* is used as an extraction process additive, NST material could be prepared from the blend of Cyclone Underflow, Thickener Underflow and MFT from the existing tailings ponds. The MFT added into the NST mixture would provide clay activity to form the yield stress in fines-water matrix when treated with *CaO* and prevent segregation.

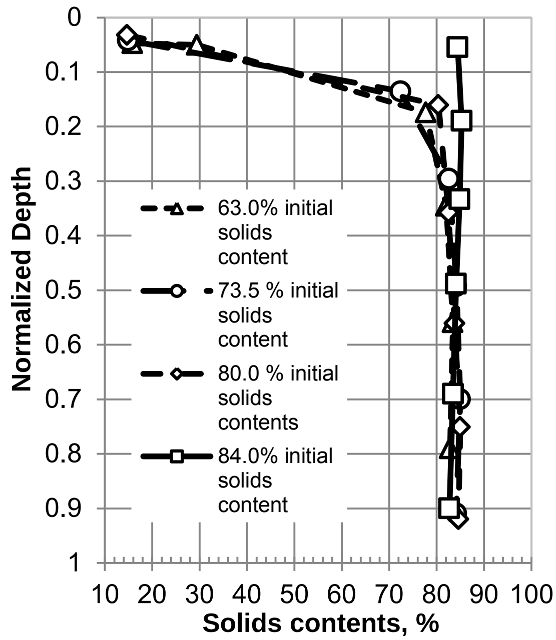

Figure 11. Solid content profiles of fine-sand mixture samples from *NaOH* extraction after one week of sedimentation and consolidation.

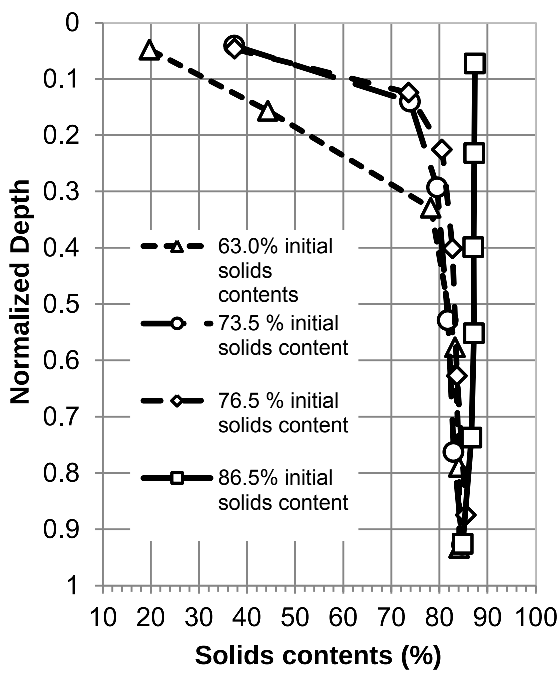

Figure 12. Solid content profiles of fine-sand mixture samples from *CaO* extraction after one week of sedimentation and consolidation.

NONSEGREGATING TAILINGS PRODUCTION

NST production using *CaO* and CO_2 additives has been studies on Albian Sands Muskeg River mine tailings material when the plant was operating with non-additive and *Na-Citrate* additive modes. Seven reports were produced summarizing these studies (Chalaturnyk et al, 2002; Scott et al, 2007). The same process was tested on Syncrude Canada's Aurora Mine tailings too (Apex Engineering Inc., 2005; Donahue et al, 2008). We think that sufficient laboratory bench scale data are available for the design, commissioning and execution of a field test. Further laboratory tests are needed to produce NST from the tailings produced by using *CaO* as an extraction additive; by treating the blend of Cyclone Underflow, Thickener Underflow and existing MFT with *CaO* or *CaO* and CO_2.

CONCLUSIONS

Existing trends in production of toxic MFT and increase in process water salinity problems of the oil sands plants could be rectified by using *CaO* as an extraction additive replacing *NaOH* used in existing plants. When *CaO* is used as an extraction process additive, a tailings stream is produced with reduced clay dispersion which reduces MFT production without harming bitumen extraction efficiency or fuel quality of bitumen. Implementation of the use of *CaO* as additives in bitumen extraction and NST production processes could provide results in reducing MFT inventor and process water salinity.

ACKNOWLEDGEMENTS

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PARAMETRIC STUDY OF THE CENTRIFUGATION OF OIL SAND MATURE FINE TAILINGS

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ABSTRACT

Centrifugation of polymer-treated mature fine tails is being tested as one method that can produce dry-stackable tailings. In this study various operating parameters were tested for their impact on centrifuge performance, defined for this case as solids content in the cake and centrate, solids capture, and cake yield strength.

These series of experiments have shown that centrifugation of flocculant-treated MFT can achieve almost complete solids capture and produce a cake with an initial yield strength of at least 2 kPa. Re-use of the centrate as dilution water for the MFT feed led to a slight increase in centrate solids content, but that trend was reversed if gypsum was also added. Centrate did not hurt bitumen recovery from an average ore.

The optimum conditions for centrifugation seem to be at centrifugal field of 3,000 g, dilution of the MFT feed with process water and with gypsum added as a pre-coagulant. From a technical view point the choice of flocculant and dose is a function of the field conditions and targets for cake and centrate quality.

INTRODUCTION

The Energy Resources Conservation Board Directive 074 delineates two performance criteria for the capture and disposal of fines from oil sand mining tailings: the portion of fines that must be captured in dedicated disposal areas and the undrained shear strength which that disposal area must achieve. There are many technologies that do or potentially can meet Directive 74, some require initial containment of the treated material (e.g. consolidated tailings, rim-ditching), and some do not (e.g. centrifuged tailings, thin-lift drying). Centrifugation has the advantages of producing a material that does not need containment, and water (centrate) that can be captured and recycled to extraction.

The objectives of this study were to 1) evaluate centrifugation performance when using different polymers at varied dosages, 2) to evaluate what effect the dilution water has on centrifuge performance, 3) evaluate the effects of recycling the centrate, and 4) to evaluate the effect that gypsum has when used in combination with the various polymers tested on centrifuge performance. Results in terms of cake strength and centrate quality are presented on tests using three polymers, with and without gypsum, three different dilution water chemistries, and with recycling centrate.

EXPERIMENTAL

Centrifugation

The centrifugation studies were conducted on a GEA 205 Westfalia centrifuge with a 10 cm radius bowl and a 112 mm pond depth. The bowl speed was typically set to 5250 rpm, or about 3000 g.

Figure 1. Photograph of GEA decanter centrifuge used in studies.

The rotating bowl develops a centrifugal field, usually expressed as a multiple of gravitation acceleration, given as follows:

$$RCF = \frac{\omega^2 r}{g}$$

where RCF is the relative centrifugal factor, g is acceleration due to gravity, r, is the bowl radius, and ω is the bowl angular velocity in radians. Figure 2 presents the bowl and scroll differential speeds of one of the centrifuge runs. The bowl speed determines the G-force, while the differential scroll speed determines the retention time of the solids in the centrifuge. The lower the differential speed the longer the retention time of the solids.

Centrifuge performance was usually evaluated against three interrelated outcomes: solids concentration in centrate, cake dryness, and fines capture. If solids contents of the centrate, cake and feed are known the solids capture can be calculated using:

$$Solids\,capture\,(\%) = \frac{S_{cake}}{S_{feed}} * \frac{(S_{feed} - S_{centrate})}{(S_{cake} - S_{centrate})} \times 100$$

where S is the weight-percent solids of the given stream.

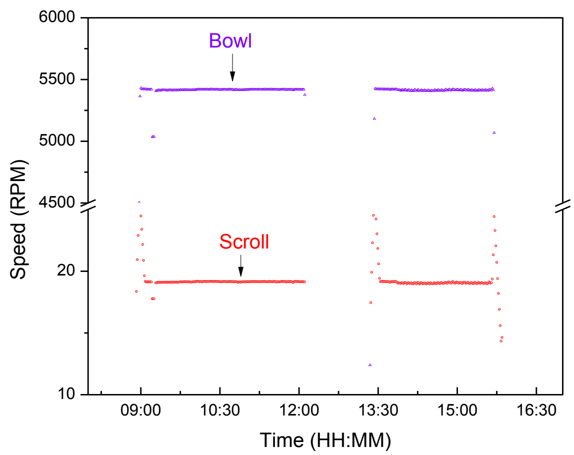

Figure 2. Decanter centrifuge bowl and scroll speeds for a typical run.

Polymer additions were made directly at the feed-tube inlet of the centrifuge. When gypsum was added, it was added to the diluted MFT prior to being fed to the centrifuge.

Material

Feed used in this study was mature fine tailings (MFT) from an operating company, 39-40 wt% solids, with 99.5% of the solids being less than 45 µm and 56.3% being less than 2 µm. Prior to being fed to the centrifuge the MFT was diluted to between 23 and 25 wt% solids with tap water unless otherwise noted.

Three different polymers were used in the studies, and have been designated A, B, and C.

Analyses

Analyses carried out on samples included bitumen content by Dean Stark, particle size distribution (PSD) of solids using a Micrometrics SediGraph III, and clay content by methylene blue titration or X-ray diffraction (XRD). Water content was determined by drying for 12 h at 105°C.

Cake yield strength was measured using a Haake Viscotester 550 viscometer equipped with a 4-blade vane tool.

OPTIMIZATION OF CENTRIFUGE PARAMETERS

Tests were carried out to determine the optimum centrifuge parameters for an ~25 wt% solids MFT feed. Figure 3 shows that while centrate solids content seemed independent of bowl speed, cake solids content increased linearly with increasing speed, as expected. A main bowl speed of 5250 rpm was chosen for subsequent experiments.

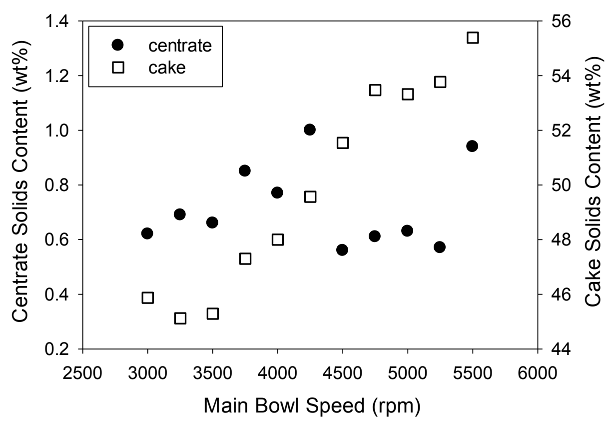

Figure 3. Centrate and cake solids content as a function of bowl speed.

A similar test was carried out for the scroll speed and it was found that centrate solids content was independent of scroll speed in the range tested while (again as expected), cake solids content decreased with increasing scroll speed (decreasing retention time). A value of 14.5 rpm was chosen for subsequent experiments.

Next, increasing the feed flow rate was found to decrease solids capture but increase cake solids content and (to a lesser extent) centrate solids content (Figure 4) so choice of feed flow would be the maximum possible that also met solids-capture specifications and centrate quality. For a centrate of ~1.4 wt% solids and 97% solids capture, the throughput was about 0.41 tonnes solid per hour.

An extended run (12 h over 2 days) showed that the centrifuge produces a consistent product: solids content in the cake varied by 2.5% (relative; actual values: 53.9 ± 1.3 wt%) and solids in centrate varied by 40% (relative; actual values: 1.04 ± 0.42 wt%). Cake yield strength was more problematic with values varying by as much as 25% over 2 days (e.g. 1.75 ± 0.43 kPa, n=29) although for other extended runs (same feed and run conditions) readings were within 15% (e.g. 1.76 ± 0.26 kPa, n=25). While there was also no real correlation between cake solids content and yield strength, there was between polymer dose and cake strength.

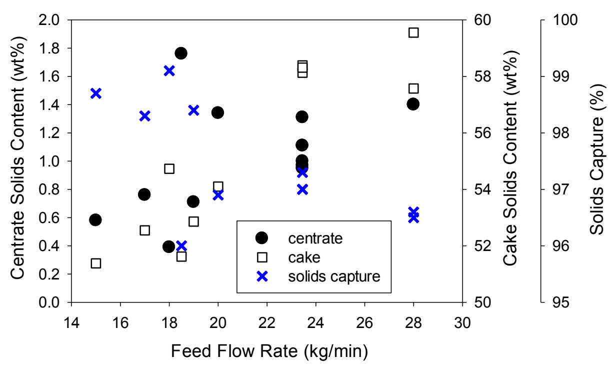

Figure 4. Centrate and cake solids contents and solids capture as a function of scroll speed.

EFFECT OF POLYMER AND DOSAGE

Results and Discussion

Increasing flocculant dose resulted in lower solids content in the centrate for all 3 polymers tested (Figure 5) but for polymer A, also resulted in lower solids content in the cake, perhaps indicative of larger floc sizes (Figure 6). Cake solids contents in B and C were relatively unchanged with increasing flocculant.

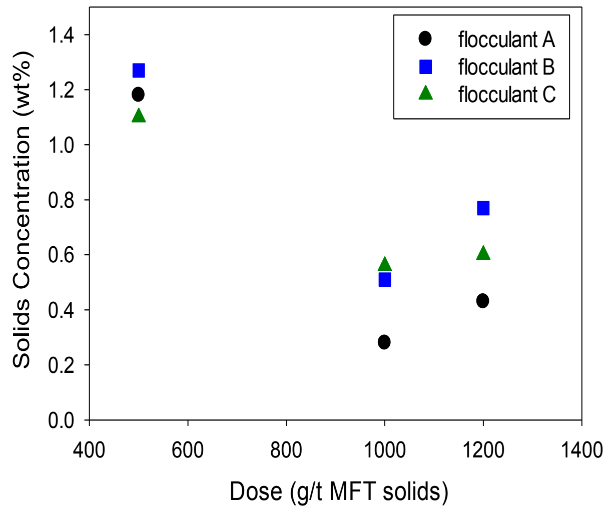

Figure 5. Effect of polymer dose and type on centrate solids content.

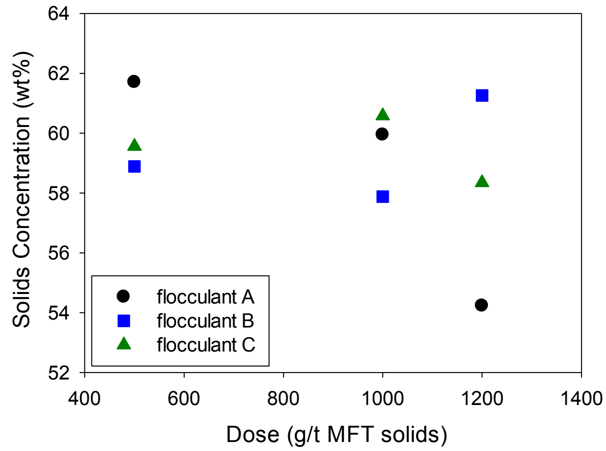

Figure 6. Effect of polymer dose and type on cake solids content.

As cake yield strength is more variable, 3 data points is insufficient to draw conclusions on effect of polymer dose: as can be seen in Figure 7 there is no apparent trend, except possibly that flocculant B showed an increase in yield strength with increasing dose. However, an extended run using flocculant B showed that it, at least, led to increased yield strength with increasing dose (Figure 8). Flocculants A and C were not run for extended times.

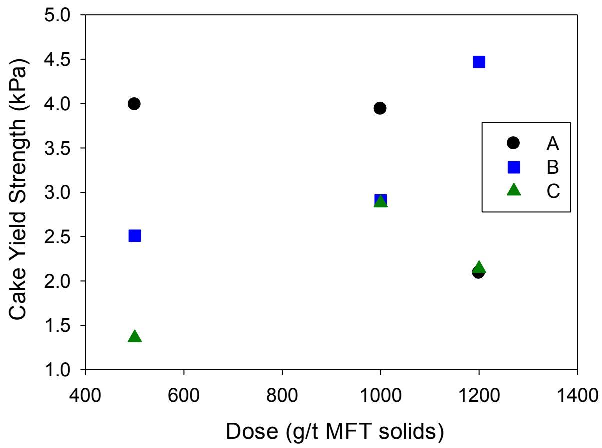

Figure 7. Effect of polymer dose on cake strength.

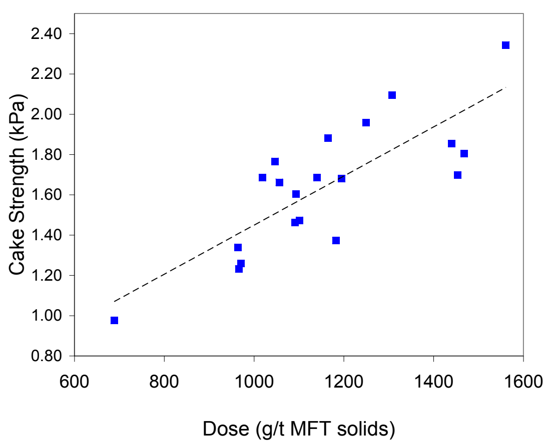

Figure 8. Effect of flocculant B on cake strength as a function of dose.

EFFECT OF DILUTION WATER

In this series of tests the MFT feed was diluted with either process water (PW), tap water, or tap water with 1000 mg/L added $NaHCO_3$, to see the impact of water chemistry on centrifuge performance.

Results and Discussion

Figure 9 and Figure 10 show the impact of MFT dilution water on centrate and cake solids content respectively. Process water seemed to give the best centrate for all three flocculants and either the best cake (polymer A), or at least no different than tap water or tap water with bicarbonate.

The largest difference in water chemistry between PW-diluted MFT and the other two was the higher total dissolved salts (TDS) in PW-diluted MFT (Table 1). As one would expect for a buffered system the pH was the same in all three, Ca^{2+} was similar for all three and HCO_3^- was similar for PW- and tap water plus $NaHCO_3$-diluted MFT. One could speculate about the higher salt content reducing the double-layer on particle surfaces and hence increasing the efficacy of the flocculant, but it is only speculation without more data.

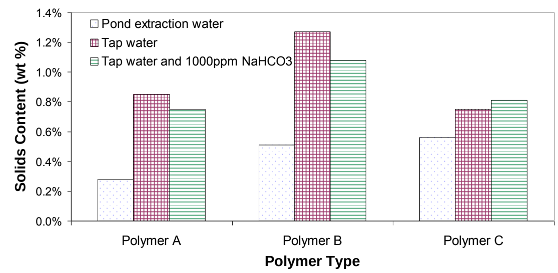

Figure 9. Impact of feed dilution-water chemistry on centrate solids content.

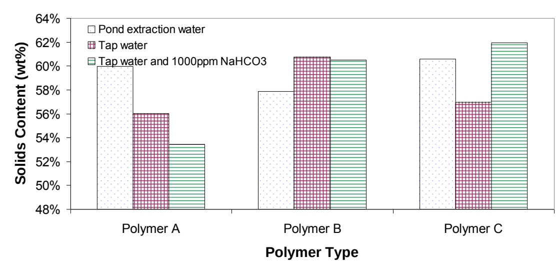

Figure 10. Impact of feed dilution-water chemistry on cake solids content.

EFFECT OF GYPSUM ADDITION

For this series, the use of gypsum to pre-coagulate the clays, as an aid to the flocculant, was tested. For all runs the gypsum was added to the diluted MFT before the MFT was fed to the centrifuge, at which point the flocculant was added. For all experiments flocculant was added at a dose of 1000 g/t MFT solids.

Table 1. Water chemistry of MFT feed diluted with process water (PW), tap water or tap water plus $NaHCO_3$.

Ion	Ion Concentration (mg/L) and pH		
	PW-diluted MFT	Tap+ $NaHCO_3$-diluted MFT	Tap-water diluted MFT
Ca^{2+}	6 ± 2	4 ± 2	6 ± 3
Na^{+}	990 ± 50	710 ± 160	590 ± 80
Cl^{-}	560 ± 40	250 ± 30	330 ± 70
SO_4^{2-}	250 ± 60	30 ± 20	20 ± 10
HCO_3^{-}	1410 ± 125	1470 ± 460	1030 ± 90
TDS	3290 ± 100	2520 ± 600	2010 ± 220
pH	8.6 ± 0.1	8.5 ± 0.1	8.4 ± 0.1

Results and Discussion

There seemed to be an optimum gypsum dose for cake solids content when used with flocculants A and B (Figure 11), which also seemed to correspond to a peak in cake yield strength (Figure 12) although why this should be is uncertain. For centrate quality there was a definite improvement with increasing gypsum (Figure 13) as seems more intuitive if Ca^{2+} is coagulating the smaller particles, making them more available for the flocculant. Solids capture varied between 97% and 99% with the higher capture rates being at the higher gypsum doses.

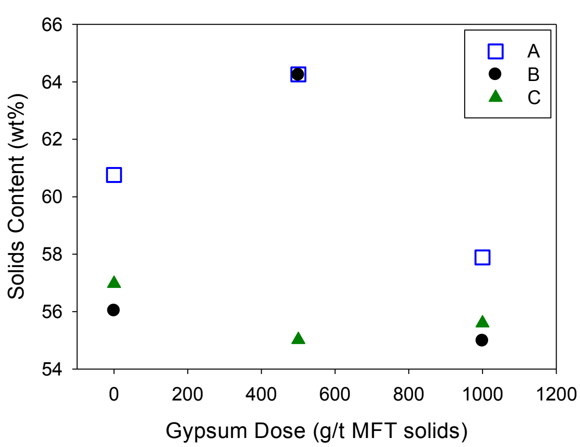

Figure 11. Impact of added gypsum on cake solids content; 1000 g/t polymer added.

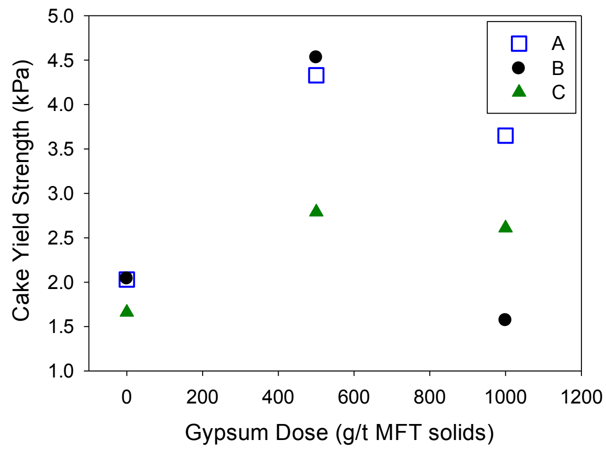

Figure 12. Impact of added gypsum on cake yield strength; 1000 g/t polymer added.

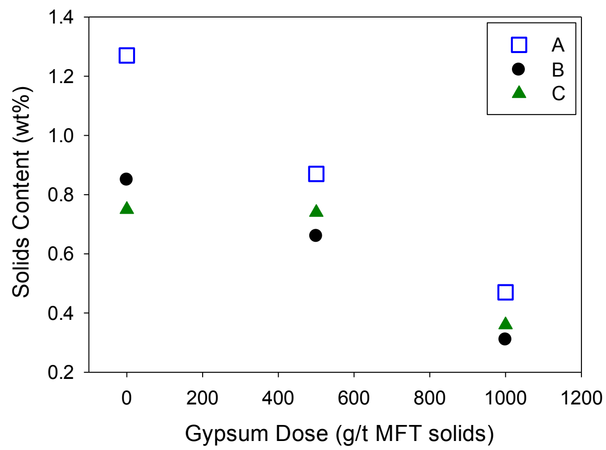

Figure 13. Impact of added gypsum on centrate solids content; 1000 g/t polymer added.

CENTRATE RECYCLE

The purpose of this experiment was to investigate the effect of centrate recycling, for use in MFT dilution, on centrifuge throughput and solids capture, as there is concern that if centrate were to be used as a recycle stream there would be a detrimental accumulation of ions and ultra-fines (< 2 µm particles). All runs used 1000 g polymer B per tonne of MFT solids (1000 ppm,w). Three scenarios were tested: straight recycle of centrate, with the centrate of a run, being used to dilute the MFT feed for the next run, the subsequent centrate used for diluting feed for the third run, and so on;

centrate recycle in which 1000 ppm,w gypsum was added to the first cycle's MFT, and centrate recycle during which 1000 ppm,w gypsum was added to the MFT feed for every cycle.

Results and Discussion

Table 2 shows that while the cake properties (solids content and yield strength) appeared unaffected by recycling centrate, the centrate solids content increased slightly (but was not significant statistically) with each iteration, and solids-capture decreased slightly with each iteration (statistically significant).

As one would expect, the aqueous ion concentrations increased approximately linearly with each cycle, from the initial dilution of MFT pore water with tap water in the first run, with the fourth cycle's chemistry being close to that of the MFT pore water (21 mg/L Ca^{2+}, 36 mg/L K^{+}, 23 mg/L Mg^{2+}, 809 mg/L Na^{+}, 583 mg/L Cl^{-}, 40 mg/L SO_4^{2-}, 1315 mg/L HCO_3^{-}, with a pH of 8.2) (Figure 14).

Table 3 data illustrate centrifuge performance with the addition of 1000 ppm,w gypsum to the MFT feed for every cycle. Compared to the no-gypsum case, overall solids capture increased, with a concomitant decrease in centrate solids content (both statistically significant). Cake solids content was not significantly different and there was also no trend in cake yield strength.

Table 2. Centrifuge Performance With Recycle of Centrate (Flocculant Only).

Cycle No.	Cake solids Content (wt%)	Centrate Solids Content (wt%)	Cake Strength (kPa)	Solids Capture (%)
1	57.9	0.94	1.75	98.2
2	56.9	0.97	1.55	98.1
3	58.8	1.48	1.63	96.8
4	59.0	1.67	2.98	95.9

The better performance for the centrifuge came at the cost of poorer water quality (Figure 15). While Ca^{2+} only doubled in concentration to 30 mg/L (indicating likely calcite precipitation and/or clay ion-exchange was occurring), SO_4^{2-} increased about ten-fold to 350 mg/L.

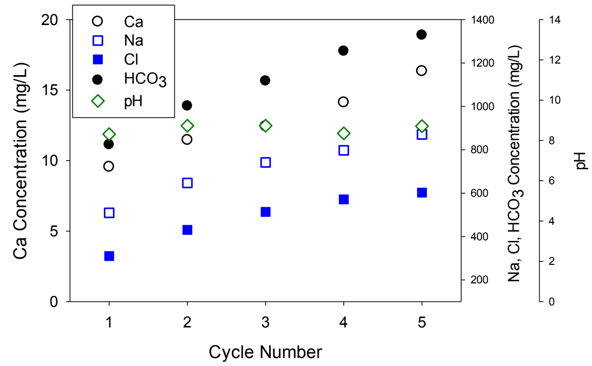

Figure 14. Centrate water chemistry as a function of cycle number, flocculant only

To see if one could get the benefit of better centrifuge performance but with lower impact on water quality, a series was run in which the first cycle's MFT feed was treated with 1000 ppm,w gypsum, but thereafter no gypsum was added.

Table 3. Centrifuge Performance With Recycle of Centrate for Gypsum-Treated MFT.

Cycle No.	Cake solids Content (wt%)	Centrate Solids Content (wt%)	Cake Strength (kPa)	Solids Capture (%)
1	56.0	0.86	1.99	98.3
2	58.0	0.87	2.89	98.4
3	57.6	0.51	3.30	99.0
4	58.1	0.31	2.75	99.4
5	58.8	0.34	2.27	99.3

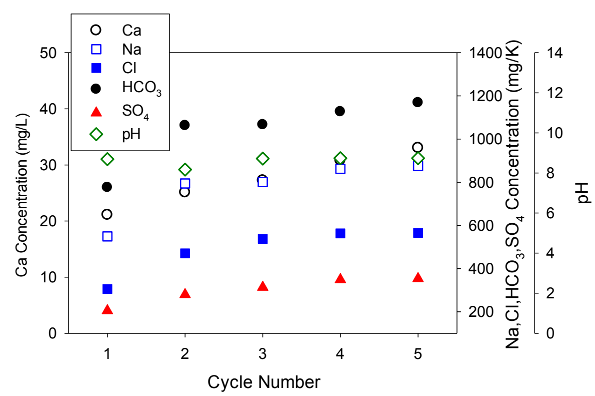

Figure 15. Centrate water chemistry as a function of cycle number for MFT feed with added gypsum.

The data in Table 4 show that the performance was similar to no-gypsum centrate recycle: cake solids and yield strength were not affected by recycle but solids capture decreased and solids content in centrate increased. Water chemistry was also similar to the no-gypsum case.

The particle size of the final centrate was determined for all 3 series of runs and it was found that 100% of the centrate solids were <2 µm (compared to 56% for the MFT feed). When the solids were analyzed various differences were observed (Table 5): the overall clay content increased and the inter-layered smectites increased significantly while quartz and illite decreased significantly in the centrate (although the decrease in illite was not observed in other runs). As the increase of inter-layered smectites could create an issue of the centrate was used in extraction, the centrate was tested in a bench-scale extraction of ore (described in the next section).

Table 4. Centrifuge Performance With Recycle of Centrate for MFT Treated With Gypsum Only For First Cycle

Cycle No.	Cake solids Content (wt%)	Centrate Solids Content (wt%)	Cake Strength (kPa)	Solids Capture (%)
1	55.8	0.70	1.62	98.7
2	56.9	0.74	1.54	98.6
3	57.2	0.94	1.49	98.1
4	57.9	0.90	1.66	98.1
5	57.3	1.18	1.58	97.7
6	57.2	0.87	----	98.4
7	55.6	1.92	----	96.1

IMPACT OF CENTRATE ON BITUMEN EXTRACTION

Many factors affect bitumen extraction efficiency in the warm-water extraction process. In particular, anything that impedes bitumen displacement from the sand or the attachment of bitumen to air bubbles could reduce bitumen recovery. Two of those factors are the amount of fines in the recycle water used to create the ore slurry and the presence of any chemical that could affect surface properties of the bitumen, mineral, or air bubbles in the conditioned ore slurry. One concern with recycling centrate from centrifuged MFT to extraction is the amount and type of fines in the centrate and the possible presence of residual polymer. To test for any effects on bitumen extraction, a centrate from a centrifuge run was used as the dilution water in a bench-scale extraction test.

Table 5. XRD Analysis Of Centrate Solids Compared to MFT Feed. The 3 Centrate Numbers are From The Final Centrate From Each of the 3 Tests.

Mineral	Mineral Concentration (wt% of total solids)	
	Centrate	MFT
Anatase (TiO_2)	1.9, 4.9, 3.5	1.3
Calcite ($CaCO_3$)	2.0,1.1,1.2	1.5
Kaolinite	28, 33, 29	32
Kaolinite + Smectite	21, 28, 32	6
Illite	1, 7, 1	16
Illite + Smectite	38, 17, 26	5
Pyrite (FeS_2)	0.4, 0.2, 0.2	0.6
Quartz (SiO_2)	4, 9, 7	33
Rutile (TiO_2)	0.4, 0.1, 0.1	0.1
Siderite ($FeCO_3$)	2.2, 0.09, 0.03	3.0
Total clay in solids	89 ,85, 88	60

[a] ND means not detected

Procedure

A bench-scale extraction unit (BEU) was used to evaluate the impact of a centrate containing 1.0 wt% fines on bitumen recovery. The warm-water (50°C) Syncrude extraction protocol, with no process aid, was chosen to run these tests, as being more sensitive to factors that may hurt recovery. A Fort McMurray oil sand ore containing average bitumen (11.6 wt%), low fines (6.5 wt% fines) and a medium amount of degraded bitumen was homogenized, split into several 600-gram samples, and kept refrigerated for a couple of days prior to the tests.

Results and Discussion

The results from the extraction tests are given in Table 6. Extraction efficiency and froth quality for the run using centrate were the same (within error) as the run using tap water. The centrate used had about 1 wt% fines, enriched in some inter-layer smectites (Table 7), which, because of their swelling potential could lead to problems, but at the amount in the centrate tested, no effect was seen.

The degraded bitumen content in the tap-water extraction froth was 55 vol%, as compared to 40 vol% in the centrate–run froth. One may have expected an increase in degraded bitumen due to the fines in the centrate, but it is possible that the higher alkalinity of the centrate as compared to the tap water had a greater impact in reducing degraded bitumen.

Table 6. Bitumen Recovery and Froth Quality From Extraction Tests Comparing Centrate and Tap Water.

Slurry Water	Ore Composition (wt%; normalised)			Primary Recov. (%)	Total Recov. (%)	Primary Froth Quality [a]
	Bit.	Min.	Water			
Tap water	11.5	86.2	2.3	37.4	84	4.3
Centrate	11.8	85.6	2.5	38.8	82	4.4

[a] Froth quality is defined as (bitumen content/solids content in the froth

Table 7. XRD Analysis of Centrifuge Centrate Solids and Middlings from Extraction Test.

Mineral	Mineral Concentration (wt% of total solids)	
	Centrate	Middlings
Anatase (TiO_2)	3.54	0.57
Calcite ($CaCO_3$)	1.21	1.15
Kaolinite	28.67	37.54
Kaolinite + Smectite	32.31	ND[a]
Illite	1.04	21.60
Illite + Smectite	25.85	ND
Pyrite (FeS_2)	0.22	1.51
Quartz (SiO_2)	7.03	35.23
Rutile (TiO_2)	0.10	1.84
Siderite ($FeCO_3$)	0.03	0.20
Zircon ($ZrSiO_4$)	ND	0.38
Total clay in solids	87.9	59.1

[a] ND means not detected

CONCLUSIONS

These series of experiments have shown that centrifugation of flocculant-treated MFT can achieve almost complete solids capture and produce a cake with an initial yield strength of at least 2 kPa.

Flocculants responded differently to changing conditions (such as dilution water chemistry) so, as is self-evident, the choice of flocculant depends on the field conditions (e.g. water to be used for dilution) and priorities (e.g. cake solids content or centrate solids content). Adding gypsum as a pre-coagulant reduced solids content in the centrate and gave a marginally improved solids capture. However, if the centrate was to be used as dilution water for the MFT, the water chemistry would quickly build up a high sulphate concentration. If instead the centrate was to be sent for use in extraction, and not re-cycled for feed dilution, its chemistry would not be significantly different from the bulk of recycle water. It was shown that use of centrate that had been cycled through the centrifuge at least four times, did not reduce bitumen recovery from an average ore. This despite the solids being all less than 2 µm and enriched in inter-layer smectite clays.

The optimum conditions for centrifugation seem to be at centrifugal field of 3,000 g, dilution of the MFT feed with process water and with gypsum added as a pre-coagulant. From a technical view point the choice of flocculant and dose is a function of the field conditions and targets for cake and centrate quality.

ACKNOWLEDGEMENTS

This research was funded by Natural Resources Canada through the Panel for Research and Development (PERD). The authors are grateful to the oil sand mining companies for the supply of tailings and ore used in these studies. We also thank Michelle Morin and Paulina Wong who performed water chemistry and particle size analyses and Behnam Namsechi who carried out the extraction tests and XRD analyses.

Session 8

Tailings Transport and Deposition

PIPELINE TRANSPORT OF THICKENED OIL SANDS TAILINGS

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ABSTRACT

The environmental performance of oil sands operations can be greatly improved by using gravity thickeners to recover water from the tailings. Once the water is recovered and returned to the process, the thickened mixtures are transported in slurry pipelines, often several kilometers away, to the tailings management area. In comparison with conventional oil sands tailings, thickened tailings may have a very high resistance to flow in pipelines. To complicate matters, this resistance is reduced over time when the thickened tailings are exposed to shear in pumps, pipes, and other equipment.

It is important that the designers of tailings transport systems have a good understanding of the complex flow behavior of thickened oil sand tailings mixtures. In 2011, Total E&P Canada and the Saskatchewan Research Council conducted a study on the rheological behavior of thickened oil sands tailings. The flow of thickened oil sand tailings mixtures was studied using equipment of industrial size. The performance of a large centrifugal slurry pump was determined and changes in the mixtures' resistance to flow were measured against the shearing energy provided by the pump. Also, the pipe flow behavior of thickened tailings samples was examined using a 265 mm diameter pipeline and the experimental results were compared with flow model predictions. Finally, tests were carried out to determine the flow conditions needed to avoid sand deposition in thickened tailings transport pipelines.

INTRODUCTION

Thickened tailings (TT) technology is an integrated system that is used widely in mineral processing. In the oil sands industry, this technology has been put to use at Shell Canada's Athabasca Oil Sands Project. Several other oil sand operators including Total E&P Canada, Imperial Oil, and CNRL are considering TT technology in their upcoming mining projects. In the oil sands extraction process, large volumes of water and large amounts of heat are required. The use of thickeners will enable significant recovery of both water and heat thus allowing more effective water reuse and improved energy efficiency for the overall extraction process. Also, sustainable oil sands mining and extraction operations require that the clays and other fine particles be captured and stored in a manner that allows effective land reclamation. Thickeners provide flexibility in handling the wide range of conditions (flow rates, solids concentrations, sand to fines ratios) that occur in oil sand operations. This flexibility is a key to effective fines management in the oil sands industry.

In developing thickened tailings technology, it is important not to overlook the pipeline transport system needed to move the tailings from the plant to their final resting area. Most slurry pipelines operate in the turbulent flow regime where eddy mixing forces are available to keep particles suspended. In a slurry pipeline transporting conventional oil sand tailings the sand particles are carried by a "fluid" composed of clays and other very finely divided solids dispersed in water. These carrier fluids are dilute mixtures that tend not have any significant yield stress. While the presence of the sand particles causes a complication, the turbulent flow characteristics of conventional slurries are well understood and tools such as the SRC slurry pipeline flow model (1, 2, 3) are available to assist with system design.

Oil sand thickened tailings are not conventional slurries. Their carrier fluids contain relatively high concentrations of clays and they tend to have significant yield stresses. Chemicals that are added to improve the flocculation process tend to augment this yield stress, sometimes increasing it by an order of magnitude. To complicate matters further, the yield stress augmentation effect tends to be gradually reversed during exposure to shear in process equipment (4).

If a mixture has a relatively high yield stress then laminar conditions can be expected to persist over the range of velocities normally associated with slurry pipeline operation (1, 5). In the absence of the powerful mixing provided by turbulent eddies, there is concern that sand and other coarse particles will settle out and accumulate in the pipeline. Even if the mixture has a yield stress that is sufficient to prevent settling under quiescent conditions, settling has been observed when the mixture is exposed to shear in a pipeline (6).

Before transport systems can be designed with confidence, we need to develop a better understanding of the behaviour of oil sand thickened tailings mixtures. In the study reported here, the flow characteristics of thickened oil sand tailings were studied using equipment of industrial scale at the SRC's Pipe Flow Technology Centre.

FLOW BEHAVIOUR OF SLURRIES

Pipeline Friction for Turbulent Flows

For macroscopically steady state operation of a pipeline of constant cross sectional area (Figure 1) transporting a constant density mixture, the following equation applies:

$$\frac{dP}{dz} + \rho g \frac{dh}{dz} + \frac{4\tau_w}{D} = 0 \qquad [1]$$

The first term of Equation 1 is the pressure gradient and the second is the gravitational effect. In the third term, the friction loss is expressed in terms of the average wall shear stress, τ_w.

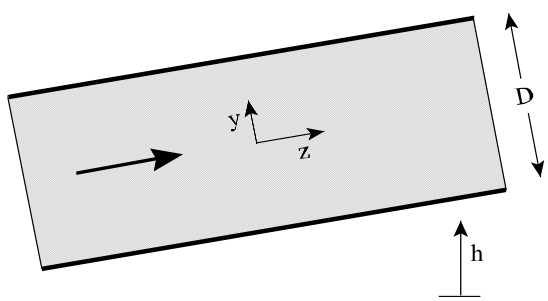

Figure 1. Pipe section schematic

It is convenient to use a friction factor to estimate the wall shear stress for the pipeline flow of fluids. The Fanning fluid friction factor f_f is defined by the following equation:

$$\tau_w = 0.5\, f_f\, V^2 \rho_f \qquad [2]$$

SRC's slurry pipeline flow model has found wide application in the design of conventional pipelines transporting settling slurries in the turbulent flow regime. The model captures the following salient characteristics of the flow:

a) asymmetrical concentration and velocity distributions
b) a kinematic (velocity-sensitive) frictional contribution to the turbulent flow pressure loss that increases strongly with increasing particle concentration but decreases somewhat with increasing particle size
c) a Coulombic (velocity-insensitive) frictional contribution that increases with both particle concentration and particle size

While the rheology of thickened oil sand tailings is much more complex than that of conventional oil sand tailings mixtures, the turbulent pipe flow behaviour is, in some respects, simpler. Because of the viscous nature of the carrier fluid, the concentration and velocity distributions for thickened tailings mixtures are expected to be nearly symmetrical and the Coulombic contribution to friction is usually small enough to be neglected. We are left with the kinematic contribution to friction.

SRC's slurry pipelining model uses separate friction factors for the fluid and the particles to estimate the overall kinematic friction for the pipeline flow of slurries (1, 2). The following equation is used:

$$\tau_w = 0.5V^2\left(\rho_f f_f + \rho_s f_s\right) \qquad [3]$$

The fluid friction factor depends on the pipeline Reynolds number and wall roughness. It is calculated using a standard correlation (7).

For turbulent pipe flow, the contribution of the particles to the kinematic friction cannot be modelled adequately using a fluid friction approach. Instead the SRC model uses a particle kinematic friction correlation of the type proposed by Shook and co-workers (1) and further developed by D. Gillies (8):

$$f_s = \lambda^{1.25}\left[b_0 \quad b_1 \ln\left(d^+\right)\right] \qquad [4]$$

In the above correlation, λ is the linear concentration which is the ratio of the average particle diameter to the average distance between particle surfaces. The dimensionless particle diameter d^+ is proportional to the particle diameter and inversely proportional to the thickness of the viscous sublayer at the pipe wall:

$$d^+ = 5d/\delta \quad [5]$$

The viscous sublayer thickness is estimated using carrier fluid properties:

$$\delta = \frac{5\mu_f}{\rho_f V\sqrt{f_f/2}} \quad [6]$$

The kinematic friction calculations in the SRC model assume that the carrier fluid is essentially Newtonian. Wilson and Thomas (9) showed that an important drag reduction effect occurs when fluids with yield stresses are transported in the turbulent flow regime. With θ defined as the stress ratio τ_w/τ_y, the fluid friction factor can be determined from the following equation:

$$\frac{1}{2.5}\sqrt{\frac{2}{f}} = \ln\left(\frac{\rho^{1/2}\tau_y{}^{1/2}D}{\mu_p}\right) + \ln\left(\frac{\theta^{1/2}(\theta-1)}{\theta+1}\right) + \frac{11.6}{2.5\theta} + \left[\ln\left(\frac{\theta-1}{\theta}\right) + \frac{\theta+0.5}{\theta^2}\right] \quad [7]$$

Because thickened tailings tend to have significant yield stresses, the Wilson-Thomas effect should be considered in pipelines designed to transport those mixtures.

Centrifugal Pump Performance

For a mixture of density ρ, the pump's total dynamic head can be determined from the pressures and fluid velocities at the inlet and outlet of the pump using the following equation:

$$H_{TD} = \frac{P_{outlet} - P_{inlet}}{\rho g} + \frac{V_{outlet}^2 - V_{inlet}^2}{2g} \quad [8]$$

Pump efficiency is determined from T, the torque applied to the pump shaft, and the rotational speed ω as follows:

$$\eta = \frac{Q\,\rho\,g\,H_{TD}}{\omega\,T} \quad [9]$$

When assessing centrifugal pump performance, it is convenient to compare the head and efficiency obtained with a slurry to those obtained with water at the same flow rate and pump speed. Centrifugal pump performance ratios are defined as follows:

$$\text{Head Ratio} = \left(H_{TD}\right)_{mixture} / \left(H_{TD}\right)_{water} \quad [10]$$

$$\text{Efficiency Ratio} = \eta_{mixture} / \eta_{water} \quad [11]$$

Slurry performance ratios are known to depend on the pump Reynolds number (10) which is calculated using the pump impeller diameter D_i as follows:

$$Re_{pump} = \frac{D_i{}^2\omega\,\rho}{\mu_p} \quad [12]$$

Shear Induced Rheology Reduction

Treinen and Cooke (4) used a concentric cylinder viscometer to track the rate at which flocculated slurries experience rheology reduction as a result of shearing energy input. The cumulative energy input per unit volume of slurry after time t_1 is:

$$E = \frac{1}{\pi\left(R_2^2 - R_1^2\right)L}\int_{t=0}^{t=t_1} T\omega\,dt \quad [13]$$

In the above equation, T is the torque required to rotate the viscometer cylinder. The equation applies only if there is shear across the entire gap between the inner cylinder of radius R_1 and the outer cylinder of radius R_2.

In a horizontal pipe section with area A and length Z the cumulative shearing energy input per unit volume of slurry flowing at a velocity V is:

$$E = \int_{t=0}^{t=Z/V} \frac{AV(-dP)\,dt}{A\,dz} = V\int_{t=0}^{t=Z/V}\left(\frac{dP}{dz}\right)dt \quad [14]$$

The above equation is appropriate if the flow is turbulent. If the flow is laminar and the slurry has a yield stress then only the material located in an annular space near the pipe wall will experience shear. The bulk of the slurry will travel in a central core that experiences essentially no shear. In this case, the material near the pipe wall will experience a relatively high rate of energy induced rheology reduction while the slurry travelling in the core will not experience any rheology reduction.

With pump bearing losses ignored, the total energy input to a unit volume of a slurry passing through a centrifugal pump is:

$$E_{total} = \frac{T\,\omega\,\Delta t}{volume} = \frac{T\omega}{Q} \quad [15]$$

It is the losses or inefficiencies in a pump that contribute the shearing energy required for rheology reduction (4). Therefore the shearing energy applied to a unit volume of slurry as it passes through a centrifugal pump is:

$$E = \frac{T\omega}{Q}\left(1-\eta\right) \qquad [16]$$

Slurries with Yield Stresses

The Bingham plastic fluid model is often used to describe the flow behaviour of slurries containing significant concentrations of clay. In terms of radial position r in a pipe, the Bingham model is:

$$-\frac{dv_z}{dr} = \frac{\tau_{rz} - \tau_y}{\mu_p} \qquad [17]$$

In the above equation v_z and τ_{rz} are the local velocity and shear stress for flow in the downstream (z) direction. The coefficients τ_y and μ_p are the Bingham yield stress and plastic viscosity. Integrating Equation 17 over the pipe cross section gives the Buckingham equation for pipeline flow of Bingham fluids:

$$\frac{8V}{D} = \frac{\tau_w}{\mu_p}\left(1 - \frac{4\tau_y}{3\tau_w} + \frac{\tau_y^4}{3\tau_w^4}\right) \qquad [18]$$

The Buckingham equation is applicable if the flow is laminar and the fluid rheological properties are constant over the entire pipe cross section.

Schaan and coworkers (11) found that, for laminar flow applications, the effective viscosity of sand mixtures increased with the concentration of the sand particles as follows:

$$\mu = \mu_f\left(1 + 2.5C + 0.16\ ^{2}\right) \qquad [19]$$

In the above correlation μ_f is the viscosity of the carrier fluid.

EXPERIMENTAL PROCEDURES

Flotation tailings from CNRL's Horizon oil sands operation were trucked to SRC's Pipe Flow Technology Centre in Saskatoon and used as the feedstock in experiments carried out in a 1.5 m diameter FLSmidth deep cone thickener. The flotation tailings were diluted to 10% solids by mass and then fed into the thickener at a solids flux of 0.54 to 0.60 T/m^2/hr. The flocculent, HyChem AF 309, was fed at a rate of 200 g/T of dry solids. Two batches of thickener underflow, each approximately 8 m^3 in volume, were produced. The properties of each batch are given in Table 1.

Table 1: Thickened tailings properties

Batch	A	B
Total solids mass fraction	0.536	0.508
Total solids volume fraction	0.303	0.280
Sand volume fraction	0.135	0.105
Sand linear concentration	2.0	1.6
Sand to fines ratio	0.8	0.6
Sand mass median particle diameter (mm)	0.09	0.09
Initial vane yield stress (Pa)	267	135

A vane shearing device was used to obtain an indication of the yield stress of the underflow material before it was exposed to any significant degree of shear and to track changes during exposure to shear. The vane apparatus (six vanes, 11 mm radius, 16 mm high) was rotated at 0.01 radians per second.

Data of the type reported by Treinen and Cooke (4) were obtained by testing samples of the thickener underflow in a Haake concentric cylinder viscometer. The samples were sheared in the narrow gap between the rotating inner cylinder (radius = 19.0 mm) and stationary outer cylinder (radius = 21.7 mm). The viscometer was operated at a rotational speed of 31.4 rad/s and the spindle torque was monitored against time.

Using a low shear progressing cavity pump, each batch of underflow material was transferred to the pump sump on SRC's 265 mm pipeline flow loop (Figure 2) where a centrifugal slurry pump (Warman 10/8 AH) is used to provide flow. At the start of each test, the valve at the bottom of the sump was opened to allow the slurry to enter then pump and travel through some 100 m of piping before reaching the drain line. After the piping had been completely filled, the valves were switched thus closing the drain line and allowing the slurry to return directly to the pump inlet. The operation continued until the tailings had completed several circuits through the pump and the pipe loop. The pump's performance was monitored by measuring the torque applied to the shaft, the pump speed, the pressure rise and the flow rate. The slurry temperature was maintained by circulating a chilling fluid through the annulus of a pipe-over-pipe heat exchanger. Samples were withdrawn to track changes in the vane yield stress during each pass through the centrifugal pump.

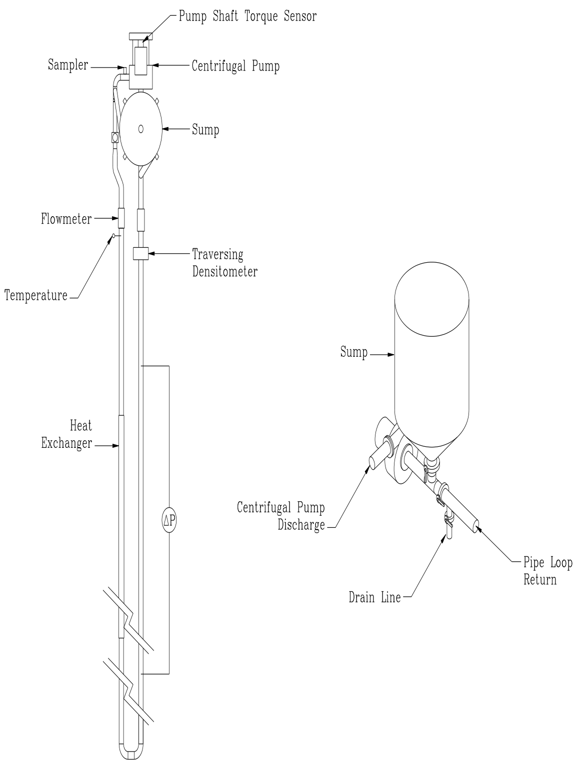

Figure 2. SRC 265 mm pipeline flow loop

The vane yield stress decreased significantly over time during the initial stage of testing. After experiencing a large number of passes through the centrifugal pump, the mixture's yield stress eventually stopped decreasing. The next stage of testing was then carried out by measuring the pipeline pressure gradient over a wide range of flow rates. The test program was completed by using a gamma ray absorption device (12) to determine how the slurry density varied with vertical position in the pipeline. One set of density scans was carried out while the pipeline operated in the turbulent flow regime and a second set was carried out under laminar flow conditions.

EXPERIMENTAL RESULTS

A significant amount of yield stress reduction occurred when the thickened tailings samples were exposed to shear. As shown in Figure 3, similar yield stress reduction rates were obtained in the pipeline circuit and in the laboratory viscometer. Equation 13 was used to determine the energy input by the viscometer. The pipe flow was laminar during these tests so the majority of the shear thinning would take place in the centrifugal pump. With the contribution of the pipe section ignored, Equation 16 was used to estimate the energy input rate in the pipeline circuit. It was found that the yield stress of the thickened tailings could be reduced to half of the original value by applying a cumulative shearing energy of roughly 1500 kJ per cubic metre of slurry.

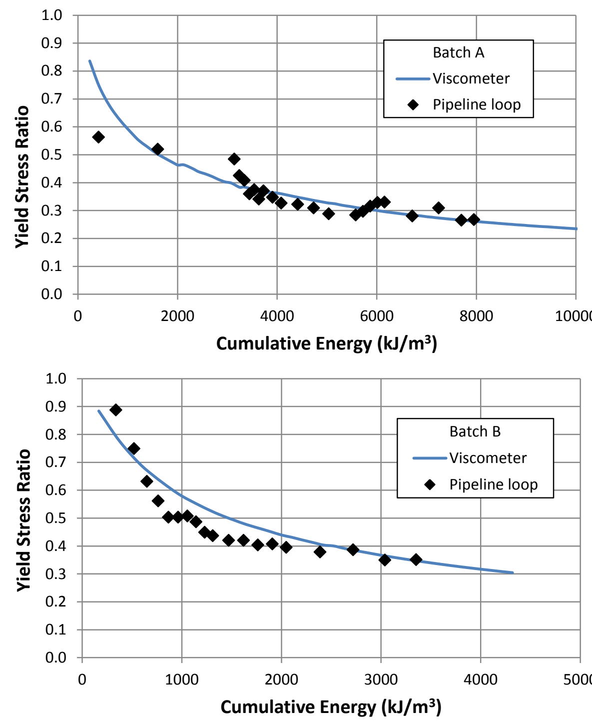

Figure 3. A comparison of thickened tailings yield stress reduction rates during shear in a concentric cylinder viscometer and in a large pipeline flow system.

Pump performance is shown in Figures 4 and 5. The pump produced disappointing results during the first three passes for batch A and during the first pass for batch B. After this initial troublesome phase, the performance improved suddenly to match the values predicted by the pump manufacturer. The reason for the poor initial performance is not known. Further investigation is required to determine whether it was an artifact of the pipe loop operation at SRC or an indication of a more general problem with centrifugal pumps on thickened tailings service.

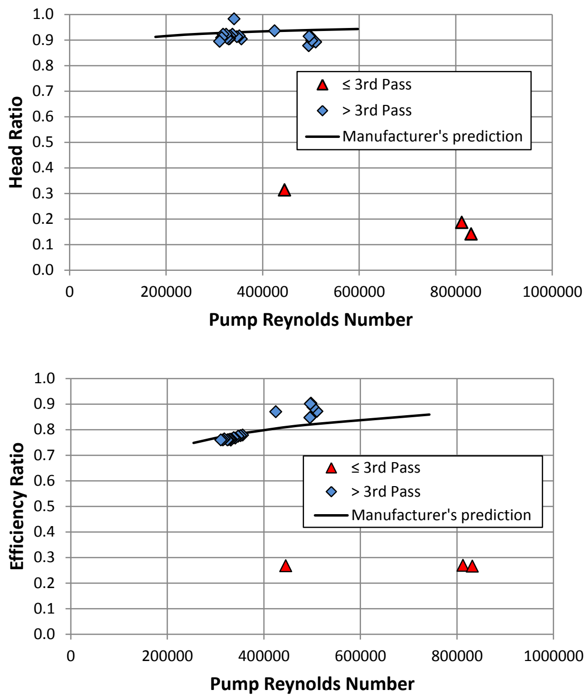

Figure 4. A comparison of experimental and predicted centrifugal pump performance ratios for thickened oil sand tailings (Batch A).

The pipeline pressure gradients for well sheared thickened tailings mixtures are shown in Figure 6. When the flow rate is low the pipeline velocity increases rapidly with a small increase in the pressure gradient. This behaviour is consistent with the laminar flow of mixtures with significant yield stresses. At higher flow rates, an increase in velocity requires an exponential increase in the pipeline pressure gradient. This behaviour is indicative of a flow dominated by fluid turbulence. For each of the mixtures, the transition from laminar to turbulent flow appears to occur at a velocity of approximately 2.3 m/s.

Densitometer scans were carried out with the 265 mm pipeline operating in the laminar regime at a bulk velocity of 2.0 m/s and then repeated in the turbulent flow regime with the velocity set at 3.0 m/s. The results are shown in Figure 7. For each of the samples, there was almost no variation in density with vertical position when the flow conditions were turbulent. For the tests carried out in the laminar flow regime, there was evidence of sand settling with a relatively high density near the bottom of the pipe. It is difficult to assess the impact of this observation. There is a fixed volume of sand available in SRC's recirculating pipeline flow loop and when sand settles out along the bottom of the pipe. There is a reduced concentration available in the mixture flowing above the settled layer. Due to compensating effects, the overall resistance to flow does not increase to any significant degree when settling occurs. An actual field pipeline with its continuous supply of sand could behave differently. In such an operation, settling could lead to an accumulation problem accompanied by an increased resistance to flow.

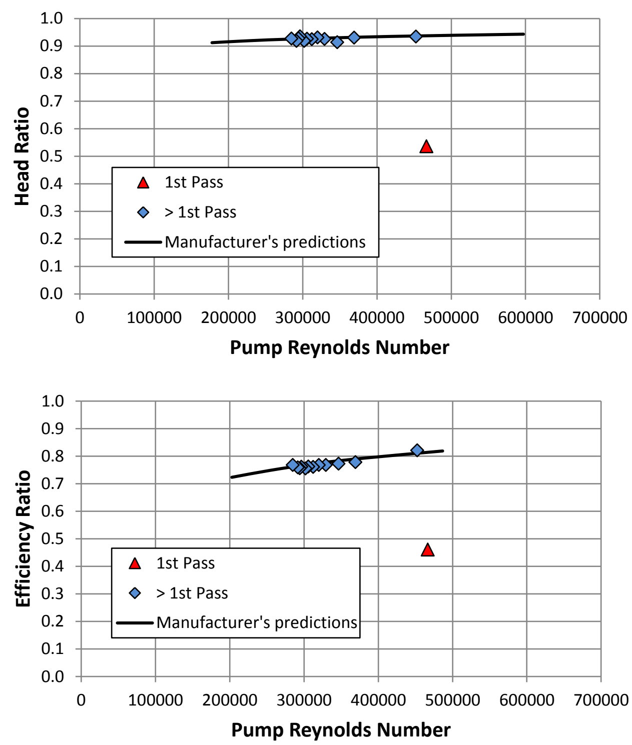

Figure 5. A comparison of experimental and predicted centrifugal pump performance ratios for thickened oil sand tailings (Batch B).

To determine the rheology of the well-sheared mixtures, after several hundred passes through the loop and the centrifugal pump, samples were removed from the 265 mm pipeline loop for testing in a 25 mm diameter vertical pipe viscometer. Equation 18 was used to assess pipe viscometer data obtained while operating in the laminar regime and the coefficients shown in Table 2 were obtained.

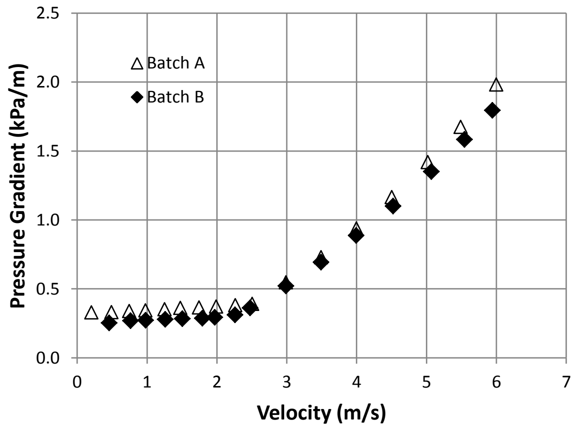

Figure 6. Pressure losses for well-sheared thickened oil sand tailings mixtures flowing in a 265 mm horizontal pipeline.

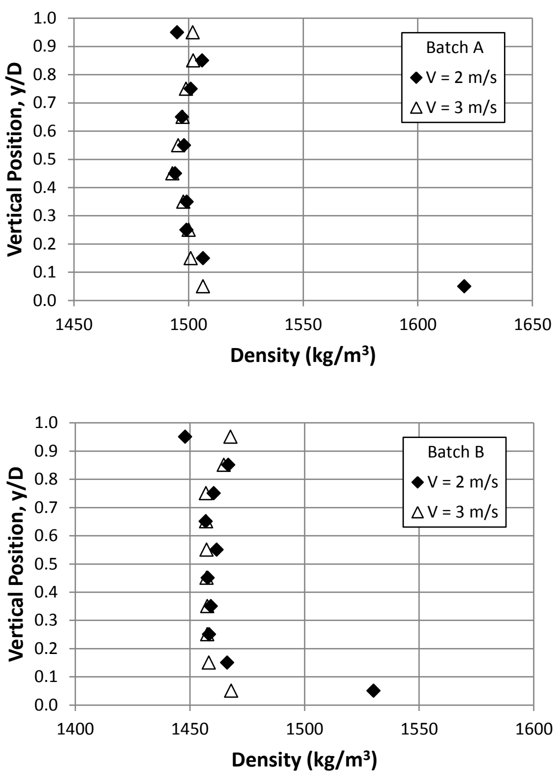

Figure 7. A comparison of the density distributions in laminar flow (2 m/s) and turbulent flow (3 m/s) for thickened oil sand tailings slurries in a 265 mm pipeline.

Table 2: Properties of the Thickened Tailings Slurries After Prolonged Shear in the 265 Mm Pipeline.

Thickened tailings batch	A	B
Temperature (°C)	24	25
Bingham yield stress (Pa)	15	16
Plastic viscosity (Pa·s)	0.034	0.028
Carrier fluid density (kg/m^3)	1320	1320
Carrier fluid plastic viscosity* (Pa·s)	0.017	0.017

* Equation 18 was used to infer carrier fluid plastic viscosities from the measured values for the total mixture.

PIPE FLOW ANALYSIS

An analysis was carried out to determine whether data obtained from tests in a small pipe viscometer could be used to predict friction losses for large pipelines. The following assumptions were made:

1. Turbulent flow pipeline pressure losses are due to the kinematic effects captured in Equation 3.
2. The plastic viscosity of the carrier fluid can be used in place of the Newtonian fluid viscosity in Equations 6 and 18 and in the correlation used to determine the fluid friction factors.

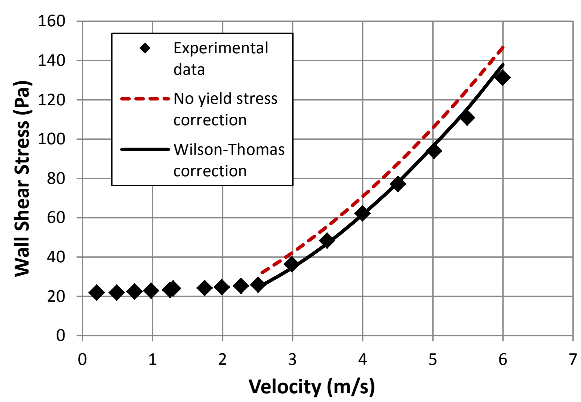

Figure 8. A comparison of experimental and predicted friction losses for thickened oil sand tailings Batch A flowing in the 265 mm pipeline.

Using these assumptions, friction loss estimates were obtained and they are compared with actual pipeline flow results in Figure 8. The experimental friction losses are for the first batch of thickened tailings flowing in SRC's 265 mm pipeline. It is apparent that Equation 3 over-predicts the pipeline friction losses when no adjustment is made to compensate for the yield stress. When the Wilson-Thomas drag reduction effect is applied to reduce the fluid friction factor, we see that Equation 3 provides reasonably accurate predictions of the pipeline friction.

Figure 9 compares the turbulent flow friction loss data for the second batch of thickened tailings with losses predicted using Equation 3 (with the Wilson-Thomas correction applied). Again, the method provides reasonably close predictions of the turbulent flow friction losses.

Also plotted in Figure 9 are the laminar flow predictions obtained using the Buckingham equation and the Bingham coefficients obtained from the pipe viscometer. The laminar flow plot intersects the turbulent flow predictions at a velocity of approximately 2.3 m/s. This, according to Shook and co-workers (1), represents the laminar-turbulent transition velocity for the flow. Using a similar analysis for unsheared thickened tailings (yield stress = 267 Pa), one finds that the laminar-turbulent transition occurs at a velocity in excess of 10 m/s.

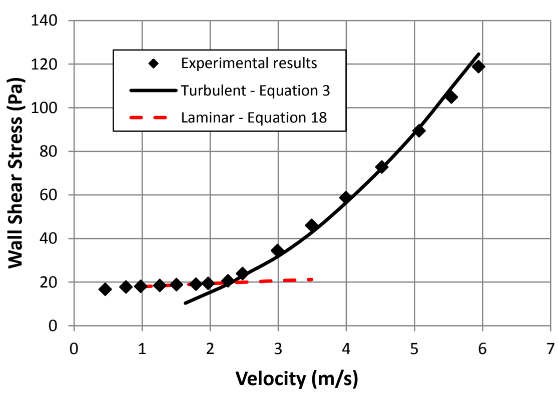

Figure 9. A comparison of experimental and predicted friction losses for thickened oil sand tailings Batch B flowing in the 265 mm pipeline.

It is important to note that when a pipeline is operated in the laminar flow regime, sand accumulation problems could result in pipeline friction losses that are substantially higher than those predicted using the Buckingham equation. Therefore, while the Buckingham equation is useful for establishing the transition velocity, it should not be used to estimate laminar flow pressure losses for thickened tailings mixtures.

KEY FINDINGS

1. The thickened oil sand tailings mixtures had very high resistance to flow as they exited the thickener. The yield stress, as determined by a vane shear device, was 135 Pa for a mixture containing 28% solids by volume and 267 Pa for a mixture containing 30% solids by volume.
2. The resistance to flow could be reduced by applying a very large amount of mechanical shear. Approximately 1500 kJ of shearing energy was required to reduce the yield stress of one cubic metre of thickened tailings to 50% of its initial value.
3. In tests carried out in a 265 mm diameter horizontal pipeline, sand deposition occurred when the pipe flow regime was laminar. No sand deposition was detected when the flow was turbulent.
4. Extremely high pipeline velocities, in excess of 10 m/s, would be required to maintain turbulent flow conditions if a thickened tailings mixtures with a yield stress of 267 Pa was injected into a pipeline without first receiving any treatment to reduce the yield stress.
5. After prolonged exposure to shear, the yield stress was reduced to approximately 15 Pa and turbulent flow could be established by providing a minimum pipeline velocity of 2.3 m/s.
6. The turbulent flow pipeline pressure losses could be predicted satisfactorily using SRC's kinematic friction model for Newtonian slurries with the fluid friction component modified by the Wilson-Thomas fluid drag reduction effect.
7. For the majority of the test conditions, the centrifugal pump's performance was excellent with head ratios in excess of 90%. There was a problem with very poor pump performance during the initial phase of each test and further work is required to establish whether this was an indication of a significant problem or just an artifact of SRC's pipe flow loop configuration.
8. Compared to results obtained by other operators, the resistance to flow observed in this study is quite high. This could be due to the relatively high flocculent dosage, the type of clays present in the flotation tailings, or the chemistry of the process water. These factors are being investigated further in a follow-up study.

NOMENCLATURE

A	Pipe cross sectional area (m^2)
C	Particle volume fraction (-)
d	Particle diameter (m)
d+	Dimensionless particle diameter (-)
D	Pipe diameter (m)
D_i	Pump impeller diameter (m)
E	Cumulative energy input (J/m^3 of slurry)
f	Fanning friction factor (-)
g	Acceleration of gravity (m/s^2)
h	Elevation (m)
H_{TD}	Total dynamic pump head (m)
P	Pressure (Pa)
Q	Volumetric flow rate (m^3/s)
r	Radial position (m)
R	Viscometer spindle radius (m)
t	Time (s)
T	Torque (N·m)
v	Local velocity (m/s)
V	Pipeline bulk velocity, Q/A (m/s)
y	Distance measured upward from the bottom of the pipe
z	Distance measured in the downstream direction (m)
δ	Thickness of the viscous sublayer (m)
η	Pump efficiency (-)
λ	Linear concentration (-)
μ	Viscosity (Pa·s)
μ_p	Plastic viscosity (Pa·s)
ρ	Density (kg/m^3)
τ	Shear stress (Pa)
τ_y	Bingham yield stress (Pa)
ω	Angular velocity (rad/s)

Subscripts:

f	refers to fluid
r	refers to radial direction
s	refers to particles
w	refers to pipe wall
z	refers to downstream direction

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OPTIMIZING THE CAPTURE OF OIL SAND FINES IN SAND BEACH AREA

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ABSTRACT

The objective of this study was to evaluate the concept of storing fines in the voids of coarse sand when oil sand tailings are deposited in a sand beach area. In particular, the study will investigate the assumption that up to 50% of fines by mass in total feed can be captured in the sand beach upon initial placement. A further objective was to evaluate the benefit of adding more mature fine tailings to the coarse sand pump box and to determine the impact on sand beach properties.

To simulate oil sand slurry disposal, actual oil sand tailing components were used in the study. Three groups of coarse sands were obtained from hydrocyclone underflow tailings. Mature fine tailings and thickened tailings were used as fine sources. The slurry mixtures were prepared based on their respective targets of feed solids concentration, fines to fines plus water ratio and sand to fines ratio. Laboratory settling cylinder tests were conducted to study fines captured by sedimentation under static conditions. Slurry mixtures were prepared and tested in an 8 m long flume to simulate fines captured in beach deposition. After draining the free flow runoff, the beach was allowed to settle for 24 hours. Feed, beach and runoff samples were analyzed. The percentage of fines captured in beach and net fines captured in beach were inferred from the analytical results.

INTRODUCTION

To reduce the fines tailings inventory in oil sands mining development, fines captured in sand beach is a promising concept which consists of spiking fines by mixing Mature Fines Tailings (MFT) or Thickened Tailings (TT) into coarse sand slurries. If done properly, it will maximize the fines captured within the inter-particular porosity of the sand particles. It has been estimated that coarse sand can capture in the Sand Beach Area (SBA) more than 50% of feed fines (1) but the detailed mechanism and the key parameters governing this capture were not yet fully understood. It was decided to conduct additional experimental studies to optimize and better understand the critical parameters involved.

The concept of fines captured in SBA was first tested by Syncrude (2) in the early 1990s'. A number of large scale tests were conducted to study fines capture mechanisms from coarse tailings deposition and MFT spiked coarse tailings. A field demonstration test was conducted at the Syncrude site to prove the concept (3). W.G. Miller et al (4) summarized the flume test results obtained by Syncrude and OSLO in the early 1990s', and concluded that flume deposition tests could provide a reliable and effective method to simulate the tailings beaching behavior.

In recent years, the reduction of fines tailings inventory has become a critical issue, especially following the ERCB's Directive 074 issued in 2009. The concept of beach fines capture has been revisited and further developed.

The objective of this study was to evaluate the concept of storing fines in the voids of coarse sand in oil sands tailings deposits, specifically:

1. To check the assumption that up to 50% of fines in total feed can be captured in the sand beach upon initial placement;
2. To evaluate the effect of feed fines to fines plus water ratio (F/(F+W)) and solids concentration (Cs) on fines captured in coarse sand;
3. To evaluate the effect on sand beach properties of increased addition of MFT to the coarse sand pump box.

EXPERIMENTAL METHODS

A rectangular cross-section flume was used in the study. The dimensions of the flume are 0.25 m (W) x 0.5 m (H) x 8.0 m (L). Figures 1 and 2 show the flume testing apparatus and the schematic locations of different measuring devices.

Figure 1. Flume Testing Apparatus Used in the Project.

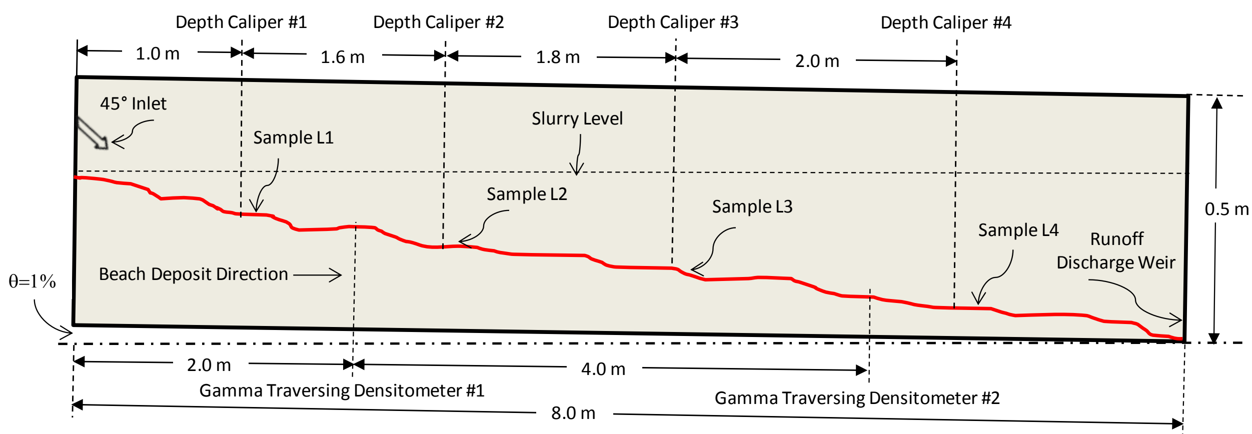

Figure 2. Schematic Locations of Measuring Devices in the Flume.

Four digital calipers were mounted 1.0, 2.6, 4.4 and 6.4 m from the flume inlet and used to record the depth of slurry or sand beach. Two traversing gamma-ray densitometers were mounted on the flume and used to monitor and measure the vertical solids concentrations. The densitometers were located 2.0 m and 6.0 m from the flume inlet respectively. Two low energy 100 mCi (3.7 GBq) Cesium-137 sources supplied the gamma ray radiation in the experiments.

Coarse sands were produced from hydrocyclone underflow tailings during extraction pilot testing conducted at SRC in 2009. The coarse sand was grouped as high grade (HG), medium grade (MG) and low grade (LG) samples based on bitumen content in the original ore. Each group of sand was blended and its sand to fines ratio (SFR_{44}) was adjusted to match target values by removing fines from the blend. MFT used in the study was acquired from Syncrude's Aurora tailings pond. TT was generated from a thickener operation having a SFR_{44} value of 0.8 (referred to as "TT0.8"). Table 1 shows the properties of all raw materials used in the project. The analyses of the relevant properties (SFR_{44}, Cs and F/(F+W)) for each raw material were repeated several times during laboratory and flume tests to update the mass balance calculation.

Settling Cylinder Tests

Eighteen settling cylinder tests were conducted prior to the flume tests. This was to obtain the relationship between feed F/(F+W) ratio and the percentage of fines captured by coarse sands under static conditions.

The feed mixture properties (SFR_{44}, Cs and F/(F+W)) and quantities for each test were controlled. All raw materials used in the laboratory tests were the same as those to be used in the flume tests. A known amount of MFT or TT0.8 was mixed with a known amount of coarse sand and process water to form a 2.1 L test sample. The sample was mixed thoroughly and a 100 mL sub-sample was withdrawn for feed analysis. The remaining 2 L sample was then transferred into a 2 L volumetric cylinder and allowed to settle for 24 hours. When settling started, the mixture separated into three layers (Figure 3) as light liquid S1 (mainly water with little fines), heavy liquid S2 (water and fines) and segregated mixture S3 (mainly coarse sand with fines and water). In an actual beach deposition process, S1 and S2 would flow to a pond as runoff. Therefore the main focus of the settling cylinder test was on layer S3 in which fines and water were captured by sedimentation of coarse material.

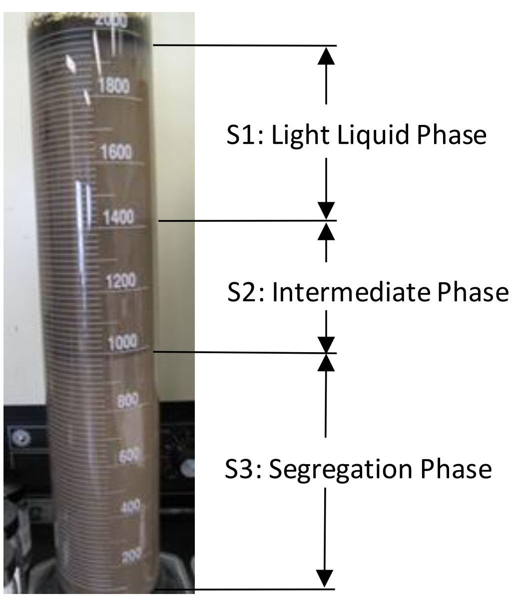

Figure 3. Schematic of the Layer Definitions in the Settling Cylinder.

The volume of each layer (S1, S2 and S3) was recorded after the sample had settled for 24 hours. Each layer was removed from the 2 L cylinder and analyzed for Cs and SFR_{44}. A field vane tester was used to measure the in-situ shear strength of layer S3. The quantity of fines captured was determined from the analytical results.

Tailings Flume Tests

Each flume test was started by preparing 600-800 L of tailing mixture in a 1.5 m^3 tank equipped with a 20 hp mixer and two sets of impellers. A known amount of coarse sand (LG, MG or HG), fines (MFT or TT0.8) and process water were added and mixed in the tank. The quantity of each component was controlled to meet the mass balance specifications of Cs, SFR_{44} and F/(F+W) for the mixture. To avoid inadequate mixing and/or entrained air during sample discharging, only 400 L was loaded into the flume. Flocculent solution (0.05% w/w) was prepared and added as specified.

The flume was always set at 1% slope to mimic the slope of SBA and help to drain the runoff. The homogenized mixture was discharged into the flume via a 0.025 m diameter pipe at 100 L/min and 45° angle to the flume bottom. Four 125 mL feed samples were taken during the discharge.

Immediately after the slurry was discharged into the flume, the level of the material contained in the flume was measured at four locations by four depth calipers. Two slurry vertical concentration profiles were determined by two traversing gamma densitometers. The vertical concentration profile measurements were repeated 2 hours and 24 hours after discharge.

The mixture was allowed to settle for 1 hour before runoff was drained by slowly opening the rubber plugs on the weir located at the end of the flume (Figure 4). The runoff was collected in separate drum(s) for weight and sample analysis.

After draining the runoff, the level of beach deposit was measured at twelve locations to determine the average beach shape and volume in the flume. After the beach had settled for 24 hours, eight beach samples were taken from four locations (L1, L2, L3 and L4) and at two depths (D1 and D2) for each location. This was achieved by inserting a 103 mm ID thin wall pipe into the sand beach to isolate the core sample from surrounding materials, followed by removing approximately half of the core sample from top and bottom layers respectively Figure 5). The eight beach samples, along with the samples from the feed and runoff, were analyzed for solids concentration, particle size distributions and other properties.

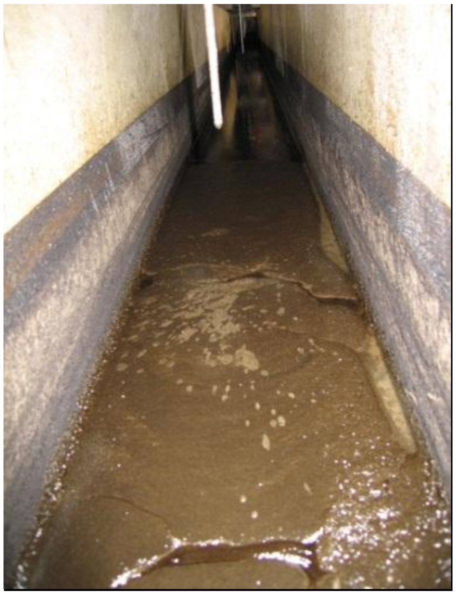

Figure 4. Beach after Draining Runoff and Sampling.

Figure 5. Rubber Plugs on Weir Board at Flume End.

RESULTS AND DISCUSSION

Feed and beach deposit properties were determined from sample analysis results. Weighted average of Cs, SFR_{44} and F/(F+W) values were determined for feed, beach and runoff. The following terms and definitions are used in the data analysis:

- Percentage of fines captured in beach (% w/w) = weight of fines in beach/ weight of fines in feed x 100%
- Net fines captured in beach (kg/m³) = weight of fines in beach / volume of beach
- Feed volume captured in beach (% v/v) = volume of beach / volume of feed x 100%
- Feed volume drained as runoff (% v/v) = volume of runoff / volume of feed x 100%

Settling Cylinder Test Results

High Grade Sand (HG) in Settling Cylinder

Table 2 shows that HG sand alone (test A3) was able to capture about 44% of feed fines in S3, with an initial feed F/(F+W) of 6.5%. When fines (either MFT or TT) were added to HG sand, the percentage of fines captured was increased to 53% and 60% respectively as shown in tests A6 and A8.

When extra MFT was added, the feed F/(F+W) was further increased to 16.4% (test A10), but the percentage of fines captured was decreased due to an increased amount of fines suspended in S1 and S2, and lost as runoff. Test A14 shows an extreme case where too much fines (MFT) were added. The much higher feed F/(F+W) value of 25.6% resulted in a non-settling slurry such that no sedimentation was observed in 24 hours. In terms of net fines captured in S3, the highest value obtained was 112 kg/m³ (test A10) from an initial feed F/(F+W) value of 16.4%.

Medium Grade Sand (MG) in Settling Cylinder

Table 3 shows that MG sand alone was able to capture about 70% of feed fines (test A2). For MG sand, the initial value of feed F/(F+W) which gave the highest final S3 F/(F+W) value and net fines capture in S3 was found again to be close to 16% (test A9).

Low Grade Sand (LG) in Settling Cylinder

Table 4 shows that LG sand alone was able to capture about 47% of feed fines in S3 (test A1). LG sand has a low sand to fines ratio due to its high fines content, the feed F/(F+W) was already high at 14% for a given 51% total solids concentration. The percentage of fines captured was increased only slightly when feed F/(F+W) ratio was increased (test A4), indicating the potential for fines capture by LG sand is limited.

For the three groups of coarse sands tested in settling cylinder, it seems that the lowest value of initial feed F/(F+W) required to reach an optimum final S3 F/(F+W) was close to 15-16%.

Tailings Flume Test Results

Each oil sands tailings flume test was started by preparing the feed in a 1.5 m³ mixing tank with a known amount of coarse sand, fines and process water.

High Grade Sand (HG) in Flume

Table 5 shows that HG sand alone was able to capture over 45% of fines from feed (test C1). Its runoff was relatively clean with 1.1% solids. When MFT was added to HG sand (test D1), the feed F/(F+W) was increased to 13.0%. The percentage of fines captured increased to 55.0%. Although some fines from MFT were lost in runoff as its solids concentration increased to 16.3%, still more fines stayed on the beach as the net fines captured in beach increased to 108.3 kg fines per m³ of beach.

By spiking with extra MFT (test D7), the feed F/(F+W) was increased to 15.2%, the percentage of fines captured remained at 53.6% and the amount of fines lost to runoff was unchanged. The optimum value of feed F/(F+W) required to optimize the capture was between 13 and 15%, slightly lower than the optimum observed in the cylinder tests.

Adding 400 gpT (grams/tonne of dry solids) flocculent reduced the runoff solids concentration and retained more fines in the beach, thereby increasing the percentage of fines captured (test D8). However the impact of flocculation on the long term development of beach shear strength needs to be evaluated.

Medium Grade Sand (MG) in Flume

Table 6 shows that MG sand alone was able to capture 56% fines from the feed (test C2A). The runoff had some solids but at a low concentration. Adding MFT and TT increased the feed F/(F+W) and increased the percentage of fines captured (tests D2 and D4). Some fines were lost in the runoff as its concentration increased, however the net fines captured increased to more than 110 kg fines per m³ of beach.

If extra MFT was added to push the feed F/(F+W) over 15% (test D6), the percentage of fines captured was reduced and more fines were lost in the runoff as its concentration increased to 37.9%, and the net fines captured reduced to 94.9 kg fines per m³ of beach. This was probably due to the increasing carrier rheology such that more fines were suspended in the runoff, instead of settling into the voids of coarse sand.

Similar to the case of HG sand in flume (test D8), adding flocculent to MG sand mixture increased the percentage of fines captured and net fines captured, and reduced the fines lost in the runoff. However its impact on beach shear strength needs to be evaluated.

In the case of MG sand, the optimum value of feed F/(F+W) required to optimize the capture was close to 13%, slightly lower than the optimum observed in the cylinder tests but consistent with the case of HG sand.

Low Grade Sand (LG) in Flume

Table 7 shows LG sand alone was able to capture 53% of fines from feed in the beach (test C3). Its runoff has a solids concentration of 7.8%, higher than the runoff of HG and MG sand. When MFT was added into the LG sand (test D3), the feed F/(F+W) was increased to 12.4%. Although fines lost in the runoff increased to 11.5%, the net fines captured was still increased to 95.7 kg fines per m³ of beach.

Because LG sand had a low SFR_{44}, its capability to capture fines was limited. The percentage of fines captured was actually slightly reduced by adding MFT (test D3).

Effect of Feed F/(F+W)

For the three types of coarse sand (HG, MG, LG) examined, it was found that the beach F/(F+W) and the percentage of fines captured in beach are governed by the feed F/(F+W) ratio. As shown in Figures 6 and 7, when the feed F/(F+W) was increased to 13-15% w/w, the segregated phase S3 F/(F+W) and beach F/(F+W) reached a plateau, and the percentage of fines captured reached a maximum. When feed F/(F+W) increased to above 15% w/w, the percentage of fines captured started to decrease due to increasing carrier rheology and unsettled fines being lost as runoff. If all other parameters were kept the same, the potential for sand to capture fines in the beach was found to be in the order HG > MG > LG.

Figures 6 and 7 have excluded the data of tests D8 and D5, both involving flocculation. The impact of the flocculation on the beach consolidation and shear strength need to be evaluated for a longer period of time to produce meaningful results.

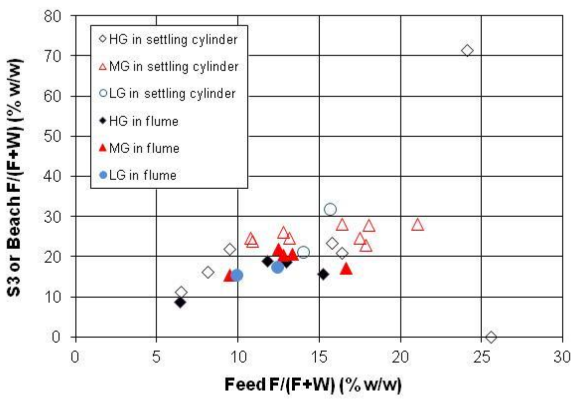

Figure 6. Feed F/(F+W) vs. Beach F/(F+W) for Settling Cylinder and Flume Tests.

Shear Strength

The shear strength of the beach deposit was measured in four locations and two depths per each location by using a field vane tester. However, the majority of flume tests generated beach thickness ranged from 100 mm to 300 mm, not thick enough to be divided into two layers and measured by a field vane tester. The consolidation period in the flume lasted about 24 hours before the shear strength measurement was taken, which is an inadequate length of time to appropriately evaluate consolidated shear strength. Therefore the shear strength data is not presented in this paper.

Other data such as beach vertical concentration profiles against settling time, beach permeability and void ratio, and beach shape will be reported in another paper.

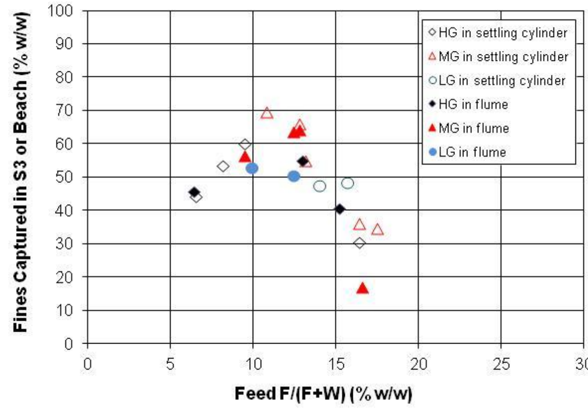

Figure 7. Feed F/(F+W) vs. Rate of Fines Captured for Settled Cylinder and Flume Tests.

KEY LEARNINGS

1. For all three groups of coarse sand (HG, MG and LG) tested, more than 50% of fines from the feed can be captured in the beach after the first placement.
2. The percentage of fines captured in beach was governed by the feed F/(F+W) ratio. The optimum feed F/(F+W) ratio was at 13-15%. Above this value, the percentage of fines captured decreased due to fines suspended and lost in the runoff.
3. LG sand's potential for capturing fines was limited due to its already high percentage of fines in solids.
4. Flocculation of feed could increase the percentage of fines captured in sand beach; however the beach strength needs to be evaluated over a longer period of time than the 24 hours tested in this study.

FUTURE WORK

1. Due to limited materials available to the project and concerns over entrained air in the feed, only about 400 L of slurry was discharged into an 8 m long flume for each of the eleven flume tests in the study. The beach was neither thick enough, nor the consolidation period long enough (>24 hours) for an accurate beach shear strength measurement. It is recommended to use more material for a

dynamic flume test, with a beach consolidation period extended to days or weeks.

2. Although no wall effect was observed in the flume tests, a wider (>0.25 m) flume or field test is recommended for potential future fines captured in sand beach studies if adequate material is available.

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Table 1. Specifications of Raw Test Materials.

Feed Component	SFR_{44}	Bitumen (% w/w)	Water (% w/w)	Solids (% w/w)	F/Solids (% w/w)
HG Sand	17.7	0.3	19.6	80.0	5.4
MG Sand	9.2	0.4	17.3	82.3	9.8
LG Sand	6.4	0.9	19.3	79.8	13.4
MFT	0.01	1.4	66.3	32.4	98.9
TT 0.8	0.8	2.3	49.8	48.0	55.1

Table 2. Settling Cylinder Results for High Grade Sand (HG).

	A3 HG Sand Only	A6 HG Sand + MFT	A8 HG Sand +TT0.8	A10 HG Sand + MFT x 2	A14 HG Sand + MFT x 4
Feed Cs (% w/w)	54.5	49.3	53.6	55.8	65.6
Feed F/(F+W) (% w/w)	6.5	8.1	9.5	16.4	25.6
Feed SFR_{44}	16.2	10.0	10.1	5.4	4.6
Feed F/Solids (% w/w)	5.8	9.1	9.1	15.5	18.0
% Fines Captured in S3 (% w/w)	44.2	53.6	60.2	30.7	0.0
Net Fines Captured in S3 (kg/m³)	50.4	75.6	108.8	112.0	0.0
% Feed Vol Captured in S3 (% v/v)	41.9	46.0	40.2	36.4	0.0
S3 Cs (% w/w)	80.2	80.4	80.8	78.1	non-distinguishable
S3 F/(F+W) (% w/w)	11.3	16.2	22.0	20.8	non-distinguishable
S3 SFR_{44}	30.8	20.3	13.9	12.6	non-distinguishable

Table 3. Settling Cylinder Results for Medium Grade Sand (MG).

	A2 MG Sand Only	A5 MG Sand + MFT	A7 MG Sand + TT0.8	A9 MG Sand + MFT x 2	A11 MG Sand + MFT x 3
Feed Cs (% w/w)	53.7	52.3	53.6	55.9	54.4
Feed F/(F+W) (% w/w)	10.8	12.2	13.2	16.4	17.5
Feed SFR_{44}	8.6	6.9	6.6	5.5	4.6
Feed F/Solids (% w/w)	10.5	12.7	13.1	15.5	17.8
% Fines Captured in S3 (% w/w)	69.7	66.1	55.1	36.2	34.7
Net Fines Captured in S3 (kg/m^3)	128.4	150.5	131.0	158.3	135.7
% Feed Vol Captured in S3 (% v/v)	45.6	43.1	44.3	30.3	37.3
S3 Cs (% w/w)	80.3	78.3	79.7	79.5	78.8
S3 F/(F+W) (% w/w)	24.6	26.2	24.6	28.1	24.6
S3 SFR_{44}	11.5	9.1	11.1	8.9	10.4

Table 4. Settling Cylinder Results for Low Grade Sand (LG).

	A1 LG Sand Only	A4 LG Sand + MFT
Feed Cs (% w/w)	51.3	47.8
Feed F/(F+W) (% w/w)	14.0	15.7
Feed SFR_{44}	5.5	3.9
Feed F/Solids (% w/w)	15.5	20.3
% Fines Captured in S3 (% w/w)	47.5	48.6
Net Fines Captured in S3 (kg/m^3)	111.3	188.0
% Feed Vol Captured in S3 (% v/v)	49.7	35.7
S3 Cs (% w/w)	78.8	79.7
S3 F/(F+W) (% w/w)	21.1	31.8
S3 SFR_{44}	12.9	7.4

Table 5. Flume Results for High Grade Sand (HG).

	C1 HG Sand Only	D1 HG Sand + MFT	D7 HG Sand + MFT X 2	D8 HG Sand + MFT + 400 gpT AF246
Feed Cs (% w/w)	54.5	51.4	55.4	42.7
Feed F/(F+W) (% w/w)	6.4	12.9	15.2	11.8
Feed SFR_{44}	16.4	6.1	5.9	4.6
Feed F/Solids (% w/w)	5.8	14.1	14.5	17.9
% Fines Captured in Beach (% w/w)	45.8	55.0	53.6	76.9
Net Fines Captured in Beach (kg/m^3)	42.3	108.3	108.8	142.8
% Feed Volume Captured in Beach (% v/v)	51.3	53.9	56.5	56.1
Beach Cs (% w/w)	76.4	74.8	76.5	61.1
Beach F/(F+W) (% w/w)	8.7	18.8	15.8	18.9
Top Beach SFR_{44}	30.7	9.5	14.7	1.6
Bottom Beach SFR_{44}	39.1	14.0	24.7	15.9
Average Weighted Beach SFR_{44}	35.4	12.3	19.8	10.0
% Feed Volume Drained as Runoff (% v/v)	48.7	46.1	43.5	48.9
Runoff Cs (% w/w)	1.1	16.3	13.6	6.4

Table 6. Flume Results for Medium Grade Sand (MG).

	C2A MG Sand Only	D2 MG Sand + MFT	D4 MG Sand + TT0.8	D6 MG Sand + MFT x2	D5 MG Sand + MFT + 400 gpT AF 246
Feed Cs (% w/w)	52.0	52.1	51.7	48.8	49.6
Feed F/(F+W) (% w/w)	9.5	12.8	12.5	16.6	13.4
Feed SFR_{44}	9.3	6.4	6.5	3.8	5.4
Feed F/Solids (% w/w)	9.7	13.5	13.3	20.9	15.6
% Fines Captured in Beach (% w/w)	56.7	64.3	63.8	27.3	88.4
Net Fines Captured in Beach (kg/m³)	86.1	127.1	118.9	94.9	153.4
% Feed Volume Captured in Beach (% v/v)	49.0	52.6	54.3	26.7	64.7
Beach Cs (% w/w)	74.6	73.1	77.9	75.9	64.3
Beach F/(F+W) (% w/w)	15.5	20.6	21.9	17.3	20.8
Top Beach SFR_{44}	20.1	6.5	10.7	14.4	2.9
Bottom Beach SFR_{44}	16.2	13.1	16.2	14.4	13.5
Average Weighted Beach SFR_{44}	18.2	10.8	13.7	14.4	7.7
% Feed Volume Drained as Runoff (% v/v)	51.0	47.4	45.7	73.3	35.3
Runoff Cs (% w/w)	4.6	11.7	10.3	37.9	6.4

Table 7. Flume Results for Low Grade Sand (LG).

	C3 LG Sand Only	D3 LG Sand + MFT
Feed Cs (% w/w)	48.9	53.6
Feed F/(F+W) (% w/w)	9.9	12.4
Feed SFR_{44}	7.7	7.1
Feed F/Solids (% w/w)	11.5	12.3
% Fines Captured in Beach (% w/w)	53.0	50.6
Net Fines Captured in Beach (kg/m³)	81.8	95.7
% Feed Volume Captured in Beach (% v/v)	52.4	52.2
Beach Cs (% w/w)	75.7	75.8
Beach F/(F+W) (% w/w)	15.3	17.5
Top Beach SFR_{44}	17.1	13.0
Bottom Beach SFR_{44}	22.6	18.8
Average Weighted Beach SFR_{44}	20.1	16.1
% Feed Volume Drained as Runoff (% v/v)	47.6	47.8
Runoff Cs (% w/w)	7.8	11.5

IN SITU HYDRAULIC CAPPING OF SOFT TAILINGS

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ABSTRACT

Large inventories of Mature Fine Tailings (MFT) have accumulated in large ponds as a result of its slow consolidation behaviour. Increasing pressure is placed on the oil sands industry to conform to strict environmental standards concerning tailings management. Varying mechanical and chemical techniques have been developed for reclaiming MFT. However, these require multiple and complex operational steps, which indicate that they are not feasible at the required scale for the oil sands industry.

Consolidation of MFT can be significantly accelerated when applying a surcharge, e.g. by a sand cap. However, installation of a cap is not straightforward, as the low strength of MFT does not allow access for land-based equipment. But the careful and subaqueous installation of sand on soft mud using floating equipment generates strength gain at the sand-mud interface, which enables installation of subsequent sand layers to form a cap of several meters.

Previous project experiences clearly indicate that hydraulic capping of MFT-like material using state-of-the-art dredging technology is a practical, safe and proven full-scale technique, at various operational scales and under different circumstances. Advanced engineering, dredging expertise and site-specific characteristics are required to design a sand cap and to ensure an efficient and safe work method. Key for a successful capping project is an advanced spreader device with an experienced crew.

A work method for capping of MFT in both inactive (no ongoing disposal operations), filled ponds and active ponds is presented. Modeling results indicate that vertical (wick) drains installed in MFT enable meeting the specific strength-requirements of Directive 074. Finally, the sustainable character of hydraulic capping of MFT follows from the limited operational steps, which minimizes CO_2 emissions, from the possibility to recycle expelled process water and because no chemicals are required.

INTRODUCTION

Background

Tailings are the waste product of the oil sands industry in Alberta, Canada, and consist of a mixture of water, sediments and unrecovered oil. After disposal, sand settles out and forms beaches along the perimeter of tailings ponds. The fine fraction slowly settles out to a yoghurt-like substance, which is referred to as Mature Fine Tailings (MFT). MFT exhibits slow consolidation due to low permeability and low effective stress. This resulted in the long-term storage of MFT: the past decades about 1 billion m^3 of MFT has accumulated in large tailings ponds, covering more than 170 km^2.

Increasing pressure is placed on the industry to conform to strict environmental standards. Directive 074 (ERCB, 2009) sets criteria for closure and reclamation of existing and newly generated tailings. However, reclamation is not straightforward as MFT exhibits typically a consolidation time scale of 100’s-1000 years and an undrained shear strength (c_u) of 10’s - 100’s Pa. Additionally, the large quantities of MFT require a method which is feasible at the required scale for the oil sand industry.

Problem Definition

A number of methods to accelerate dewatering and reclamation of oil sands tailings are currently being tested. Most operators apply a multi-pronged approach, involving a combination of mechanical and/or chemical treatments (BGC Engineering Inc., 2010). However, these techniques involve multiple operational steps and significant resources. This indicates that they are not feasible at the required scale. Moreover, they often involve environmental impacts, e.g. due to the use of chemicals.

The in-situ installation of a surcharge on MFT by sand capping provides, however, a feasible alternative, forming a stable platform for further reclamation (e.g. relocation of muskeg). But cap installation is not straightforward as MFT exhibits a fluid character upon accessing and subsequent dry cover installation, even with light construction equipment (Neukirchner and Lord, 1998).

To prevent the occurrence of instabilities, a light-weight cover material may be applied, which is currently being tested at Suncor's Pond 5 (e.g. Wells and Caldwell, 2009). The test concerns the placement of a geotextile on frozen MFT during winter, and subsequent capping with petroleum coke. However, the installation of light material involves multiple, complex operational steps which reduce the overall feasibility.

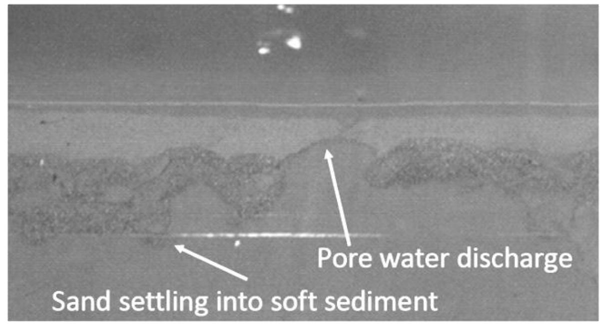
E.g. Wels et al. (2000) and Costello et al. (2010) indicate that cover material may also be placed hydraulically using a subaqueous discharge technique. Sand capping in thin layers (cm's) is already possible on material with a shear strength of about 100 Pa (see e.g. Jakubick et al., 2003), similar to the typical strength of MFT.

The installation of especially the initial layer is complex and requires a controlled movement of the discharge point. During a first pass, settling sand grains are trapped in the upper layer of soft mud (Figure 1a). These grains form a layer, which generates consolidation and strength gain, providing a firm base for subsequent passages (Figure 1b). The initial irregular sand-mud interface (Figure 1a) follows from varying sand and mud characteristics and small irregularities of the installation process.

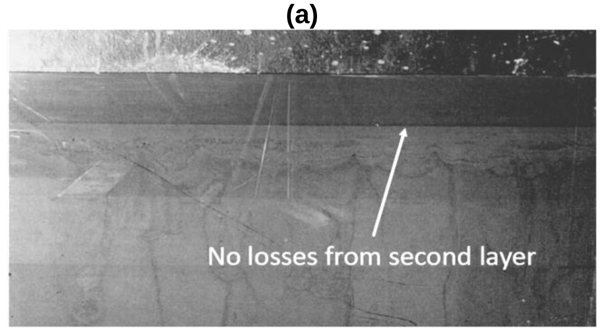
Multiple dynamic and static failure mechanisms of a cap on soft mud may occur, at various spatial and temporal scales. However, by carefully adjusting the cap installation rate to the strength gain of the underlying, consolidating mud, a sand cap of several dm's to m's can be installed, layer-by-layer and with gradual side slopes. But the stability of the cap, both during and after installation, is rather sensitive to the capping process. A controlled and accurate method involving multiple pass of a spreader device is therefore required.

When accurate installation of thin layers is not possible, unequal loading conditions and accompanying instabilities will occur, both during and after capping. To prevent this, a geogrid or geotextile may be used to evenly spread the loading upon the subsoil. However, these reduce the feasibility, especially when applied at a large scale.

It is further noted that the design of a sand cap and accompanying installation work method are always site-specific, strongly depending on the characteristics of the capped material and capping material, as well as on pond geometry.



(a)



(b)

Figure 1. Section of Deltares laboratory flume showing fine tailings ($c_u \approx 15$ Pa) capped with a 1st layer of sand (a), forming a stable basis for the 2nd layer (b).

The main advantage of subaqueous, hydraulic sand capping is that the submerged weight of the cap is applied to the soft tailings in small increments, without the need for land-based equipment. Further, the costs involved with hydraulic capping are relatively low compared to other techniques (BGC Engineering Inc., 2010).

Sand capping using floating equipment is a commonly applied technique in the international dredging industry. However, the oil sands industry has limited experience with advanced, state-of-the-art floating dredging equipment with respect to controlled and accurate capping.

Objective and Approach

The objective of the current paper is to demonstrate that subaqueous, hydraulic capping of MFT using floating equipment is a viable, safe and full-scale technique for providing a trafficable surface, allowing for final reclamation.

A review of successful capping projects is presented first, and typical applications of hydraulic capping using floating equipment at varying production rates are illustrated. The installation and effectiveness of vertical drains in soft material in combination with a surcharge are discussed as well.

To illustrate the applicability of hydraulic capping to the oil sand industry, a work method for both inactive (no ongoing disposal operations), filled ponds and active ponds is presented. A work method for installing vertical drains in MFT using floating equipment is presented as well. Modeling results are presented to illustrate the effect of a sand cap on the long-term consolidation of MFT, both with and without vertical drains. Finally, conclusions are presented with respect to the feasibility of the proposed work method, in view of the current challenges in the oil sands industry.

MELBOURNE PROJECT

Project Description

The Melbourne Deepening Project in Australia was executed in 2008/2009 by Royal Boskalis Westminster[1], under the direction of the Melbourne Port Authority. Part of the project concerned accurate and controlled capping of 1.7 million m^3 of contaminated mud in a confined submarine disposal site (Figure 2) to ensure full containment of the mud. A minimum cap thickness of 0.5 m was required.

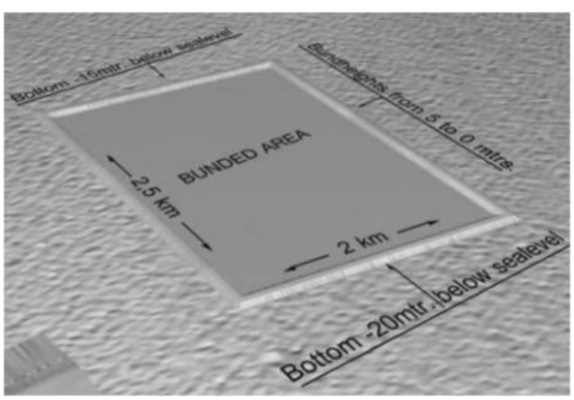
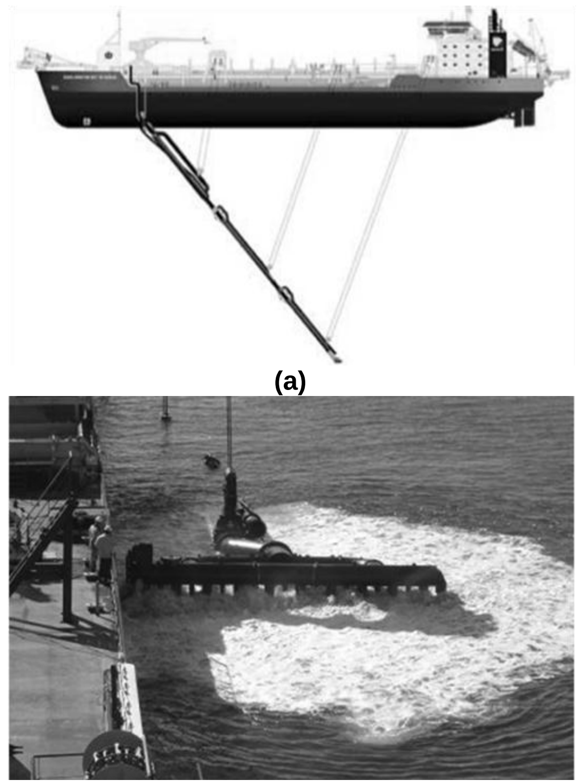


Figure 2. Submarine disposal facility for contaminated mud at 20 m water depth.

A client-contractor alliance agreement was developed to share risks and to benefit from Contractor's practical expertise. The state-of-the-art consolidation model Delcon was applied by the Dutch research institute Deltares[2] to model consolidation behaviour of the capped mud before, during and after capping. This allowed to design the thickness of subsequent capping layers, to develop a work method and to ensure a stable cap installation process.



(a)

(b)

Figure 3. TSHD applied for sand capping (a) and a 12 m wide spreader device (b) at the lower end of the TSHD's suction pipe.

Soft mud was dredged by a large Trailing Suction Hopper Dredger (TSHD, Figure 3a). The mud was subsequently disposed using a Boskalis-patented diffuser (see Mastbergen et al., 2004) during a 80-day period. After a consolidation phase of 5 months, the 2.75 m mud layer was capped with fine to medium sized sand ($d_{50} \approx$ 200-350 μm). The undrained shear strength of the mud at the start of capping was ~25 Pa, at a bulk density of 1370 kg/m^3.

[1] www.boskalis.com

[2] www.deltares.nl

Work Method and Verification

The relative small thickness of the cap required an accurate work method. Therefore, a 12 m wide sand spreading device developed by Boskalis' Research Department was installed at the lower end of a suction pipe of a large TSHD (Figure 3b). Then, the hopper of the TSHD was pumped out and the mixture was discharged at constant and low density, at a few m's above the surface of the deposited mud.

Using the TSHD's dynamic tracking system, straight tracks could be sailed at a slow speed (3 knots) over the disposal area. This allowed for the installation of successive sand layers in parallel strips of about 50 m, at 5-10 cm layer thickness. The total thickness of the sand cap was about 0.5-0.8 m, with a total volume of about 1 million m^3. The in-situ cap installation rate was 9,000 m^3/hr; the duration of the cap installation was about 6 weeks. Real-time hydrographical surveys of the placed sand allowed for process optimization during capping, which prevented the occurrence of instabilities.

Immediately after capping, visual inspections by divers were executed and tube samples were taken for laboratory testing. These showed a sharp, well-defined transition from sand cap to contaminated mud (Figure 4). The surcharge of the cap accelerated the consolidation of the mud layer, which was illustrated by c_u of about 0.5 kPa at about 5 cm below the sand-mud interface.

Figure 4. Core sample indicating a clear sand-mud interface (due to sampling some smearing of mud occurred).

IJBURG PROJECT

Project description

The IJburg project (1998-2002) in the Netherlands was executed by a joint venture including Boskalis; employer was the Municipality of Amsterdam (de Leeuw et al., 2002). Objective was to develop a 200 ha housing area in the shallow Outer IJsel Lake (0.5-3 m water depth, Figure 5). This involved the reclamation of extremely soft subsoil consisting of organic clay (c_u of 0.3-3 kPa, 40-50% clay content) with a thickness of 0-4 m, and located on a 6-12 m layer of peat and clay. Over 10 million m^3 of sand ($d_{50} \approx$ 150-250 μm) was installed to create a stable platform at 3 m above water level.

Figure 5. Overview of the IJburg Project showing the partly reclaimed area consisting of a layer of sand on top of a soft subsoil.

Work method

The design prescribed the controlled installation of thin layers of sand by means of floating equipment. This prevented instabilities of the subsoil and avoided the use of costly geo-textiles on top of the existing subsoil for stabilization of the sand-clay transition. Therefore, 4 million m^3 of sand was sprayed in thin layers (20-50 cm) and at high accuracy (± 5 cm) by means of a custom-built spreader pontoon (Figure 6). The remaining

6 million m^3 was reclaimed by conventional methods.

The initial sand-layer (median particle size $d_{50} \approx 150\text{-}250\ \mu m$) exhibited slopes of 1:5 to 1:10. A consolidation period of 4 weeks after installation of each layer was obligatory to guarantee stability of the subsoil. Following layers exhibited a thickness of 0.6-0.8 m and 1:4-1:10 slopes. The width of each sand layer was about 12 m, following from the width of the chute at the stern of the pontoon (Figure 6). The installation rate was around 100,000 m^3 of sand per week. With respect to the shallow depth, minimum drafts were required for the pontoon and all auxiliary equipment. Based on a successful pilot trial of the accurate spraying process, 95% of the area was reclaimed without geo-textiles. These were only used at future dike locations to ensure even spreading of the future load onto the subsoil.

Figure 6. Computer-controlled spreader pontoon in shallow water. The sand-water mixture is distributed by a 12-m wide chute.

Process control and results

The required installation of thin layers implied that the spreader pontoon had to move relatively fast, resulting in frequent anchor handling and potential delays. Furthermore, anchoring in soft soil and shallow waters requires specific anchors and procedures, requiring an experienced crew. High accuracy was obtained by applying computer-controlled winches, which automatically adjusted the speed of the pontoon to the density and flow rate of the provided sand-water mixture.

Thanks to the controlled placement system, the cap was installed in accordance with the contract, without major instabilities. Continuous and extensive monitoring ensured a safe construction and minimized risks. Finally, CPT-readings showed a clear transition between the soft subsoil and the reclaimed sand layer.

FOX RIVER PROJECT

The Fox River Cleanup Project[3] at Green Bay, USA is currently being executed (2009-2016) by Tetra Tech, with Stuyvesant Environmental Contracting (affiliate of Boskalis Environmental) as subcontractor. Client is a consortium of regulatory agencies. To reduce human health and environmental risks caused by the presence of PCBs in Fox River sediments (Feeney et al., 2011), 3 million m^3 of contaminated dredged material is dredged and processed. This includes the separation of clean sand and the dewatering of fine sediments with membrane presses.

Parts of the river bed (sediments at lower PCB-levels) are covered with a protective cap (total area of 240 ha). The first layer consists of sand (cm’s), with subsequent layers of gravel and quarry stone for protection against erosion. Sand is installed with a spreader device mounted on a small pontoon (Figure 7), similar to a salt-spreader at the back of trucks during winter. This sand placement method is particularly suitable for accurate capping of relatively small surface areas and at low installation rates.

Figure 7. Detailed view of the sand capping process at the Fox River, showing a conveyor belt (right upper part) providing dry sand to a spreader device (middle).

[3] www.foxrivercleanup.com / www.boskalisdolman.nl

CHANGI AIRPORT PROJECT

The projects discussed above concern capping of existing volumes of soft material. But the principle of sand capping has also been applied for reclamation of slurries during filling of a disposal area. This enhances consolidation and increases the storage capacity during the filling process. An nice example for the latter is the Changi Changi Airport project in Singapore. Reclamation for this project in the early 1990's involved an area of 3000 ha requiring 320 million m^3 of sand. As the availability of sand was limited and marine clay was ample available, a trial was executed using 3 million m^3 of dredged and hydraulically placed marine clay as fill material in a 40 ha confined area.

A relatively simple system of a perforated floating pipeline manoeuvred by a barge was used to install the clay-slurry, sandwiched between sand layers at 0.5-1 m thickness (Figure 8). The intermediate sand layers formed a horizontal drainage system, which significantly accelerated consolidation of the clay. For more information see Choa (1994), Lee et al. (1987) and Karunaratne et al. (1990).

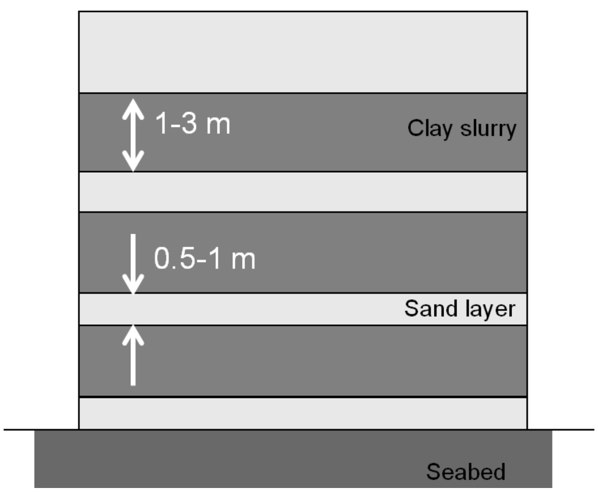


Figure 8. Schematized depiction of the sandwich structure (total thickness 5-9 m) as applied at the Changi Airport reclamation.

VERTCIAL (WICK) DRAINS

Introduction

The hydraulic conductivity of soft, compressible soils is typically low. This implies that consolidation time scales are considerable, also after installing a surcharge (sand cap), especially for large layer thicknesses. This may generate instabilities, e.g. when a surcharge load is increased during further reclamation operations. Prefabricated synthetic vertical drains (or 'wick drains') significantly shorten the dewatering path length of consolidating soils. A fast and tailor-made consolidation process can be obtained by adjusting the horizontal spacing and vertical length of drains. Challenges for the application of drains in soft mud concern the accessibility for installation rigs, and the potentially decreasing effectiveness of drains during consolidation due to clogging, the formation of a filter cake (i.e. solid-like tailings) and/or deformations.

Dijkstra (2012), Yao et al. (2010) and Wells and Caldwell (2009) report positive results for laboratory tests on the effectiveness of drains in oil sands tailings. Neither fines, bitumen nor gas caused serious clogging of the filter jackets. The feasibility of vertical drains for dewatering of oil sands tailings is further illustrated by the current installation of wick drains at Suncor's Pond 5 (e.g. BGC Engineering Inc., 2010). Vertical drains have successfully been applied at many projects round the world; two examples are briefly discussed below.

IJburg Project

At the IJburg Project, 8.3 million m of vertical drains were installed to increase the stability of the subsoil. Production reached 350,000 m per week with nine land-based installation rigs. Compared to sections for which no drains were installed, the consolidation rate and strength gain were up to ten times faster for drains at a 1.0-2.5 m triangular grid spacing and a length of 9 m.

Beirut Project

As argued above, one of the challenges for drains installation in soft mud concerns the accessibility.

This may require e.g. lowering the water table and/or a considerable consolidation period to allow for strength gain of the subsoil after capping. An alternative is to install drains using an installation rig mounted on a floating pontoon (Figure 9), which was successfully applied at a harbour project in Beirut, Libanon (Goot, van der, 2011) by Cofra BV[4], a subsidiary of the Royal Boskalis Westminster Group.

About 180,000 m^2 of container storage area was developed at water depths ranging between 4.5-22 m. Drains were installed down to 52 m below the bottom of the installation rig. Using a computer-controlled pontoon allowed for an accurate installation of the drains according to a predefined grid. Special features of the operation concerned an automatic system for drain-cutting above the seabed, and a water-jet system to penetrate the hard seabed.

Figure 9. Installation of vertical drains by an installation rig mounted on a floating pontoon during the Beirut Project.

CAPPING OIL SANDS TAILINGS

Proposed work method

A work method is presented for hydraulic capping of a filled, non-operational tailings pond. Objective for the work method is to create a trafficable surface allowing for final reclamation. The large scale of the oil sands industry requires a practical and proven full-scale technology, with limited rehandling steps. Equipment should have limited draft and/or no propeller propulsion to prevent agitation of the MFT. Further, the size of the equipment is limited as it needs to be mobilized by road.

Based on these requirements as well as on previous project experiences as discussed above, the following work method is proposed for a schematized tailings pond as shown in (Figure 10a). Sand is dredged by a Cutter Suction Dredger (CSD), either from a beach or stockpile along a tailings pond. The sand-water mixture is subsequently pumped through a floating pipeline towards a spreader pontoon, similar as applied for the IJburg Project (Figure 6).

For example, this work method enables installing a 2 m sand cap on a 2 km diameter tailings pond (total sand volume: 6.3 million m^3) within two operational seasons (i.e. 2 times 6 months). For a schematized, conical shaped tailings pond with a depth of 40 m this implies the reclamation of around 42 million m^3 of MFT. A typical cap installation rate based on six consecutive layers at increasing layer thickness (5; 7.5; 12.5; 25; 50 and 100 cm) is 120,000 m^3/week.

Controlled and accurate sand capping requires a state-of-the-art spreader pontoon. This involves a computerized system controlling anchors and winches to adjust the speed of the pontoon, based on the on-line feedback of the winches and the GPS position of the pontoon. In addition, the flow rate and concentration of the sand-water mixture and bathymetrical data of the capped sand was used by the system. The on-line feedback of data further ensures process optimization and risk management.

4 www.cofra.nl

A dedicated dredging contractor is required for developing a site-specific work method. Further, dredging expertise should be incorporated into the design for capping of an oil sands tailings pond. Finally, an experienced crew is indispensible for the efficient and safe operation of the spreader pontoon.

Vertical drains

Vertical drains have to be installed to accelerate dewatering of capped tailings (Figure 10b). By adjusting the vertical length and horizontal grid spacing it is possible to obtain a customized consolidation process with strength-gain over the full-depth. Drains also allow for a controlled recycling of expelled water. Using installation rigs mounted on a pontoon (Figure 9) excludes the need for heavy land-based equipment and allows for drains installation directly after or even during capping. The pontoon should be computer-controlled to ensure accurate and efficient installation. Installing drains in a conical shaped tailings pond (2 km diameter, 40 m depth) requires 1-2 operational seasons, mainly depending on the horizontal grid spacing.

Capping operational tailings ponds

Directive 074 sets timelines to process fluid tailings at the same rate they are produced. This can be obtained by constructing a ‘sandwich structure, as discussed for the Changi Airport Project (Figure 8). Installing alternating layers of MFT and sand (Figure 10c) allows for equal MFT-filling and reclamation rates, and increases the storage capacity of tailings ponds.

Numerical model simulations

To illustrate the effect of the described work method, one-dimensional modelling results are presented on the consolidation of a 35 m layer of MFT. Simulations were executed by Deltares using the Delcon model, which was applied and validated for previous oil sands studies, as well as for the Melbourne project described before. It is noted that the simulation results indicate typical tailings behaviour during and after capping. A particular pond exhibits particular characteristics (e.g. tailings composition, pond geometry, etc) and therefore site-specific model input.

The first 10 years (starting in 2010) of the simulations reflect the disposal of MFT at a Sand-Fines-Ratio (SFR) of 5 to generate a realistic vertical MFT profile. A clay-water slurry at SFR of 0.7 and an initial void ratio equal to 8 ($c_u \approx 10$ Pa) reaches the pond; coarse particles settle on the perimeter beaches. The filling period was followed by one year of sand capping at small increments. The remaining part of the simulation concerns long-term consolidation. The total simulation period is 100 years.

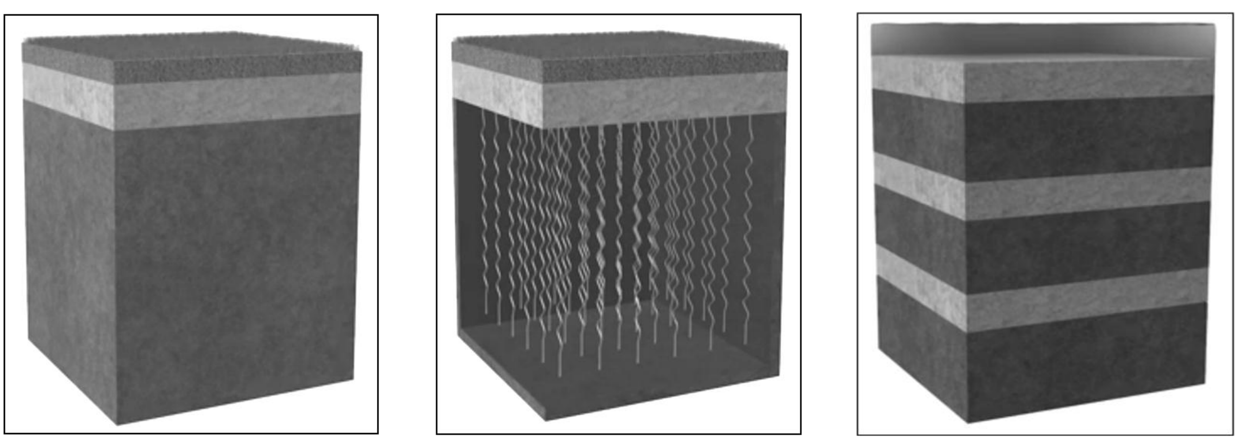

(a) (b) (c)

Figure 10. Sections of a schematized oil sands tailings pond with a cap of sand and muskeg (a), with additional vertical drains (b) and a sandwich structure of alternating layers of tailings (~m’s) and sand (~dm’s) (c).

To define the consolidation behaviour and strength development, empirical coefficients for effective stress, hydraulic conductivity and undrained shear strength formulations are derived from Jeeravipoolvarn et al. (2009), Shaw et al. (2010) and Masala and Matthews (2010).

Results for three scenarios are presented, reflecting MFT-height and undrained shear strength as a function of time. Figure 11a shows the results for consolidation after capping with a 5 m sand layer (Figure 10a). After 100 years, the depth-averaged strength of the capped MFT is only 0.5-5 kPa. Similar results were observed for a 1 m sand cap, as well as for MFT without a cap. This illustrates that the low permeability and large MFT layer thickness govern the consolidation behaviour and strength gain, rather than the presence and/or magnitude of a surcharge.

Figure 11a further indicates that only the upper part of the capped MFT exhibits significant strength gain, supporting the load of the sand cap (about 46 kPa for a cap porosity of about 40%). This strength gain at the sand-mud interface follows from the transition of MFT into a filter cake, exhibiting cohesive strength with tensile capacity. In time, the front of the consolidating filter cake layer moves slowly downwards, which explains the observed stratified character of the MFT-strength.

Consolidation is significantly accelerated when installing drains over the full depth Figure 10b), as they shorten the path length of the expelled water.

Figure 11b shows that a 5 kPa MFT strength is obtained after about 30 years when installing vertical drains at a horizontal spacing of 5 m. The importance of the grid spacing is further illustrated by Figure 11c, which shows a 10 kPa strength directly after installation at a 1 m grid spacing. This clearly indicates that vertical drains significantly enhance consolidation and strength gain.

CONCLUSIONS

The careful, subaqueous installation of sand on soft mud generates strength gain at the sand-mud interface, which enables installation of subsequent layers to form a sand cap of several meters.

Previous project experiences clearly indicate that hydraulic capping of MFT-like material using state-of-the-art dredging technology is a practical, safe and proven full-scale technique, at various operational scales and under different conditions.

Advanced engineering, dredging expertise and site-specific characteristics are required to design a stable sand cap and to ensure an efficient and safe work method. Key for a successful capping project is an advanced spreader device with an experienced crew.

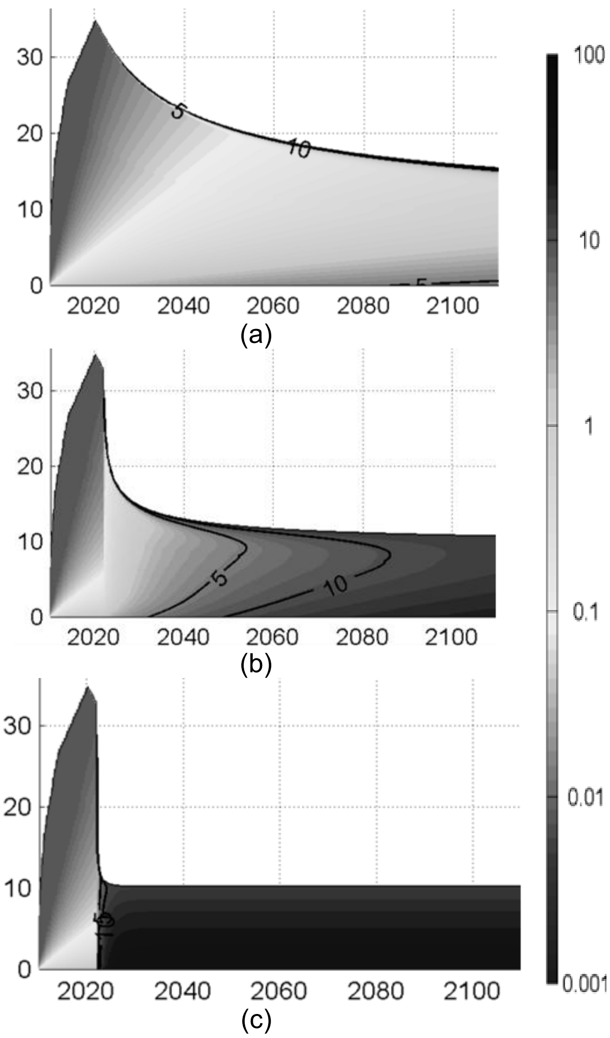


Figure 11. Model results with MFT height [m] as a function of time [yrs]. The shading, contour lines and colorbar reflect undrained shear strength [kPa]. After a fill-period the effect of consolidation is shown for a 5 m cap (a), a 5 m cap with drains at 5 m spacing (b) and a 5 m cap with drains at 1 m spacing.

Continuous and accurate placement of sand in thin layers requires the on-line feedback of the winches and the GPS position of the pontoon to adjust the speed of the spreader to the flow and concentration of the sand-water mixture. This implies that light-weight cap material and/or the application of geotextiles below the cap, which both require additional and complex operational steps, is not needed.

The applicability of hydraulic capping to the oil sands industry is further illustrated by the described preliminary work method for capping of a tailings pond. The method allows for creating a trafficable surface for final reclamation, in-line with the overall objective of Directive 074. The large production rates and limited operational steps due to the in-situ character of the technique further demonstrate the feasibility of hydraulic sand capping.

Vertical (wick) drains significantly accelerate consolidation and strength gain of capped soft material. Modeling results indicate that vertical drains installed in MFT enable meeting the specific strength-requirements of Directive 074. The installation of a sandwich structure, with alternating layers of MFT and sand, minimizes the need for long-term storage of MFT and further minimizes the footprint of the industry.

Finally, hydraulic capping of MFT exhibits a sustainable character, as it (1) requires only a limited number of operational steps and therefore limited fuel consumption and CO_2 emissions, (2) does not require the addition of chemicals, and (3) allows for a continuous recycling of expelled process water.

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TSRU TAILINGS PILOT–SCALE THICKENING AND DEPOSITION TRIALS

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ABSTRACT

Imperial Oil Resources Limited, Shell Canada Energy, Teck Resources Limited, and Total E&P Canada Ltd. are oil sands operators with facilities in operation, under construction, or planned in northern Alberta using the paraffinic froth treatment process for bitumen extraction. This type of extraction process produces tailings from the Tailings Solvent Recovery Unit (TSRU), which contain residual solvent and a significant content of heavy hydrocarbons (asphaltenes) rejected from the froth treatment process. TSRU tailings exhibit unique geotechnical properties and depositional behaviour as compared to other fluid fine tailings types.

Preliminary research indicated a potential for thickening TSRU tailings, which can result in both heat recovery and creation of a paste-like material that can be safely and economically deposited to meet Directive 074 requirements. The four operators implemented a pilot-scale deposition trial in the late fall of 2011 to investigate the potential of this approach.

Pilot-scale deposition trials were conducted at the Muskeg River Mine site, using Shell's TSRU tailings and a 1-m-diameter thickener provided by Outotec (Canada) Ltd. (Outotec). The deposits were instrumented for monitoring of geotechnical behaviour during deposition and consolidation, including desiccation and freezing/thawing effects.

This work found that untreated and in-line flocculated TSRU tailings segregated upon discharge into a coarse beach, which had negligible settlement and was little affected by freeze/thaw, and a fines slurry pool, which had significant prolonged settlement and significant freeze/thaw settlement. Thickened TSRU tailings did not segregate, settled quickly and uniformly over the cell length, and had fairly uniform settlement due to freeze/thaw.

INTRODUCTION

In early 2011, Imperial Oil Resources Limited (Imperial), Shell Canada Energy (Shell), Teck Resources Limited (Teck), and Total E&P Canada Ltd. (Total) decided to study a common technical basis for disposal of froth treatment or TSRU tailings. Previous research indicated a potential for thickening of TSRU tailings to increase hot water recovery and deposit early strength gain. The TSRU testing program for 2011 was based on pilot-scale deposition of thickened tailings generated using a 1-m-diameter thickener operated by Outotec. The test site was located on the External Tailings Facility (ETF) dyke at Shell's Muskeg River Mine (MRM) site.

The primary objectives of the 2011 TSRU testing program included:

- Verification of bench and mini-pilot performance data,
- Optimization of thickener operation to achieve target thickener underflow and overflow streams; and
- Monitoring of geotechnical performance of the deposits over time, and field and laboratory testing to provide data for understanding full-scale behavior.

The start of the program was delayed for various reasons from August to mid-October 2011. The duration of the field campaign was thus compressed to about 12 days so that deposition of material was completed before the onset of consistently sub-zero daily temperatures. Full implementation of the geotechnical scope was restricted due to time constraints and cell access issues.

One or two lifts of TSRU material were deposited into six rectangular cells and one long flume (trench) excavated in the sand. One cell and the flume were filled with untreated TSRU tailings (thickener feed), one cell with in-line flocculated TSRU tailings, and four cells with thickened TSRU tailings. Three cells with thickened TRSU tailings were instrumented for monitoring of temperatures, and total and pore pressures in tailings, while all cells and the flume had posts with staff gauges for measurement of tailings thickness.

The geotechnical program included sampling during and post-deposition for geotechnical index properties as well as consolidation and strength testing. In-situ density and strength of tailings were measured at the end of the field program.

THICKENER PROGRAM

Thickener feed was drawn from the "short pour" line, one of Shell's two low temperature paraffinic tailings lines, through a double block and bleed valve arrangement. A peristaltic (hose) pump, located adjacent to the valves, transferred the tailings to the thickener at a controlled rate.

A prepared 0.1 wt% solution of SNF A3338 polymer was metered by a small peristaltic pump to several locations (three available) in the tailings feed line at the thickener. Thickener overflow flowed by gravity into a large plastic tote. An air-driven diaphragm pump recycled a portion of the overflow solution to the thickener feed line, before the polymer injection locations, to achieve the target solids concentration as introduced to the thickener feed well. The remaining portion of thickener overflow was pumped to a nearby shallow swale that drained by gravity to the main tailings area. The thickener underflow was directed by hose pump to the deposition cells. A schematic of the project arrangement is provided in Figure 1.

Samples of thickener feed, underflow, and overflow were taken throughout the program for various analyses to characterize these streams. Solvent evaporation from the thickener surface was anticipated and so a Lower Explosive Limit (LEL) detector was installed close to the thickener surface. The alarm was never triggered.

The late start of the test program precluded adequate commissioning time for Outotec to assess fully a range of operating conditions. Some process adjustments were attempted during operation to explore different operating regimes (principally feed flow rate), although much of the range of process operating parameters (such as polymer dosage and feed dilution) resulted from variability in solids concentration from the main tailings line.

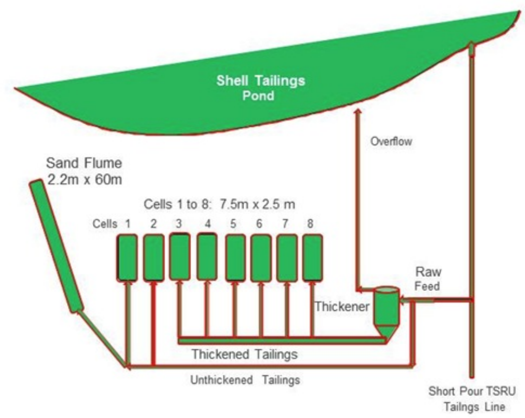

Figure 1. Schematic of Pilot Program Test Area.

Underflow containing 44-48% solids (bitumen plus mineral wt%) was achieved for 0.50-0.66 t/m^2h feed solids flow rate in feed containing 20-30% solids. The overflow was under 0.1% solids and preliminary data suggested that solvent was not released at the thickener surface. The following recommendations were offered by Outotec:

- Feed solids loading - 0.6-0.7 t/m^2h
- Flocculant dosage - 70-100 g/t solids
- Achievable underflow solids - 43-48%
- Achievable overflow clarity - <0.3%

Bench settling tests in 2-L graduated cylinders were also conducted during commissioning and operation to test the effectiveness of flocculant dosage, type, and age; as well as feed solids content on the settling rate of TSRU tailings and on supernatant solids content. Dosages of 40-80 g/t accelerated settling somewhat compared to no flocculant. Polymer addition was found necessary to achieve acceptable supernatant clarity.

A value of about 10% solids in thickener feedwell is generally accepted to be near the optimum for many thickener applications and successful operation was achieved during the pilot program at somewhat higher values. The thickener operated well through variation in feed solids content to produce consistent underflow material.

Rheology measurements on samples from the thickener bottom cone showed significant loss of yield stress after the shearing imposed as part of the instrument measurement procedure. One sample was also collected at the pipe discharge to a cell, for which the instrument unsheared yield stress was similar to the instrument well-sheared cone samples. A loss in yield stress would be expected for slurry flow in a small-diameter pipe and for slurry flowing across a beach [Winterwerp and van Kesteren, 2004].

DEPOSITION PROGRAM

Eight cells approximately 2 m wide and 7.5 m long at the base were dug in the MRM tailings dyke at Cell 25. The walls of the test cells had slopes of approximately 2:1. Seven of the cells were about 2 m deep and one was about 3 m deep. The cell bases sloped from the feed end to downstream (DS) end at slopes ranging from 1.2% to 2.8%. As an example, construction of Cell 4 is pictured in Figure 2 (empty), after the first of two lifts in Figure 3, and after the second lift in Figure 4.

Each cell was created to demonstrate a different deposition strategy as listed below:

- Cell 1 was filled with approximately 0.5 m of unthickened and unflocculated TSRU tailings (thickener feed) to demonstrate base case TSRU deposition.
- Cell 2 was filled with approximately 0.5 m of in-line flocculated TSRU tailings to demonstrate the effect of flocculant without the use of a thickener.
- Cell 3 was filled with approximately 0.5 m of thickened tailings. After three days, 0.15 m of sand was placed over the tailings and a second lift of approximately 0.5 m of thickened tailings was placed over the sand. The idea was to demonstrate the effect of sand interlayering on dewatering and strength gain.
- Cell 4 was filled with two lifts of approximately 0.5 m of thickened tailings, separated by a drying period of about two days to achieve 1 to 2 kPa strength in the first lift. The idea was to demonstrate dewatering and strength gain for multi-lift placement of thickened tailings.
- Cell 5 was filled with one lift of approximately 0.5 m of thickened TSRU tailings to provide a comparison between thickened tailings, in-line treated tailings (Cell 2), and untreated tailings (Cell 1).
- Cell 6 was planned to be filled with a 0.5 m lift of thickened tailings which had been shear-thinned before deposition. This cell was not filled during the test program due to time constraints.
- Cell 7 was lined with waste ore to create a low permeability layer to assess dewatering and strength gain with no under-drainage. This cell was filled with one lift of approximately 0.5 m of thickened tailings and a partial second lift separated by a drying time of about 5 days to achieve 1 to 2 kPa strength in the first lift.
- Cell 8 was planned to be filled with a single 2-m lift of thickened tailings to measure dewatering and strength gain of a thick lift, such as in a deep basin decanting method. This cell was not filled during the test program due to time constraints.

Figure 2. Cell 4 Before Filling.

Figure 3. Cell 4, 21 h After Placing First Lift.

GEOTECHNICAL FIELD PROGRAM

Tailings geotechnical behaviour was monitored at a single central location using an instrumented post in Cells 3, 4 and 7. The instrumentation was connected to data acquisition systems (DAS) for continuous recording of data.

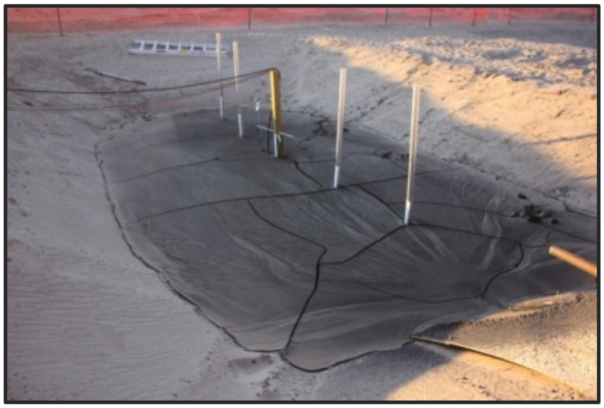

Figure 4. Cell 4, Four Days After Placing Second Lift.

Tailings thickness was measured in all cells using four posts fitted with staff gauges. The instrument posts in instrumented cells were also fitted with staff gauges. The four or five posts were distributed approximately equally along the cell length, as seen in Figures 2 through 5.

The data from piezometers and total pressure cells were judged unreliable and no conclusions were drawn from them; these data are not presented here. Thermistors provided reliable temperature data.

Monitoring of tailings drying was planned using tensiometers. These were not used, as overnight freezing temperatures arrived before the beginning of drying and destroyed the instruments.

Visual observations of process conditions, flow behavior, deposit front velocity, cell coverage, and release water were also recorded. Photo and video documentation was collected regularly.

Segregation and Water Release During Deposition

Visual observations during deposition revealed that untreated and in-line flocculated TSRU tailings in Cells 1 and 2, respectively, segregated upon discharge, the latter to a slightly lesser extent. In both cells, segregated coarse tailings created a relatively stiff beach on the upstream (US) side and formed a pool of thin fine slurry on the DS side. This behaviour is illustrated by Figure 5 showing a sudden change of slope between coarse beach and fine slurry pool (beach-below-water or BBW), typical for coarse tailings deposits in commercial oil sands ponds. This is in contrast with the uniform tailings surface typical for thickened TSRU tailings shown in Figures 3 and 4.

In-line flocculation for deposition into Cell 2 was performed for demonstration purposes in that polymer dosage, injection and mixing were not optimized. Nonetheless, the in-line flocculation had some effect on tailings behaviour in Cell 2, as will be noted later.

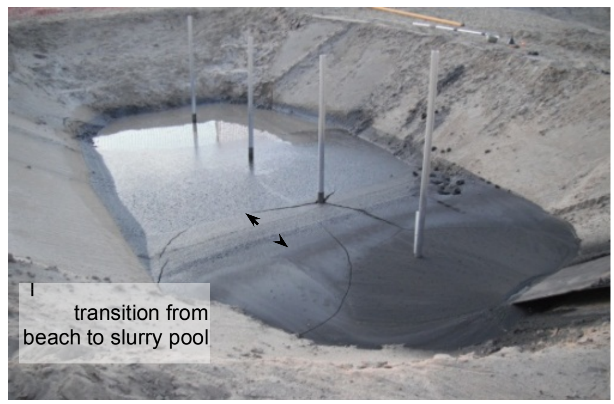

Figure 5. Cell 1, 19 Hours After Deposition.

Thickened TSRU tailings in the other four cells were not observed to segregate over the length of the test cells (6-7 metres). However, there remains uncertainty about the potential for segregation at commercial-scale over long deposition lengths. Testing the segregation potential of thickened TSRU tailings in a long flume or over a long beach would thus be recommended.

Water was released rapidly from thickened TSRU tailings. For instance, even though the material placed into Cell 4 was thickened to about 43% and 46% solids, for Lifts 1 and 2, respectively, water was released within 20 h after stopping deposition of Lift 1 (see Figure 3), and almost immediately for Lift 2, and collected at the DS end of the cells. Water release was also noticed to occur within minutes from samples of thickened material placed in graduated cylinders for laboratory measurements. Further, significant water release occurred from the thickened tailings placed into drum storage for the companion box drying program. Clearly, consolidation of thickened TSRU tailings occurred quite rapidly upon deposition.

Water drained quickly from thickened TSRU tailings. Only a minor amount of water remained in the thickened TSRU cell, Cell 4, at 21 hours after deposition (Figure 3), while a significant pond of water remained in Cell 1 at 19 hours (Figure 5). Permeability tests in the sand base of the two cells reported similar values: 1.6 x 10^{-3} cm/s at Cell 4 and 3.2 x 10^{-3} cm/s at Cell 1, so the difference in performance is attributed to the difference between the properties of thickened and raw TSRU tailings rather than to drainage differences through the sand base and walls.

Settlements and Slopes

Untreated and in-line flocculated TSRU tailings in Cells 1 and 2, respectively, displayed similar settlement behavior. In each, there were segregated coarse and fine fractions, with negligible settlement of the coarse beach material, and significant prolonged settlement of the pooled fine slurry. This is illustrated in Figure 6; post A was located on the segregated coarse beach and post B on the steep slope down to the slurry pool, while posts C and D (for which the settlement lines overlap in the plot) were in the pooled fine slurry.

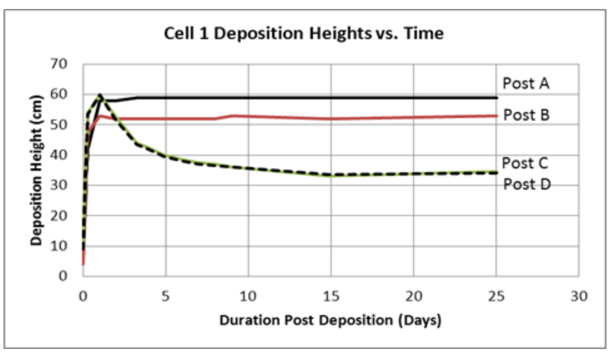

Figure 6. Cell 1, Deposit Height over Time.

Thickened TSRU tailings in the other cells were characterized by quick settling of two days or less. The settlement was roughly uniform in magnitude over the cell length, as there was no segregation (Figure 7). It should be noted that settlement behaviour is dependent on deposit thickness and may also be affected by cell geometry. Therefore, caution is advised using the 2011 data for commercial design.

Surface slopes of segregated TSRU tailings (untreated in Cell 1 and in-line flocculated in Cell 2) were distinctly different between the coarse and fine components. In these cells, tailings flow and the resulting surface slopes were governed by material properties of the coarse and fine components. High slopes were developed on the segregated coarse beach (US side), but very low slopes were recorded in the fine slurry on the DS side.

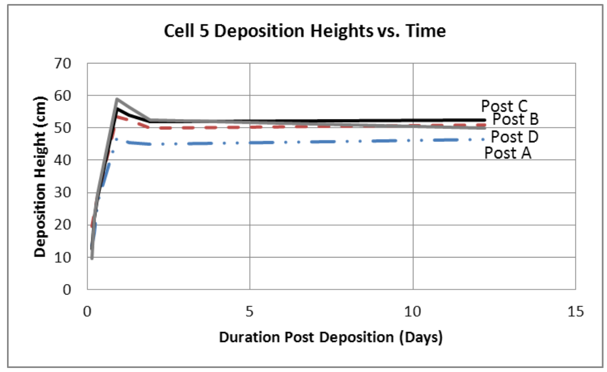

Figure 7. Cell 5, Deposit Height Over Time.

In Cells 3, 4, 5, and 7, with thickened TSRU, the tailings surface longitudinal profile was generally concave due to channelization of slope formation. Recorded slopes varied from 5-12% on the US side to 2-3% on the DS side. It was assessed that the cell geometry had been a constraint for development of tailings slopes and that the deposition was more like flow into a pool than flow across a beach. The recorded slopes are significantly steeper than, and consequently should not be taken as indicative of, slopes that would be created at full-scale tailings production.

Desiccation

Freezing temperatures before tailings deposition damaged installed soil suction measuring instrumentation and precluded further tensiometer monitoring of deposit drying. Based on visual observations, weather conditions at the time and the moisture of samples taken from the deposits, possible desiccation effects in 2011 were practically negligible. Measurement of negative pore pressures in the TSRU deposits was thus re-scheduled for the second phase of investigations in 2012.

Cracking was observed in all the deposits. In Cell 4, cracking of the US portion of the deposit initiated from the corners and depth marking posts within only about 20 h for Lift 1 (Figure 3), whereas there was no cracking after 20 h for Lift 2, even though the solids density during placement of Lift 2 was about 3% higher. Cracking in the second lift was evident after several days (Figure 4). The cracking behaviour in the other cells with thickened tailings was similar. Cracking was also observed in the segregated coarse tailings beach on the US side in Cells 1 and 2. In addition to water-loss-induced-cracking normally expected for sediments containing fines and clay-like materials, two other mechanisms might have contributed in this case. It was observed that thickened tailings cracked while still very wet. The tailings were discharged at temperatures of around 60°C, and within 20 hours, the upper 10 cm were below 10°C. Therefore, thermal shrinkage resulting from this 50+°C temperature change is thought to have promoted cracking. Also, breaking of the cohesive structure associated with flocculation is believed to have contributed to such cracking.

Freeze/Thaw

Changes in the temperature of the instrumented deposits were monitored by a thermistor string along the instrument post, with one end buried in the cell sand base, and the other suspended in the air above the tailings. Temperature data were thus collected continuously from the cell subgrade, up through the tailings deposit to the air above the deposit. Automatic collection and storage of thermistor data was performed by a separate DAS. The thermistor data were downloaded in 2011 before the complete freezing of deposits. In addition, freeze/thaw effects were monitored by the installed instrumentation over the winter of 2011-2012. These data will be collected and processed in 2012.

An indication of the freeze/thaw effect on TSRU tailings settlement that occurred during the winter-spring period of 2011-2012 relative to their respective deposit thicknesses before freezing is shown in Figure 8.

Again, two distinct patterns were revealed for untreated and in-line flocculated tailings (dashed lines for Cells 1 and 2) and thickened TSRU (solid lines in Figure 8). Freeze/thaw settlements were fairly uniform in magnitude and distribution along cell length in thickened TSRU deposits. In contrast, or Cells 1 and 2, the segregated coarse fraction that had formed a relatively stiff beach on the US side (Post 1 in Figure 8) settled about 5%, whereas the DS side, consisting predominantly of soft fine slurry that had pooled during deposition (Posts 3 and 4 in Figure 8), settled more than 20%.

Field Vane Shear Test

Due to time constraints, shear strengths in 2011 were measured only in Cells 1, 2 and 3, using a hand-held vane apparatus. The choice of testing locations was dictated by the access and the cracking pattern at the surface.

Vane strength in Cells 1 and 2 are presented in Figures 9 and 10, respectively. Measurements were performed about 15 days after deposition.

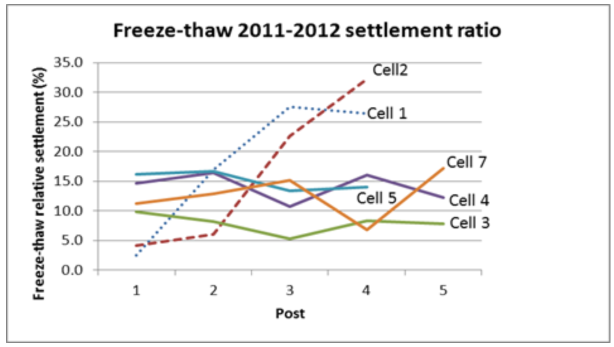

Figure 8. Settlement Due to Freeze/Thaw.

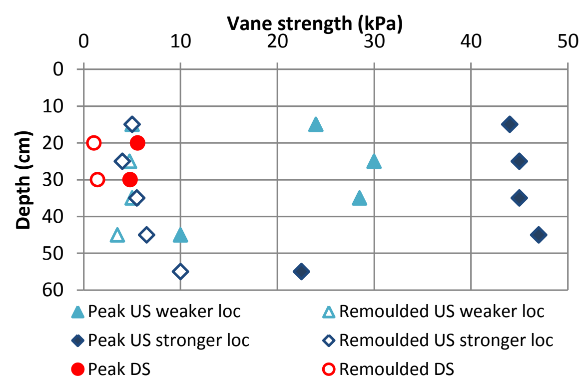

Figure 9. Vane Strengths in Cell 1.

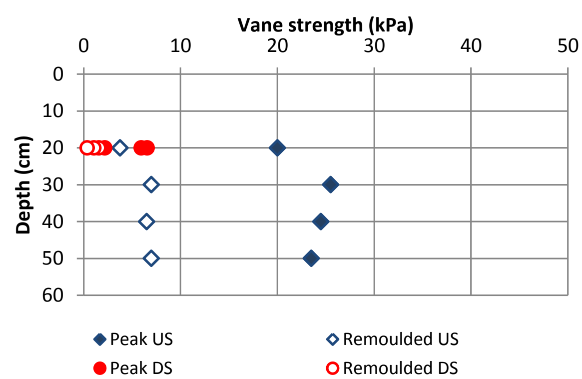

Figure 10. Vane Strengths in Cell 2.

The US test locations in both cells were situated on the segregated coarse tailings beach-above-water (BAW). Consequently, high strengths were measured.

The Cell 1 and 2 DS strengths were much less than the US strengths. The DS strengths were not measured in the soft sediment of the fine tailings pool formed during deposition because this location was still soft and therefore not accessible. Rather, they were located on the slope from the BAW to the pool. As would be expected, the slope samples had lower solids contents and sand-to-fines ratios due to segregation, and consequently lower strength than the BAW. However, measured peak strengths were still near or above 5 kPa, with the exception of a point on the far DS side in Cell 2 with the lowest peak strength – of only 2 kPa.

Vane shear strengths in Cell 3 were measured just two days after deposition of each lift. The data are plotted in Figures 11 and 12. Figure 12 includes data for both Lift 2 (above the 10-15 cm intermediate sand layer placed on Lift 1) and Lift 1 (below the sand layer).

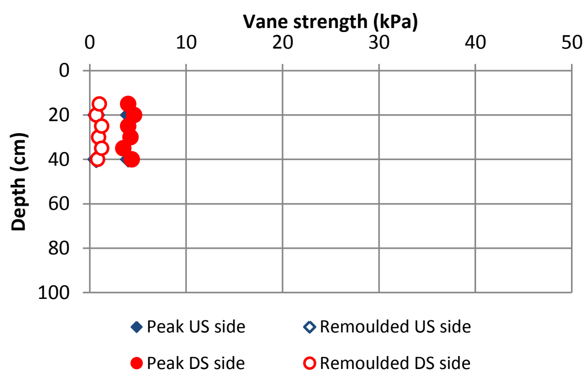

Figure 11. Vane Strengths in Cell 3, Lift 1.

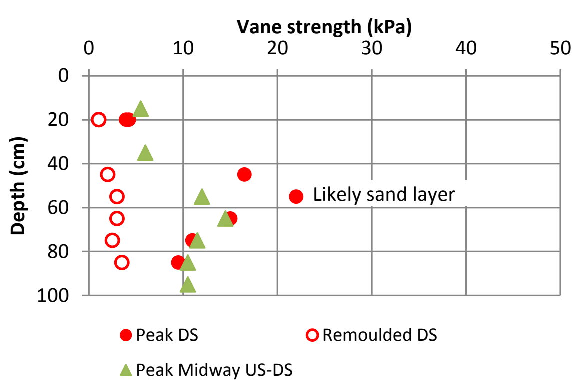

Figure 12. Vane Strengths in Cell 3.

For Lift 1, the US and DS strengths were virtually identical (DS sample points are obscured by US in Figure 11), meaning there was no sand segregation during deposition. Strengths of almost 5 kPa were achieved in just two days.

No US vane measurements were taken for Lift 2. Peak mid-location and DS strengths are very similar with depth, increasing with depth from about 5 kPa in Lift 2 to about 10 kPa in Lift 1, with higher values near or in the intermediate sand placed between lifts.

Finding similar strengths near the surface and at depth means there were no significant consolidation or drying effects, and likely points to the importance of flocculation or some "cohesiveness" due to bitumen binding the minerals. Strength sensitivity, expressed as the ratio of peak to remoulded strength, is significant for all three tailings types (non-treated, in-line flocculated and thickened).

Susceptibility to Surface Erosion

After 6 months of weathering, including a freeze/thaw cycle, significant surface erosion was evident in the BAW portion of Cell 1 (Figure 13) but, interestingly, very little in Cell 2 (Figure 14), perhaps attributable to the in-line polymer addition for Cell 2. No signs of erosion were noticed in thickened tailings.

Figure 13. Cell 1 in Spring 2012.

LABORATORY TESTING PROGRAM

The geotechnical sampling program had two stages: during- and post-deposition. During deposition, samples were collected by scoop from the cells at two locations, US and DS, and were tested for particle size distribution (PSD) to check for segregation due to shear during flow. After deposition, the samples were collected using a piston sampler, with full profiling over the deposit depth, and the cores were divided into several intervals. Each interval was tested for composition, PSD and Atterberg limits to obtain indications of possible segregation during tailings settlement in quiescent conditions.

Geotechnical Index Properties

Particle size distributions were determined by first applying the ASTM hydrometer method to a whole tailings sample (with bitumen), and then wet sieving the hydrometer residue to 0.074 mm. The untreated and in-line flocculated materials from Cells 1 and 2 showed noticeable segregation during deposition (Figure 15). In contrast, thickened tailings placed into Cells 3 and 4 showed negligible segregation during deposition (Figure 16).

These “during-deposition” findings (segregation in Cells 1 and 2 but not in the thickened TSRU cells) were corroborated by testing post-deposition samples. Those samples also demonstrated that the thickened TSRU tailings had not segregated in the post-deposition quiescent conditions.

Figure 14. Cell 2 in Spring 2012.

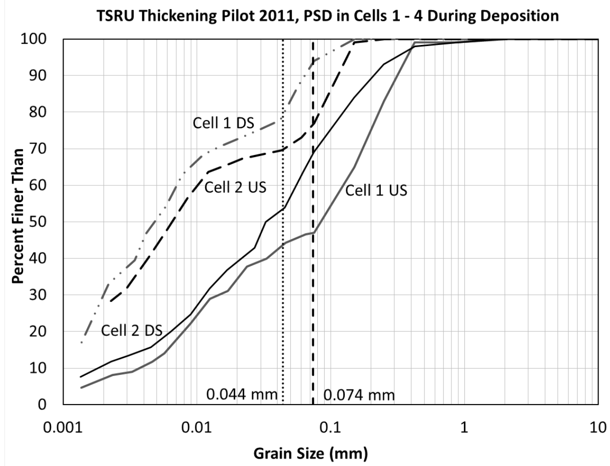

Figure 15. Cells 1 and 2, PSD during Deposition.

Measurement of in-situ density was performed in Cells 1, 2, and 3 using a piston sampler, with rough estimation of overcompaction during sampling. In Cells 1 and 2, the densities were 1.25 to 1.35 t/m^3 at the US side and 1.1 to 1.2 t/m^3 at the DS side. This is consistent with the observed segregation of coarser tailings US and finer DS and the small settlement reported US versus significant settlement DS. Sampling of Lift 1 in Cell 3 produced densities that were the same US and DS, ranging between 1.4 and 1.5 t/m^3. Substantially the same densities were reported for samples from both lifts in the final Cell 3 deposit. The generally higher densities for Cell 3 than Cells 1 and 2 reflect a material that did not segregate.

Difficulties were encountered determining the Atterberg limits using the standard ASTM procedures: Casagrande’s method for liquid limit (LL) and thread rolling for plastic limit (PL). Some tests could not be completed; others suffered from insufficient accuracy (such as higher LL than PL). These measurement problems precluded development of usable strength correlations with normalized moisture content for thickened TSRU tailings.

Two areas are recommended for future study on TSRU tailings: (1) explore the effects of flocculation and bitumen on geotechnical index properties, and (2) evaluate other methodologies for determination of geotechnical index properties.

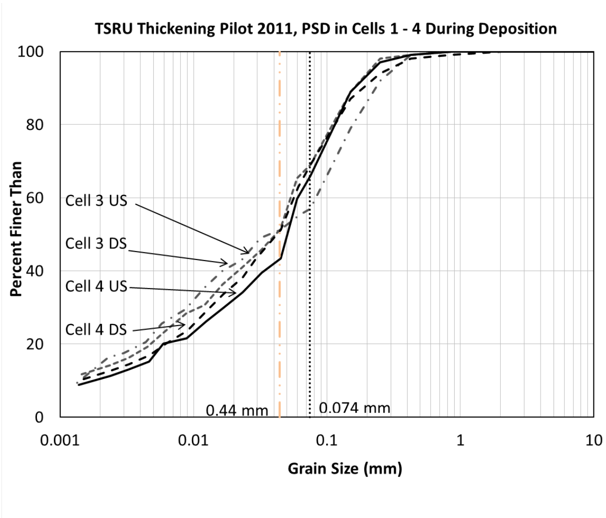

Figure 16. Cells 3 and 4, PSD during Deposition.

Consolidation

Large strain consolidation testing was conducted to produce laboratory-scale estimates of compressibility and hydraulic conductivity as functions of vertical effective stress and void ratio, respectively. All three tested samples were taken from thickened TSRU tailings in Cell 4. Two samples were tested in both intact and remoulded state, to demonstrate the effect that the breaking of the flocculated structure had on the consolidation behaviour and corresponding material properties.

The compressibilities of undisturbed samples were described by similarly shaped smooth curves, meaning similar compressibility of the tested samples. A small quasi-preconsolidation pressure of 20-40 kPa was observed in all undisturbed tests. It was most likely caused in part by flocculation, and is related to the strength of the flocculated structure. Remoulded samples did not show any pre-consolidation effects (Figure 17). Due to the potential significance of this effect, an experimental program on the effect of flocculation on the consolidation behaviour of thickened TSRU tailings is recommended.

Figure 18 shows a consequence of breaking of the flocculated structure, plotting the results of both intact and remoulded samples. The hydraulic conductivities for the remoulded samples were consistently about one order of magnitude less than for the intact samples. In other words, once the flocculated structure was broken down, the tortuosity of the pore spaces, even at the same void ratio as intact samples, was permanently affected. This finding may have important implications with regard to tolerable durations of slurry transport and shear environment at discharge.

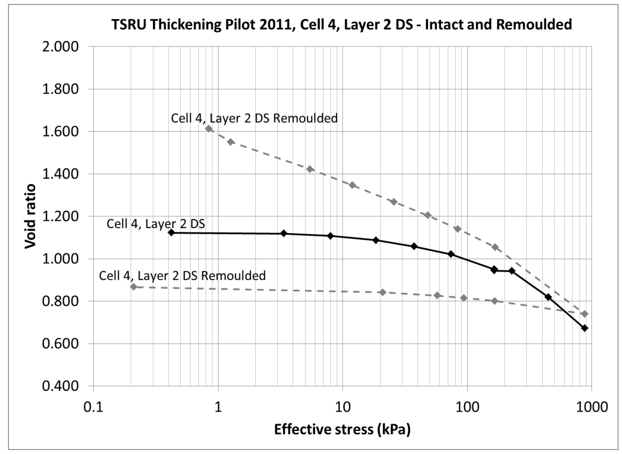

Figure 17. Cell 4, Compressibility, Intact vs. Remoulded.

Strength

Isotropically consolidated undrained and drained triaxial tests for estimation of the compression and shear strength properties of thickened TSRU material were performed on samples from Cell 4. The equivalent drained friction angles varied between 34° and 39° for effective stresses up to 400 kPa.

SUMMARY

Thickener Program

The TSRU tailings was thickened to 45-50% total solids (mineral plus asphaltenes) at throughputs up to 0.7 t/m^{2}h using 70-150 g/t SNF A3338, with corresponding overflow solids in the range 400-800 mg/L. The thickener operated well through variation in feed solids content to produce consistent underflow material. The yield stress of thickener underflow seemed to increase somewhat with increasing polymer dosage, but the thickened tailings were also found to be shear-sensitive.

Geotechnical Program

Flow and segregation during deposition: Untreated and in-line flocculated TSRU tailings segregated upon discharge, the latter to a lesser extent, although in-line flocculation was not optimized. Thickened TSRU tailings did not segregate over the length of test cells (6-7 m).

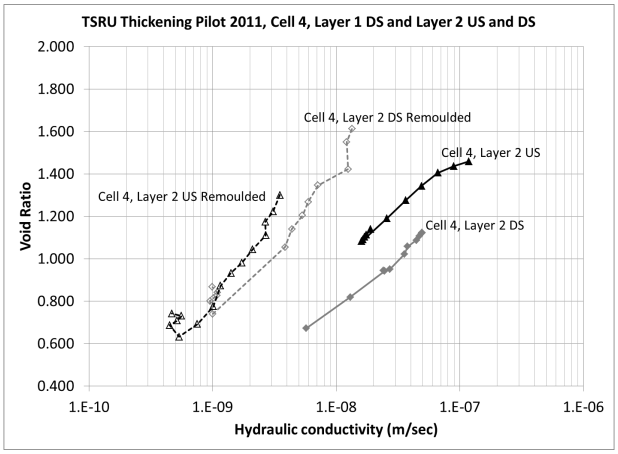

Figure 18. Cell 4, Hydraulic Conductivity, Undisturbed vs. Remoulded.

Deposit settlement and slopes: Untreated and in-line flocculated TSRU tailings segregated into coarse and fine fractions. There was negligible settlement of the coarse beach material, and significant prolonged settlement of the pooled fine slurry. Thickened TSRU tailings exhibited quick settling and were uniform in magnitude over the cell length because there was no segregation.

Surface slopes of segregated TSRU tailings (untreated and in-line flocculated) were steep for segregated coarse and flat for fine deposits. The tailings surface longitudinal profile for thickened TSRU was concave. Recorded slopes varied from 5-12% on the US side to 2-3% on the DS side. It should be noted that slope development was judged scale-dependent and that the 2011 data should not be directly used for design.

Freeze/thaw: Freeze/thaw settlement in untreated and in-line flocculated tailings (Cells 1 and 2) was characterized by little settlement in the segregated coarse fraction, and significant settlement where fine slurry was pooled during deposition. In contrast, freeze//thaw settlement of thickened TSRU was fairly uniform in magnitude and distribution along cell length.

Consolidation: Testing was performed on the thickened TSRU samples in their intact and remoulded states. A small quasi-preconsolidation pressure of 20-40 kPa was observed in all undisturbed tests. It was most likely caused in part by flocculation, and is related to the strength of the flocculated structure. Remoulded samples did not show any pre-consolidation effects. The hydraulic conductivity of deflocculated samples was an order of magnitude smaller than the conductivity of intact flocculated material.

Strength: The equivalent drained friction angles from triaxial tests on thickened TSRU tailings from Cell 4 varied between 34° and 39° for the effective stresses up to 400 kPa.

ACKNOWLEDGEMENTS

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Session 9

Environmental Aspects

NOVEL POLYMERIC ADSORBENTS FOR REMOVAL OF NAPHTHENIC ACIDS FROM OIL SANDS PROCESS AND TAILINGS WATER

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ABSTRACT

Naphthenic acids (NA) are among major toxic species in the oil sands process-affected water (OSPW). Given the estimated oil sands reserves, millions of liters of OSPW will be produced. The zero liquid discharge policy for the Canadian oil sands industry has led to major environmental and industrial concerns and demonstrated a need for viable strategies that can remove NA from OSPW. The carboxylic acids obtained in crude oil are christened as 'naphthenic acids' in the petroleum industry terminology. Generally, naphthenic acids are a mixture of different alkyl substituted cyclo-aliphatic carboxylic acids with acyclic aliphatic acids as minor components. These compounds are highly toxic to aquatic life, algae and mammals.

During oil sands extraction using Clark's hot water extraction process a considerable amount of NA is solubilized in the OSPW. Removal of naphthenic acids from OSPW is of primary concern in oil sands mining operations. In this paper we present a comparative evaluation of an adsorbent technology to the use of solid phase extraction materials. The use of fluorescence spectroscopy has demonstrated that a novel polymeric adsorbent was effective at reducing the amount of NA-type substances, although further studies are necessary to bring NA concentrations down to non-toxic levels.

INTRODUCTION

In the surface mining of oil sand, the conventional process of extracting bitumen involves slurrying the oil sand ore with warm or hot water to release bitumen from sand grains (Masliyah et al, 2004). In the alkaline environment of the process provided by addition of caustics, organic acids in the bitumen are neutralized to form natural surface active agents soluble in water (Sanford and Seyer, 1979). After collection of bitumen in a series of froths, the remaining water, organic and inorganic contaminants, and solids are transferred to ponds as tailings. The tailings cannot be discharged into the environment owing to the concentration of naphthenic acids and other toxic species present in the tailings water.

OSPW may contain in excess of 100 ppm of NA, (Headley et al, 2007) whereas the concentration of naphthenic acids in the Athabasca River is less than 1 ppm. At these levels of NA, OSPW can be considered potentially toxic to aquatic life (Kavanagh et al, 2009).

The traditional definition for naphthenic acids is a carboxylic acid containing one or more saturated ring structures with the formula $C_nH_{2n+z}O_2$, where z is the described as the hydrogen deficiency and is a negative, even integer (Lai et al, 1996). In the petroleum industry, naphthenic acids are described as a mixture of a wide range of alicyclic and paraffinic organic acids found in crude oil (Brient et al, 1995). When these acids are solubilized to produce in-situ surfactants, the surface active characteristics lead to toxicity concerns (Headley et al, 2007). Although discharge limits for NA have not been set in Canada, a good guideline would be to reach the levels in local surface- and groundwater of ~1-5 ppm (Allen, 2003).

One method for dealing with the toxicity of the OSPW is water-capped lakes. In this method, tailings are transferred to mined-out pits and covered with a layer of surface water. While in the pits, bioremediation is expected to reduce the toxicity of the water to levels which would be safe to aquatic life within 1-2 years (Kavanagh et al, 2009). However, a very limited knowledge is available on this method and the ability to populate the wet landscapes with aquatic life is still under investigation. Furthermore, the extensive time period involved may not be practical if direct discharge is required at some point in the future. Another probable concern is resistance of higher molecular weight NA to bioremediation (Allen, 2003). Under above circumstances it is imperative to evaluate alternate technologies capable of removing naphthenic acids.

Fluorescence spectroscopy techniques are used to detect compounds such as proteins, surfactants, humic and fulvic acids, phenols, polyaromatic hydrocarbons, and oils. These compounds exhibit characteristic fluorescent emissions after excitation by visible or near-UV light. This method has been shown as a fast and reliable technique for monitoring the presence of naphthenic-type compounds in OSPW (Mohamed, 2008; Kavanagh, 2009; Rowland, 2011). Fluorescence signatures are not observed in the classical naphthenic acids due to absence of aromatic rings. However, the organic compounds obtained in the OSPW exhibit fluorescence. The probable reason for exhibition of such fluorescence behaviors have been attributed to association of aromatic contaminants in the OSPW along with NA (Kavanagh, 2009). The structures of the organic compounds in the OSPW were analyzed using atmospheric pressure photoionization and a broader range of NA with higher hydrogen deficiencies was observed (Barrow, 2010). These could potentially be naphthenoaromatic type compounds formed due to the interaction of aromatic contaminants and naphthenic acids co-existing in the oil sands process affected water (Barrow, 2010).

Solid phase extraction (SPE) media is an efficient means of extracting contaminants from wastewater (Richardson and Ternes, 2011). The basis of SPE is the affinity of contaminants in a liquid phase with functional groups grafted on mineral surfaces of the extraction media. The contaminants which are adsorbed on the SPE media can then be extracted with solvents to be analyzed by analytical techniques such as GC-MS (Jeanneau et al, 2007). The use of C_{18} reverse-phase SPE and SAX quaternary amine ion-exchange SPE have been utilized for the removal of traditional naphthenic acids (Jones et al, 2001). It has also been shown that reducing the pH to 2.5 followed by reverse-phase solid phase extraction with C_{18} could remove acute toxicity of settling basin water (Verbeek et al, 1993). Measuring the fluorescence of OSPW before and after eluting through different SPE could be used to view what type of functional groups are effective at removing these naphthenoaromatic structures from OSPW.

EXPERIMENTAL

Materials:

A sample of commercial NA (Naphthenic Acids, pract) was obtained from Aldrich and was known to be a collection of NA structures. Additional NA samples of defined molecular structure were also received from Aldrich. The samples include:

1. Cyclohexanecarboxylic acid
2. Cyclohexanepropionic acid
3. Cyclohexanebutyric acid

Oil sand, OSPW, recycle water samples, and mature fine tailings (MFT) were obtained from oil sands producers in Alberta, Canada. Hydrochloric acid and sodium hydroxide were purchased from Fisher Scientific. SPE tubes were purchased from Varian Inc (Agilent Technologies).

Procedures:

For the commercial NA solutions, deionized water was heated to 50 °C and the pH was adjusted using sodium hydroxide solution. NA was added and stirred for one hour and then aliquots were taken for further experimentation. The addition of a novel polymeric adsorbent KA (Kemira, USA) was made while mixing the test water on a magnetic stir plate and blending for 1 hour. The treated solutions then rested overnight before sampling for analysis. Processing of oil sand ores was performed in a Denver flotation cell as described elsewhere (Mahmoudkhani et al, 2012). The resulting tailings water was filtered through a 0.1 μm filter before analysis.

When screening water samples by SPE, a washing step was first performed by delivering 20 mL of deionized water through the tube. To flush any remaining deionized water, 5 mL of the test sample was passed through the tube and discarded. An additional 10 mL was then delivered and collected for analysis by fluorescence spectroscopy to determine the extent to which NA had been removed from the water. Descriptions of the SPE media can be found in Figure 1. Further details on the mechanism for each are provided in the Results section.

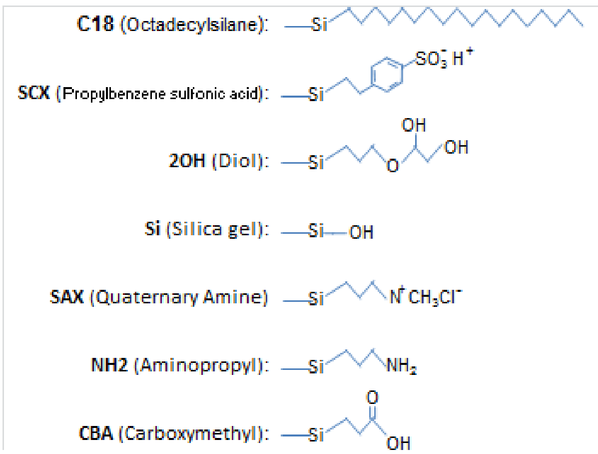

Figure 1. Solid phase extraction chemistries.

Fluorescence Spectroscopy

A Shimadzu RF-5301 Spectrofluorophotometer with Panorama 3.1 fluorescence software or a Varian Cary Eclipse Fluorescence Spectrophotometer with Cary Eclipse software was used for spectrophotometric measurements. All samples were measured in a quartz cuvette. Excitation and emission monochromator slit widths were set at 5 nm. The scan rate was 5500 nm/min for 3D contour plots of OSPW and a solution of commercial NA. Once the excitation wavelength of the NA signal was determined, emission wavelengths were scanned at a set excitation wavelength and a scan rate of 600 nm/min.

RESULTS AND DISCUSSION

Comparing the 3D fluorescence spectra for water in Figure 2 and the commercial NA sample at 200 ppm in Figure 3, the peaks attributed to NA were determined. The other peaks observed in both spectra can be attributed to the Rayleigh-Tyndall effect (Hudson et al, 2007).

The excitation wavelengths below 400 nm can be viewed more closely in Figure 4. The arrow points to the peak fluorescence for the commercial NA, which was at an Ex/Em wavelength of 230/340 nm. This range was earmarked as the NA peak.

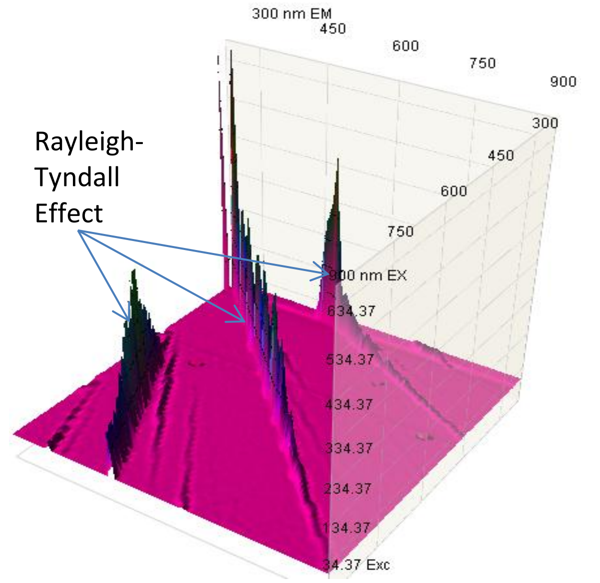

Figure 2. 3D contour plot for water.

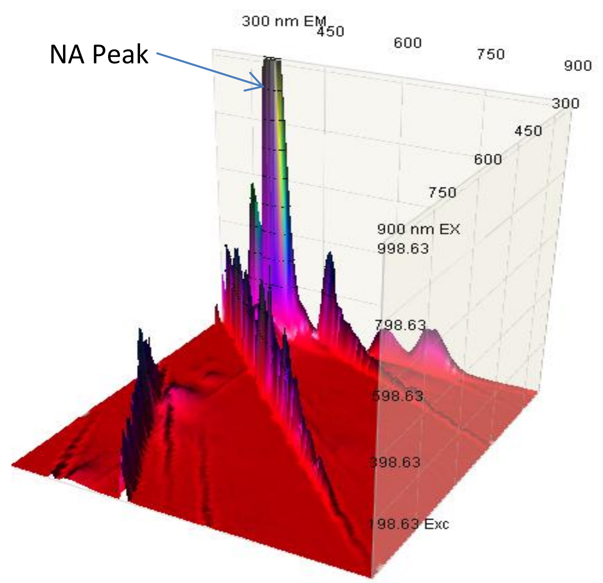

Figure 3. 3D contour plot for 200 ppm commercial NA; pH 9.

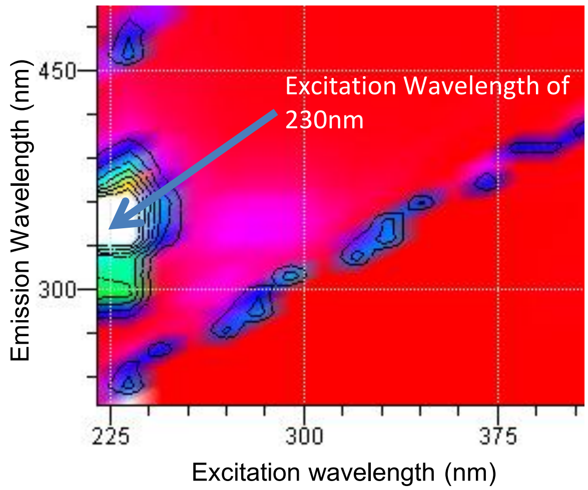

Figure 4. **Contour plot for 200 ppm commercial NA; pH 9.**

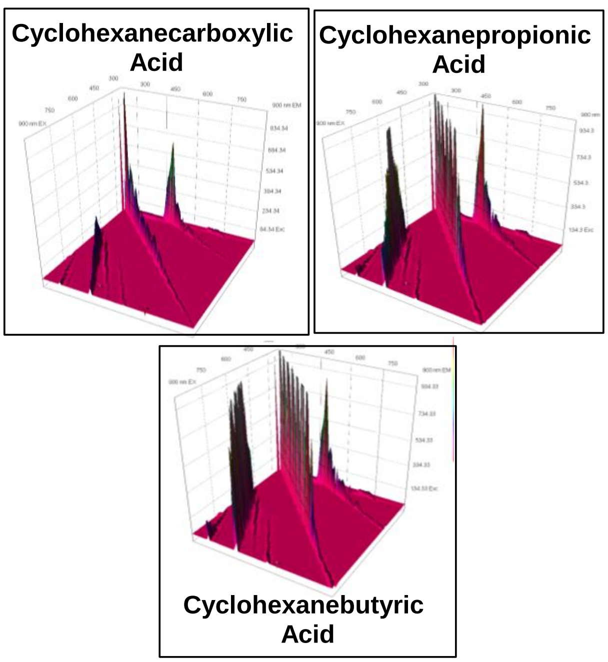

Figure 5. **3D contour plots for defined naphthenic structures at 200 ppm, pH 9.**

Several compositions of traditional naphthenic acids were analyzed after being added to water at the same dose and pH as the commercial NA sample. The results are shown in Figure 5.

These compositions do not contain aromatic rings, and as expected did not fluoresce. This supports the concept that what are referred to as NA in the petroleum industry may often include naphthenoaromatic structures that are not NA by traditional definitions.

A series of dilutions were made to gain an estimate on emission intensity versus concentration of napthenoaromatics in solution. Starting with a 200 ppm commercial NA sample at pH 9 and 11, dilutions were made down to NA concentrations of 25 ppm. Scans of each dilution sample were made at Ex/Em wavelength range 230/300-400 nm and the results are shown in Figures 6 and 7.

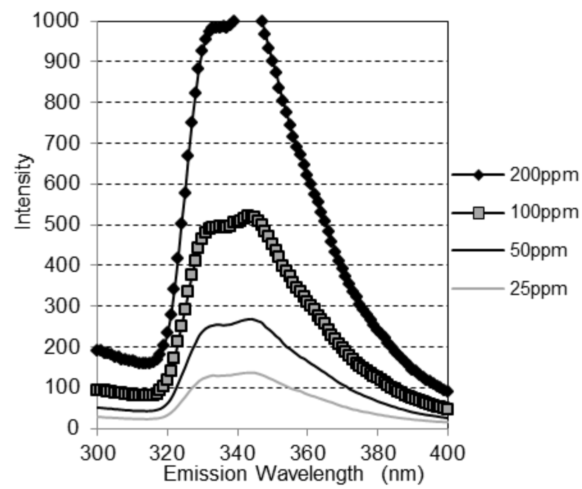

Figure 6. **Dilutions at pH 9. Ex λ = 230nm.**

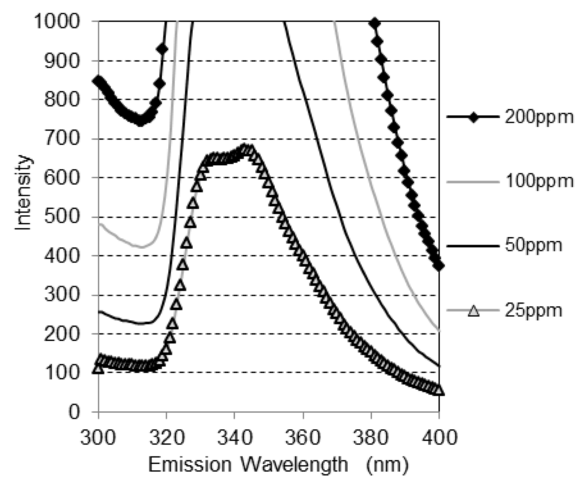

Figure 7. **Dilutions at pH 11. Ex λ = 230nm.**

For pH 9, the peak height at 200 ppm was just beyond the intensity limit for the instrument but all other doses were measurable across the range of emission wavelengths. With the correlation between dose and peak intensity, it would be feasible to make a dilution calibration curve for treatment comparison, although it should be noted that fluorescence spectroscopy should only be considered as a semi-quantitative method (Kavanagh et al. 2009). The influence of pH on solubility was demonstrated when observing the results at pH 11, where peak heights were more intense at each dose and remained outside the detection limits at doses equal to and above 50 ppm.

To determine if OSPW would present similar fluorescence peaks, a low-grade oil sand was processed in the Denver cell and the tailings water filtered through a 0.1 μm filter. The Denver cell tailings water was compared to OSPW of an oil sands producer, recycle water from an alternate producer, and water separated from mature fine tailings (MFT) by centrifugation and filtration without chemical treatment. Scans of Ex/Em wavelength (nm) 230/300-400 were compared in Figure 8.

Naphthenoaromatics were present in each of the process waters. The lower intensity shown in the Denver cell tailings was most likely due to the test conditions of the flotation experiment which are more dilute than the processing of oil sand in the hydrotransport pipeline. It is important to note that the MFT water had shown the most intense peaks. In the future, as water is recovered from MFT via various treatment methods, it would be important to track the levels of naphthenoaromatics present before reuse or discharge as this appeared to be a concentrated source of the chemical class.

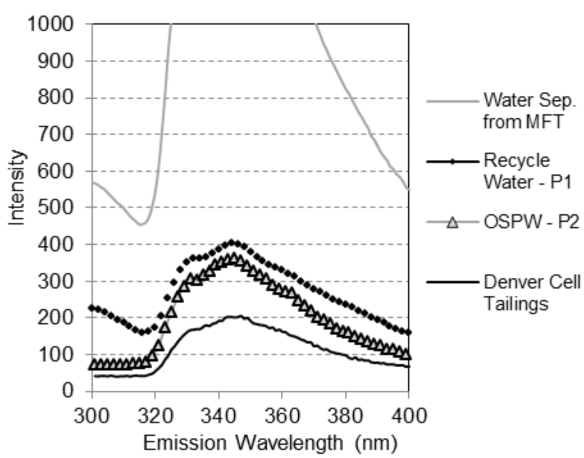

Figure 8. Fluorescence spectra for process-affected waters. Ex λ = 230 nm.

The 200 ppm commercial NA sample was dissolved in water at pH 11 and used for the treatment tests with KA. After the mixing and settling steps for these procedures, the water was filtered through a 0.1 μm filter. Note that in the use of KA, the solid absorbent rapidly settled after stopping agitation so recycling of the supernatant would not be hindered by suspended particles. The effect of the treatment is shown in Figure 9.

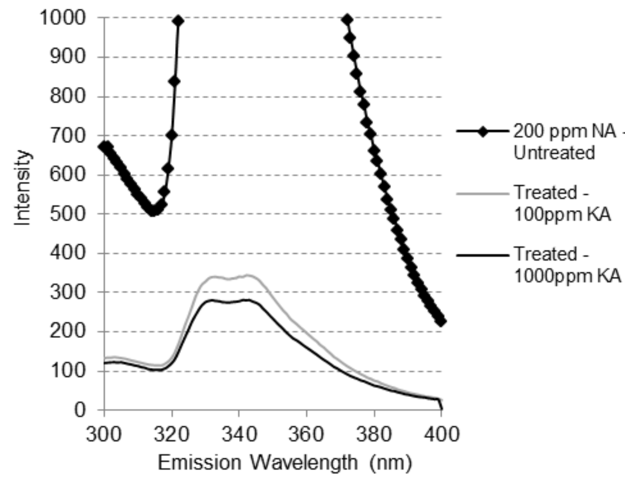

Figure 9. Addition of Kemira Adsorbent. Ex λ = 230 nm.

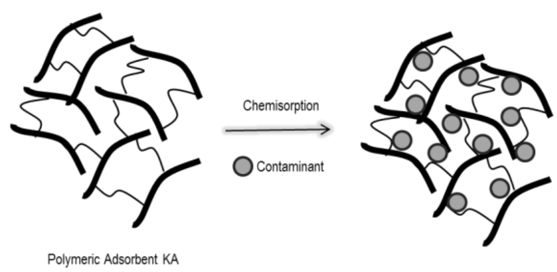

Figure 10. Proposed mechanism of NA removal by KA adsorbent.

The addition of KA provided significant reduction in the concentration of naphthenoaromatics. Using the dilution data of Figure 7 as an estimate of concentration, the decrease of peak intensities at 320 to 360 nm indicated there was a reduction from 200 pm to less than 15 ppm.

The mechanism of NA removal by polymeric adsorbent KA is schematically shown in Figure 10. KA comprises of active sites capable of chemisorption of NA molecules.

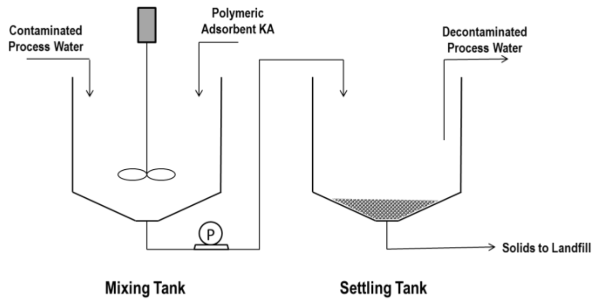

Figure 11. Schematic representation of water treatment in this work.

The process for water treatment, as schematically shown in Figure 11, involves chemisorption of the contaminated species, NA, on active sites of the polymeric adsorbent in a single stage treatment in a continuously stirred mixing tank. Polymeric adsorbent KA may be dosed at 50 – 500 ppm to the contaminated water based on the level of contaminant. After 1 – 3 hours mixing, the aliquot is transferred into a gravity settling tank to allow suspended solids precipitation. The clear supernatant is a decontaminated process water and may be reused as make up water or undergo further treatment for discharge, if necessary. Faster solid separation may be achieved by use of flocculants aids or mechanical separation by filtration or centrifugation. The separated solids may be safely landfilled.

The 100 ppm commercial NA solution shows a peak intensity of approximately 400 at the Ex/Em wavelength of 230/340. The reduction of peak intensity after passing through various SPE media is depicted in Figure 12.

There were varying levels of reduction when comparing the peak intensities for emission wavelengths 320 to 360nm. C_{18}, a monomerically bonded reverse-phase extraction media, performed poorly in comparison to other SPEs. The interactions are based on non-polar / non-polar attractive forces which did not appear to be in effect. Monomerically bonded C_{18} can potentially have lower performance when attempting to retain hydrophobic compounds with hydrophilic functionality, such as aromatics (Supelco, 1998). The silica (Si), aminopropyl (NH_2), and diol (2OH) are all normal phase extraction media and are not expected to provide the strongest interaction since they are typically used for adsorption of materials from non-polar systems. The three ion exchange media performed well, indicating the removal of charged compounds. SAX is a strong anion exchanger, suggesting that once dissolved in water the fluorescent NA structures contained anionic sites which interacted with the SPE media. Reduction improved as the media was changed to CBA and even more so with SCX, a weak and strong cation exchanger, respectively. In order to obtain the largest reduction in peak height for the naphthenoaromatics present in the commercial NA sample, the addition of material with anionic functional sites have the strongest interaction with cationic sites of the NA in solution.

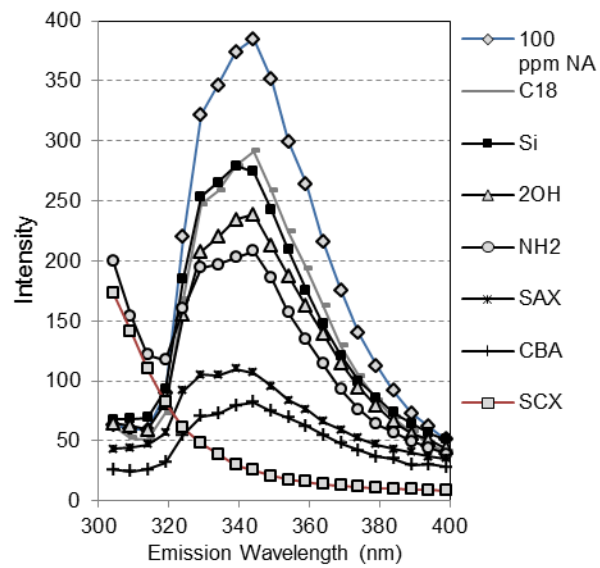

Figure 12. SPE of 200 ppm commercial NA; pH 9. Ex λ = 230 nm.

CONCLUSIONS

The new polymeric adsorbent KA utilizes both physical and chemical characteristics for removing NA from OSPW. It is proposed that NA is chemically adsorbed on the active sites within the cavities of the sorbent material that separates the contaminant from process water when insoluble products precipitate. Treatments with KA provided a significant reduction of naphthenoaromatics from 200 ppm to less than 15 ppm in a single step. Removal by adsorption is a fast method for reduction of contaminants; however, complete removal may require additional steps. The chemical nature of the adsorbent system is diverse and can be tailored to further reduce the concentration of the contaminant from process water.

This technology was evaluated for removing NAs in comparison to organic adsorbing media. The use of solid phase extraction has pointed to a number of functional groups which are able to bind with naphthenoaromatics moieties. An increase in the anionicity of the SPE functional groups resulted in the largest reduction in peak intensity.

Results from this study show that polymeric adsorbent KA can potentially offer a cost effective and versatile option for treatment of contaminated oil sands process water. Work is ongoing to evaluate this technology for treatment of other contaminates in process waters.

Tracking the fluorescence spectroscopy of OSPW and commercial NA at Ex/Em wavelength (nm) of 230/300-400 is a quick and efficient method for observing naphthenoaromatics concentrations.

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FATE OF NAPHTHA COMPONENTS IN OIL SAND TAILINGS

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ABSTRACT

The rate at which volatile organic carbon compounds (VOCs) leave oil sand tailings ponds has implications for operators and regulators concerned with air quality. The mobility of VOCs also has to be understood for reclamation when solvent-containing tailings may be moved about and re-handled. In addition, an understanding of the factors that affect VOC release from tailings ponds would help oil sand operators in designing tailings containment and handling of solvent-rich tailings such as froth treatment tailings.

In this report n-heptane and toluene, which are two components with usually the highest concentration in naphtha, were used to characterize the behaviour of volatile hydrocarbons in multiphase systems.

Impinging jet experiments showed that irrespective of any added mineral or mineral-bitumen, both heptane and toluene showed an initial rapid release from the water, which gradually decreased, with heptane being much more rapidly released than toluene.

When heptane and toluene were mixed with MFT and capped with water, their release from the MFT exhibited a lag period where there was not much change in MFT solvent concentration. However the concentration of heptane and toluene in the MFT eventually showed a linear decrease in concentration with about 80% of heptane being released within 9 months, and between 25% (frozen during winter) and 75% (never frozen) toluene being released.

INTRODUCTION

Oil sand tailings contain volatile hydrocarbons from solvent that is not completely recovered from froth treatment tailings. Solvent sent to tailings ponds can end up in the pond water, adsorbed on mineral surfaces or dissolved in residual bitumen. Solvent which is not decomposed by bacteria can be released to the atmosphere through diffusion and convection. The objective of this research was to determine and model the factors that affect volatile hydrocarbon retention in tailings deposits.

The fate of VOCs in oil sands tailings is ultimately determined by the thermodynamics of partitioning between gaseous, aqueous, solid and hydrocarbon phases, and the kinetics of these processes. In this study, a representative alkane hydrocarbon (n-heptane) and aromatic hydrocarbon (toluene) were used to characterize the behaviour of volatile hydrocarbons commonly found in naphtha. The evaporation of these VOCs from water phase was studied using an impinging jet apparatus and in addition, two mini-ponds were built to simulate solvent loss in a tailings pond. One of the pools was placed outdoor, to determine the impact of weather on the release of VOCs. The other pool was kept indoors to minimize weather effects.

EXPERIMENTAL

Solvent release kinetics

Figure 1 shows the experimental setup used to study the kinetics of VOC release from water, and water with added solids (sand, clay, or bitumen-coated clay). It consists of a glass cylinder connected to a nitrogen source and a gas chromatograph (GC). Solvent-saturated water (or solids suspension) was added to the cylinder, keeping headspace to a minimum, and then left for 2 h to equilibrate with the headspace. A laminar flow of nitrogen was directed at right angles to the water surface, and the gas then exited to the side at the top and was carried to the GC. The total amount of solvent released during any time period was calculated from integration of the solvent peaks from the GC chromatogram. When solids were present, they were allowed to settle to the bottom of the cylinder after the set up.

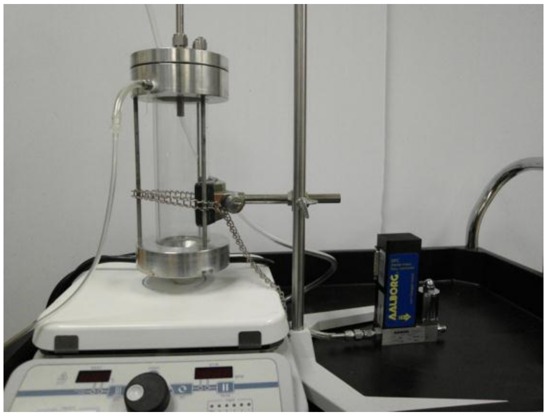

Figure 1. Apparatus used for solvent release experiments.

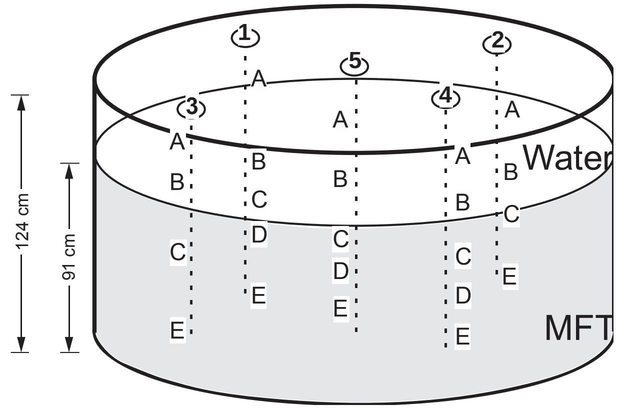

Figure 2. Sampling locations for pools.

Figure 3. Collecting samples from outdoor pool.

VOC release from MFT pools

In order to approximate the VOC release from a tailings pond, an outdoor and an indoor pool were set up using mature fine tailing (MFT), solvent and tap water. The pools had a volume of 5 m^3, and were 2.4 m in diameter.

To fill the pools, MFT was homogenized in a tank and then pumped to a second tank. When the second tank was almost full, solvent was added and the MFT and solvent gently mixed for 30 minutes. After mixing, a thin layer of MFT was added on top to reduce solvent loss during a 2-day equilibration period. After 2 days, the MFT was mixed for 1 more hour before being pumped into the pool. Immediately after all the MFT had been added, tap water was carefully layered on top. The finished pool had a water layer about 33 cm deep over an MFT deposit about 91 cm deep. The outdoor pool was set up in November 2011 and the indoor pool was set up in January 2012. The pools were sampled at 4 or 5 depths (A to D or E) depending on the location (1 to 5) (Figure 2). A and B were in the water layer while C, D and E were (initially) in the MFT layer. After spring-thaw, the solid/water interface in the outdoor pool had dropped below the C-depth. The water level was kept constant during the course of the study by adding tap water to make up for evaporation losses.

Solvent content was determined in the MFT samples by analyzing o-xylene extracts on a GC-FID. An aliquot from the same extract was used to determined bitumen content by gravimetry. To measure the solvent concentration in the water the sample was injected directly into a GC-FID.

RESULTS AND DISCUSSION

Solvent Release Kinetics

Assuming that transport of gas across the air/water interface is a steady-state process, it follows that:

$$F = k_g(c_g - c_{sg}) = k_l(c_{sl} - c_l)$$

where F is the flux of gas through the gas-liquid surface, c_g is the VOC concentration in the gas phase, c_{sg} is the VOC concentration in the gas phase near the surface, c_{sl} is the VOC concentration in the liquid phase near the gas-liquid surface and c_l is the VOC concentration in the liquid phase. k_g and k_l are the exchange constants for the gas and liquid phases, respectively.

If the exchange gas obeys Henry's law, then $c_{sg} = k_H C_{sl}$, where k_H is the Henry's Law constant. A three-compartment model was built using Comsol software to simulate fluid dynamics in free and porous media. The modeling results can be used to compare with the experimentally obtained data. Based on modeling of other alkanes, the concentration of VOC in the water phase should decrease exponentially with time.

The initial concentrations of heptane and toluene in water were 8 ± 1 mg/L and 450 ± 25 mg/L respectively. Figures 4 and

Figure 5 show the release with time of heptane and toluene from water, water with sand, water with clay (kaolinite) and water with clay that has adsorbed bitumen (2.5 wt%) on its surface. The data are the concentrations of hydrocarbon in the purge gas from the headspace of the impinging jet apparatus. The results show that heptane was rapidly released, and that there was essentially no heptane detected in the headspace after 10 minutes, for all systems. Toluene was released more slowly and was still evolving after 30 h. The presence of a mineral phase did affect the rate of release but not the ultimate concentration for heptane. For toluene the slowest release of was from the water-clay system but as these are preliminary data, this has to be checked before any conclusions can be drawn, especially as toluene release from the water-bitumen/clay system, was about the same as that from pure water.

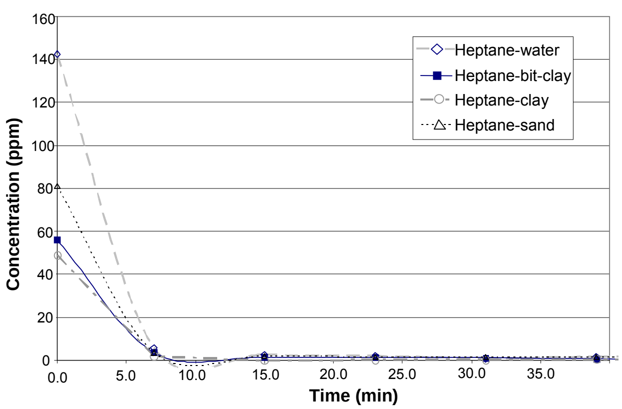

Figure 4. Graph of release of heptane from water saturated with heptane.

VOC release from MFT pools

The amount of toluene and heptane that was added to the MFT was about 5-times what one would calculate based on 4 vol. solvent per 1000 vol. bitumen production. The initial compositions of the MFT in the pools is given in Table 1.

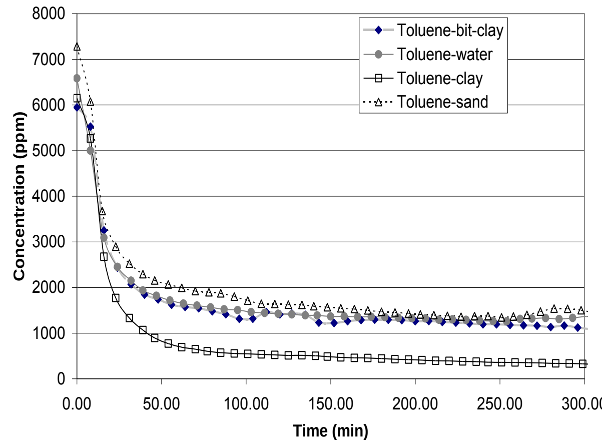

Figure 5. Graph of release of toluene from water saturated with toluene.

Table 1. Initial Composition of MFT in Pools.

	Concentration (wt%)	
	Indoor Pool	Outdoor Pool
MFT bitumen	3.0 ± 0.3	2.5 ± 0.2
MFT solids	37.9 ± 0.9	37.1 ± 0.3
MFT water	57.2 ± 0.7	57.8 ± 1.3
Heptane	0.074	0.089
Toluene	0.076	0.072

The outdoor pool froze shortly after being filled in November. Samples taken since spring-thaw showed a continuing evolution of toluene (Figure 6) from the MFT into the water[1] and, what appeared to be, a trend of decreasing heptane (Figure 7) and toluene in the MFT (Figure 8). We roughly estimate that about 25% of the toluene had

[1] Heptane in all water samples tested was below the detection limit of our method (1.0 ppm), but, as heptane was disappearing from the MFT we assume that the transport of heptane from the water phase to the air is much more rapid than from MFT to the water phase.

been released between November 2011 and August 2012 and about 75% of the heptane.

As one might expected the behaviour of the indoor pool was different than the outdoor. For a 3-month period, toluene in the surface water of the indoor pool was below the detection limit (0.1 ppm) (Figure 9). As the MFT showed no change in toluene concentration, no toluene (or heptane) was being released from the MFT. Since that period however, there was a steady release of toluene from the MFT to the water. As mentioned, the concentrations of heptane (Figure 10) and toluene (Figure 11) in the MFT showed little change for about 3 months, but since then have declined steadily. We estimate that about 80% of the heptane has been released and about 75% of the toluene.

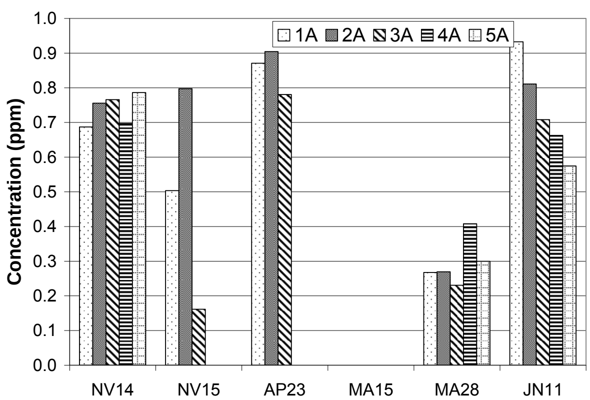

Figure 6. Toluene in water layer of outdoor pool.

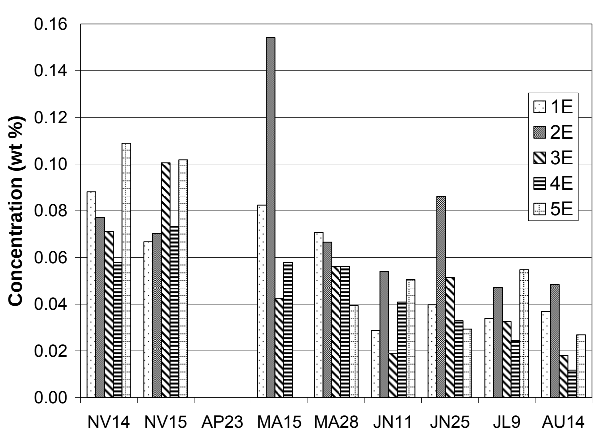

Figure 7. Heptane concentration in lowest MFT layer in outdoor pool.

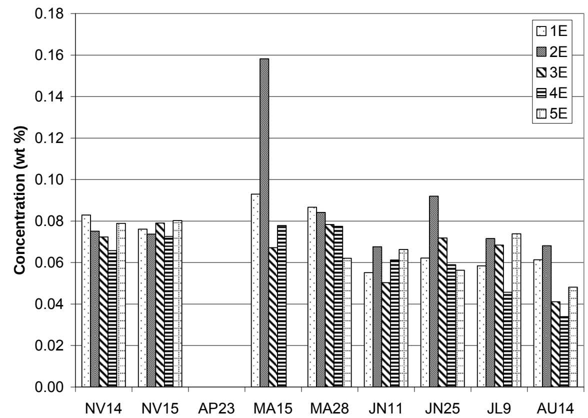

Figure 8. Toluene concentration in lowest MFT layer in outdoor pool.

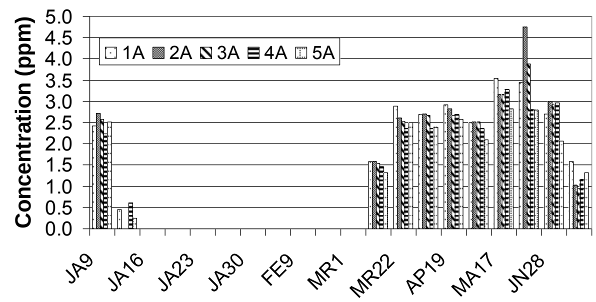

Figure 9. Toluene concentration in water layer of indoor pool.

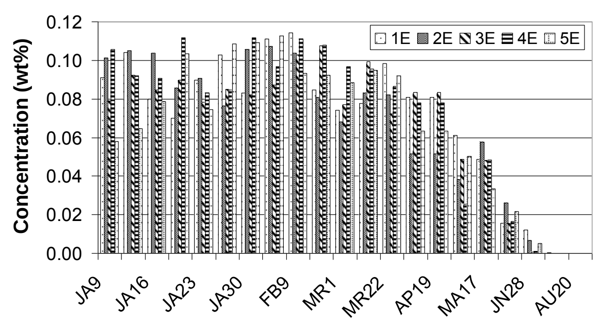

Figure 10. Heptane concentration in the lowest MFT-layer of the indoor pool.

Composition of Release Gases

A few months after setup, bubbles were observed rising to the surface of the indoor pool. Analysis of the gases showed mostly methane (Table 2), consistent with previous observations (1,2). A more detailed analysis showed that, aside from the expected methane, heptane, and toluene, there were also many branched linear and cyclic alkanes.

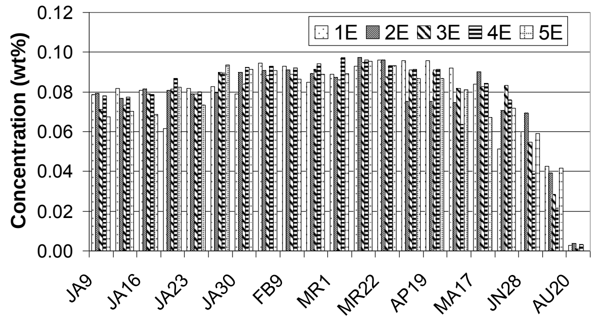

Figure 11. Toluene concentration in the lowest MFT-layer of the indoor pool.

Table 2. Composition of Gases Bubbling to Surface of Indoor Pool.

Compound	Concentration (vol%)	
	June	September
Methane	87	82
Carbon dioxide	7	3
Nitrogen	6	14
Hexanes	0.01	0.04
Heptanes	0.01	0.01

CONCLUSIONS

When movement of molecules is relatively unconstrained, as in the impinging jet experiments, heptane and toluene show an initially rapid release from water, that slows exponentially. Heptane was released from solution faster than was toluene, as one might intuitively expect from their Henry's Law constants, ($k_{H,\mathrm{heptane}}$ reported from 2272 to 833 bar kg/mol; $k_{H,\mathrm{toluene}}$ reported from 4.8 to 7.1 bar kg/mol) (3), although k_H is a thermodynamic constant, not a kinetic constant. The presence of a mineral phase seemed to slow the initial release of heptane but seemed to have no impact on the initial rate of toluene release. However, the subsequent release of toluene seemed to be affected by the mineral present.

One sees the effect of constrained transport pathways when the data from the impinging-jet experiments is compared with the MFT-pools. There was a lag period observed in both indoor and outdoor ponds during which there was no apparent release of solvent. However, once started, there was a steady, linear decrease of solvent in the MFT. We speculate that during the lag period, drainage channels were established in the re-settling MFT solids by bacterial action, or some other mechanism, which allowed the physical release of the solvent.

ACKNOWLEDGEMENTS

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THE ROLE OF RADAR-ACTIVATED WATERFOWL DETERRENTS ON TAILINGS PONDS

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ABSTRACT

Like other groups of animals, birds habituate to repeated threats that do not harm them. Consequently, they gradually learn to ignore continuously activated scare devices such as spinning lights, scarecrows, and propane cannons. Two approaches can be used to reduce or eliminate the rate of habituation. The first is to lethally reinforce threats; a practice that is unacceptable for tailings ponds. The second is to reduce bird exposure to threats that are not reinforced. Continuous or random activation of deterrents exposes birds to deterrent stimuli almost continuously, causing the birds to habituate rapidly. Activating the deterrents only when the birds are present greatly reduces their exposure to the deterrents and simultaneously causes them to associate offensive stimuli with a specific, defended location.

We have developed and deployed an operational wide-area, radar-activated deterrent system (RADS) using these principles in the Alberta oil sands. The implemented control and activation strategy is based on networked digital avian radars and deterrent devices that respond only to the presence of birds in designated locations. This approach provides defence of not only tailings ponds and other sensitive areas but also produces inner and outer defence perimeters. For most oil sands tailings operations, the sand beach and the water surface of these industrial facilities serve as the outer and inner perimeters, respectively. Responding to birds as they approach a sensitive area means that they can be turned away using long-range deterrents before reaching the sensitive area, offering greater protection. Those birds which penetrate the outer line of defence and continue are met by the inner deterrents.

A comprehensive information system at the heart of the RADS retains forever the movements of birds and their responses to the deterrents in support of risk assessment and management, investigations, reporting and information sharing, system performance assessment including habituation, system improvements, and R&D.

INTRODUCTION

Background

The Clark hot-water extraction process produces tailings ponds that retain residual amounts of bitumen. This bitumen can be hazardous or lethal to birds that contact or ingest it from a tailings pond or its adjacent shores. A variety of potentially useful deterrent technologies, used alone or in combination, are available to keep birds off the tailings ponds. These technologies differ in their efficacies.

One problem faced by tailings pond managers using deterrents to keep the birds away is preventing the birds from becoming used to the repeated application of deterrents. Because they constantly hear and see deterrents that activate on a regular or irregular schedule, birds learn that the threat is meaningless. They soon come to ignore the empty threats the deterrents represent; a behavior called habituation. On-demand activation of deterrents has been shown to prevent habituation and to prevent waterfowl from landing on small contaminated ponds (Stevens et al. 2000).

The Challenge

The challenge is to maintain the efficacy of the deterrents, by keeping the birds from habituating to and ignoring the deterrents without harming the birds. Habituation can be slowed or perhaps prevented by not continuously exposing the birds to the deterrents. The more “false alarms” a bird experiences each day, the more rapidly it will habituate to the stimulus. Slowing or avoiding habituation requires a mechanism to activate selected bird deterrents only when the birds are in position to land on a contaminated tailings pond. Birds that are migrating overhead should not activate the deterrents for two reasons. First, they are transient and will not attempt to land on the tailings pond. Their migration altitude is hundreds of meters above the terrain and directing deterrents at them is useless. Second, activating deterrents frequently (in response to migration which usually continues for several hours through the night) and in irrelevant situations to birds near the tailings ponds will produce more rapid habituation.

For the purposes of this paper, we say that a flight path, trajectory, or track of a bird or flock of birds is *geo-feasible* with a tailings pond landing if the bird's 3D location (latitude, longitude, and altitude), speed and heading make such a landing reasonably possible. Birds approaching a pond at altitudes and on headings capable of a pond landing must be turned away by selectively activating only those deterrents proximate to the birds in question. The radar-activated deterrent system (RADS) must be able to distinguish between birds on geo-feasible flight paths from those that are not; for example, between low-flying local birds heading towards a tailings pond and high-flying migrating birds passing through overhead.

All this must be accomplished on a large scale to include all of the tailings ponds at a mine which can be spread across a collective area larger than a thousand square kilometers; and it must operate automatically, be user-friendly and be controllable by a single human operator from a single location for economic feasibility.

In the remainder of this paper, we present a state-of-the-art approach undertaken by Accipiter Radar and installed at Syncrude Canada Ltd that addresses this challenge for maximum bird protection.

OPERATIONAL CONCEPT

The operational concept of the RADS is illustrated in Figure 1. Waterfowl a geo-feasible flight path towards a tailings pond turn away in response to deterrents activated as they approach the sensitive area. In this case, deterrents consist of two varieties, high-power (pressure) acoustic devices (HPADs) and floating deterrent devices. HPADs are used as long-range deterrents in a first line of defence. They are spaced around the perimeter of a tailings pond as illustrated in Figure 1 and are directed outwards, away from the tailings pond. Each HPAD is typically responsible for responding to geo-feasible flight paths in a specific sector around the tailings pond. HPADs are effective at directing very loud acoustic sounds within a focused spatial sector (approximately 10 degrees in azimuth and 10 degrees in elevation) at distances of 0.5 km to 1.5 km from the device. An HPAD is illustrated in Figure 2.

If birds make it through the first line of defence and continue to approach a tailings pond, they are met by a second line of defence as they attempt to land on the tailings pound. These deterrents are necessarily mounted on floating platforms and arranged on a grid-like pattern (Figure 1). Groups of floating deterrents are logically programmed to activate together to protect a sub-region of a pond, and respond directly to birds on geo-feasible flight paths overhead or on near-approach to the tailings pond. As birds alter their course and enter other sub-regions, other deterrents in respective sub-regions will activate until birds fly clear of the pond. A typical floating deterrent platform is illustrated in Figure 2. Each floating platform houses multiple deterrent devices for maximum effectiveness.

Avian radars are used to provide 360^o 3D surveillance of each tailings pond, the air space above, and the approach to each. Their coverage and monitoring are optimized to respond to birds on geo-feasible flight paths. A radar track is automatically formed for each bird (or flock) and the 3D coordinates of its trajectory (latitude, longitude, altitude versus time), in addition to speed, heading and size information is capture by the radar system. Because more than a single radar sensor is typically needed to monitor a mine, the surveillance coverage from multiple radar sensors is seamlessly integrated to provide the desired overall coverage. Figure 1 illustrates two radar trailers covering a large tailings pond, each with two, 360^o 3D radar sensors providing complimentary coverage. 3D surveillance is essential to distinguish birds at different altitudes; i.e., to determine geo-feasibility to reduce habituation. 360^o coverage is essential to detect geo-feasible flight paths from all directions approaching the tailings pond. Figure 2 provides a close-up of a typical, dual-radar surveillance trailer.

Figure 1. Illustration of radar-activated deterrent system protecting tailings pond showing a trailer with two X-band radars, shore-mounted high-power acoustic devices, and floating deterrent modules.

Automatic System Control

A large tailings pond (say at least 10 square kilometers) may require a few radar sensors and tens of deterrent devices to provide good protection. To minimize habituation, numerous virtual alert zones are formed in association with the multiple lines of defence within the integrated coverage pattern of the radars. Each alert zone causes a subset of associated deterrents to automatically activate in response to a geo-feasible bird threat penetrating such zone. Because of the vastness of the areas being protected, the deterrents cannot be hardwired to the RADS controller economically, which is responsible for the deterrent activation logic. Rather, the deterrents are located wherever it is deemed necessary and convenient; then they are networked wirelessly. Each season, they may be deployed differently. Floating deterrents may move a little during a season with wind, and as ice forms before they are removed for the deep freeze. A GPS is installed on each deterrent to deal with movements and the deterrent's position is communicated regularly so that the RADS always knows its location.

As a result of the varied and changing landscape common to mining operations, line of sight communications cannot be assured. As a result, the RADS controller uses robust mesh networks to communicate with its deterrents, send activation commands to the deterrents, and receive status information from the deterrents.

RADS subsystems are typically self-powered through generators and solar panels. The radar trailers and HPADs are typically powered with diesel generators, while the floating deterrents are typically powered with solar and batteries. Propane cannons on the floating modules require propane to operate. Fuel/battery levels need to be automatically monitored by the RADS as environmental conditions can cause significant variations in energy consumption. System health must be monitored and remotely diagnosed and corrected to minimize the need to send out personnel for maintenance.

Figure 2. Radar-activated deterrent system subsystems: dual-radar trailer, floating deterrent, and HPAD deterrent.

A RADS bird activity information system is essential to recording all bird movements, zone alerts, deterrent activations, and bird responses. In addition, system health information must be monitored and maintained to assess system availability and support subsequent analyses.

User Requirements

With hundreds of sensors and devices to monitor and control over a large mining operation, a common operating picture (COP) is essential for RADS usability. The RADS COP integrates all system components and information in real-time and presents a single, unified display and console so that a single user can quickly gain situational awareness and can apply command and control to system elements. The COP provides the user with an integrated view of all of the birds in the air-space, all of the alert zones and their status, and all of the deterrents and their status.

TECHNICAL APPROACH

Bird Behaviour and Control Logic

The birds to be targeted are limited to those that might potentially land on the tailing ponds or their shores. This constraint means that the radar surveillance sensors must provide monitoring of the complete surface of the pond as well as the shores and surrounding area to about 2 km from the shore. Large ponds require multiple radars to provide coverage with the resolution that is necessary to determine bird behavior and selectively control the deterrents.

Two criteria are used to indicate geo-feasible birds that might be landing on a tailings pond: (1) they are flying towards the pond and (2) they are at an altitude (typically < 200 m) suitable for landing.

The birds of interest range in size from gulls and small ducks, such as teal, to flocks of waterfowl. Songbirds do not land on the tailings ponds and rarely on the ground around the pond where they might encounter bitumen. Using the size classes of birds, as measured by the radar, we filter out radar returns from songbirds (Beason et al. 2012, Nohara 2010, Nohara et al. 2011). This prevents unnecessary activation of the deterrents and reduces the rate of habituation to the deterrents.

Birds flying away from the pond are ignored because they are engaging in a desirable behaviour and activating the deterrents might be perceived as “punishment”. A second reason for not triggering on such birds is that it would increase exposure of all birds in the area to the deterrents and could increase the rate of habituation.

A multistage strategy of defence is employed. The surrounding approaches of each pond are divided up into multiple virtual zones as a first line of defence protected by HPADs. If these are penetrated, a second line of defence is met by the birds. In this case, the pond surface is divided up into a number of sub-regions or surface zones, each of which is protected by an array of floating deterrents that respond to birds in that zone.

Geo-feasible radar tracks from birds or flocks that come within about 1.5 km of the shoreline and are flying more-or-less towards a tailings pond cause an automatic zone alert. The long-range deterrents associated with that zone respond and broadcast a very loud programmed sound with user-defined timing. If the birds continue towards the pond, the sound is broadcast a second time (say about 60 seconds later) until the birds depart the zone.

If the birds continue and cross the shoreline of a pond entering a sub-region or zone on the pond, a group of short-range floating deterrents in that zone are automatically triggered. The short-range deterrents are triggered at intervals of say 60 seconds for as long as the birds are within the zone over the pond. As soon as the birds exit a given zone, those respective deterrents cease.

Architecture

The RADS network architecture is illustrated in Figure 3 and further described in Nohara 2010. The collection of Accipiter radar nodes used to provide surveillance coverage of the tailings ponds communicates over an intranet/Internet to the Bird Activity Information System which is the heart of the RADS and tracks all geo-feasible birds in the airspace. Land-based acoustic devices (i.e., HPADs) and Floating Deterrent Units are distributed around and on the tailings ponds and controlled over a mesh radio/communications network by a Deterrent Activation Processor. The mesh network supports non-line-of-sight communication among the deterrents. The Deterrent Activation Processor interacts directly with the Bird Activity Information System in real-time, and implements the zone alerts and deterrent activations.

Radar Sensors

X-band avian radar systems with dish antennas are selected for use because they have proven to provide the best performance in terms of high-resolution, 360^{o}, 3D capabilities (Beason et al. 2012, Nohara et al. 2011). Specialized radar capabilities such as multipath and sidelobe suppression to mitigate extraneous returns from large machinery, and adaptive clutter mapping to mitigate the changing clutter environment, are especially needed for mining environments. X-band is ideal for providing excellent, local weather data, which has proven to be essential during spring and fall migration periods to predict when birds may be forced down by unfavorable flying conditions.

Deterrents

Two categories of deterrents are presently used; one for long range and one for short range. The long-range deterrents consist of high-power acoustic devices (HPAD) to replace traditional shore-mounted propane cannons. Propane cannons have been used as bird deterrents because they produce a sound that (to humans) resembles the blast of a shotgun. Thus, the sound from the cannon is reinforced by hunters shooting at the birds elsewhere during migration.

The HPADs (HyperSpike HS-24 is used at Syncrude) are spaced along the perimeter of the ponds and directed away from the pond towards approaching birds. The projected sounds are typically transmitted for short periods (e.g., 10 seconds) and repeated at longer (e.g., 1-minute) intervals if geo-feasible birds are still present. To further reduce the rate of habituation, the projected sounds can be changed daily.

The short-range floating deterrent platforms provide multi-sensory stimulation produced by a combination of visual and auditory deterrents that are activated simultaneously. The visual stimuli include a super-normal sized falcon effigy with flapping wings when activated and a strobe light. The strobe light functions to draw the bird’s attention to the effigy. Because birds do not estimate distance based on binocular vision but size-constancy (Martin 2009), the large falcon effigy appears to be much closer to the birds than it actually is. The auditory stimuli include the call of a predatory bird, to go with the falcon effigy, and on-board propane cannons.

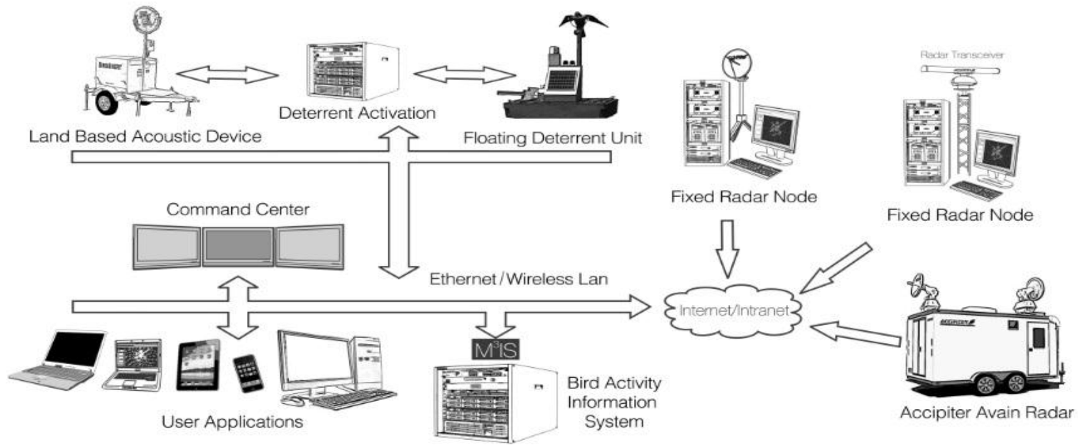

Figure 3. Radar-activated deterrent system network architecture.

The short-range deterrents are arranged together as a set and, for tailings ponds, mounted on a floating platform. Several platforms are distributed within a deterrent zone or sub-region and activated simultaneously when birds intrude into that zone. These deterrents are activated for a short interval (on the order of 10 seconds) when activated. The short-range deterrents have a longer time-out (on order of 1-minute) similar to the long-range deterrents. This interval gives the birds time to clear the area without repeated stimuli, which might increase the rate of habituation.

OPERATIONAL EXAMPLES

In this section, we provide examples to illustrate the concepts and designs described earlier. We begin with a view of the Common Operating Picture (COP) at Syncrude, shown in Figure 4. The left-hand pane of the COP is used for controlling the various views availabe and the current view is shown in the image window on the right-hand pane. Figure 4 shows a zoomed-out view with few outlined tailings ponds visible, along with birds currently being tracked, alerted zones (one is lit up), and symbols representing each deployed deterrent.

The image window is developed in GoogleEarth™ so the user can zoom, pan and tilt to provide 3D, real-time views of the current activity. Geo-feasible bird tracks will cause appropriate zones to activate (shaded) so the user has instant awareness; the triggered deterrent devices (i.e., their symbols) also light up. Right-clicking on a deterrent symbol raises a dialog which reports status attributes (such as state, fuel levels, system health, location) and provides control functions (e.g., reset, test).

Each tailings pond is organized as a logical subsystem that can be separately controlled. Redundancies are built into the system to accommodate its size and complexity.

Broad-scale migration is visible in Figure 5 as birds fly over multiple tailings ponds. The radar tracks from all ponds are integrated into the single COP display, providing complete situational awareness to the operator in a single view.

In Figure 6, geo-feasible tracks from a flock of birds are seen turning away from a tailings pond in response to five floating deterrents modules (numbered 28-32) activating on an inner alert zone. The inner alert zone containing the five deterrents is automatically activated in response to the geo-feasible birds (Figure 6).

FUTURE WORK

With the large-scale RADS only recently deployed, there is considerable opportunity for continued improvement of system effectiveness, based on performance metrics derived from the extensive observations collected by the bird activity information system.

Figure 4. Common operating picture for real-time viewing and command and control of system. The symbols on the ponds represent floating deterrent modules.

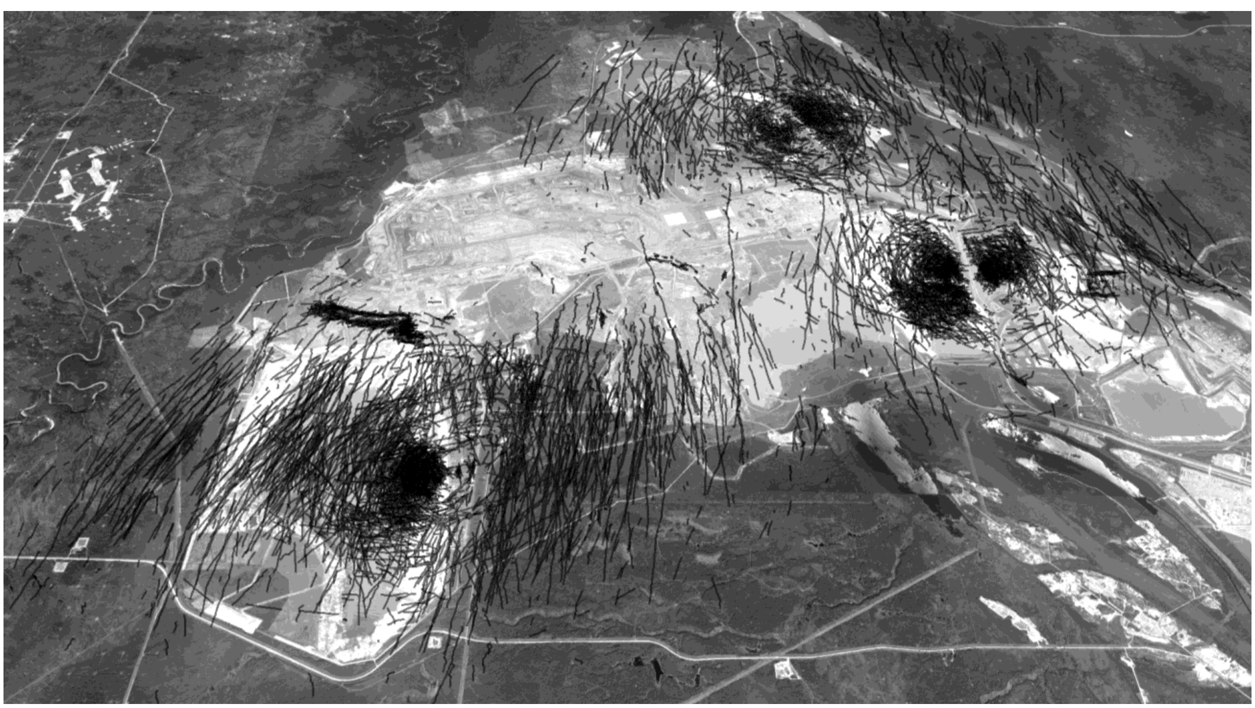

Figure 5. Broad-scale, night-time migration across multiple tailings ponds. The black lines represent the tracks of individual migratory birds.

An extremely important need is to determine the efficacies of various auditory stimuli. The HPAD devices are highly directional and efficient at transmitting sounds long distances. The most common sound used is the explosion to resemble a shotgun blast. Other sounds, such as avian alarm and distress calls, have been used with varying success. While it is unlikely that a single sound will be effective against all species of waterfowl, researchers can evaluate a variety of sounds and determine which are most effective against specific species or groups of birds. The radar-activated deterrent system is the ideal device to collect those data. It records when the deterrents are activated and the behavioral responses of the birds to the stimuli, as well as the time-lag from stimulus to response. Because the system is automated, it can collect data continuously without human oversight or bias.

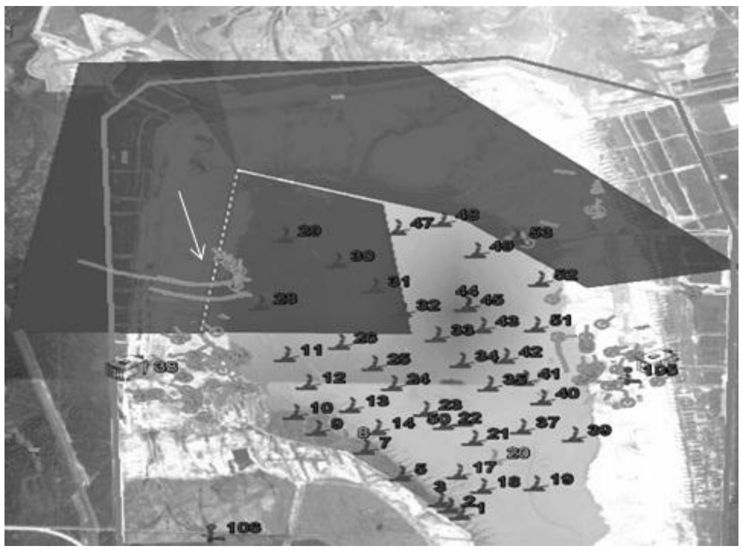

Figure 6. Birds (arrow) turning away in response to floating deterrent modules 28-32 after entering sub-region of tailings pond. The numbered symbols on the pond represent individual deterrent modules.

Developing additional deterrents is also important. Although the deterrents should not be activated by migrating birds passing overhead, they need to be activated by migrating birds that approach a pond to land. Migrating waterfowl can stop their night's migration at any time during the night or the following morning. Although the auditory deterrents should be equally effective at night, the visual deterrents will not be. Red and green lasers have been reported to disperse some waterfowl from water surfaces (Blackwell et al. 2002, Werner and Clark 2006). In a recent study, Syncrude and Accipiter scientists verified this response to lasers (Beason and Polak, 2012). As a result, radar-activated laser deterrents may be added to the deterrent system in the future to provide additional nighttime protection.

Since the deterrents can be deployed and grouped arbitrarily, and since any number and design of virtual electronic zones can be created by the RADS to trigger deterrents in response to geo-feasible bird flight paths, additional research should lead to further optimization of the RADS protection strategies based on the collected observations.

Furthermore, the wide-area bird behavior observations collected by the RADS has the potential to supply researchers including biologists, ornithologists, conservationists, public health officials and others with valuable data sets and understanding.

The RADS architecture described herein fully supports real-time integration and sharing of data from adjacent mining operations providing early warnings to neighbours of migration events. In the fall, operators to the north will be able to provide early warning to operators to the south; in the spring, the situation reverses.

Finally, with the deployment of high-resolution X-band avian radars that can track weather well, a wide-area, high-fidelity, parallel weather channel can be developed and added to the RADS to provide real-time warning of significant weather events, and to provide local weather pattern forecasting. High-quality, real-time, local weather information will not only improve bird protection, but can protect operations and humans as well.

CONCLUSIONS

The highest standard in bird protection has been presented for mining applications involving the need to keep waterfowl away from tailings ponds. A state-of-the-art approach undertaken by Accipiter Radar and rolled out at Syncrude Canada Ltd addresses this need by employing a large scale, integrated, radar-activated deterrent system that only responds to birds that are at risk in order to reduce habituation and maintain effectiveness. Notwithstanding the harsh (wind, blowing sand, -40^{o} C temperatures) and ever changing environment presented by the Alberta oil sands, the deployed systems have proven capable in operating reliably and providing the necessary data, situational awareness and control to be operationally useful.

The concept of operation, technical approach, implemented systems, and examples we present are applicable to other situations such as off-shore platforms, wind-farms, and airports.

ACKNOWLEDGEMENTS

The authors wish to express their thanks and gratitude to Syncrude Canada Ltd for their sincere commitment to protecting birds from exposure to tailings ponds. The state-of-the-art, radar-activated deterrent system described here would not have resulted had it not been for Syncrude's determination and vision to challenge system developers to provide state-of-the-art bird protection today, while factoring in the flexibility for improvements tomorrow. The authors are also indebted to Alarm Control Systems, Salt Lake City, Utah, whose personnel are responsible for the state-of-the-art deterrents that are integrated into the RADS. Finally, our thanks goes out to the entire team at Accipiter Radar who continue to work tirelessly and innovatively with our partners in developing and improving RADSs for the protection of birds in energy-generation applications.

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RESPONSES OF BENTHIC MICROORGANISMS (THECAMOEBIANS) TO OIL SANDS PROCESS-AFFECTED MATERIALS; PROVIDING ENDPOINTS FOR GAUGING AQUATIC RECLAMATION SUCCESS

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ABSTRACT

Constructed wetlands and end-pit lakes will play an important role in reclamation options for fluid tailings (OSPW/M) at surface oil sands operations. Through time and with natural bioremediation viable aquatic habitats will develop, but currently few tools are available to determine the rates of remediation in produced ecosystems. A micropaleoecological environmental proxy (thecamoebians) has been demonstrated to provide a time-averaged indicator of ecosystem health. Thecamoebian communities in sediments from both impacted and non-impacted wetlands and lakes in the vicinity of oil sands operation have been compared. An index of response to stress has been compiled with the goal of using it as a predictor of the path of remediation that will produce sustainable ecosystems. This information also provides an endpoint for remediation efforts.

Thecamoebian assemblages in cores and surface samples from 63 natural lakes across the region were used to establish natural ecological ranges and remediation targets. These were compared to those present in wetland sediments impacted by oil sands materials (OSPW/M). The process-affected sites had lower thecamoebian diversity and were dominated by centropyxid taxa, whereas more abundant and diverse assemblages dominated by difflugiid taxa characterized less-impacted sites. Moreover, assemblages responded quickly to changes in OSPW/M input and to various reclamation strategies, such as nutrient input. Preliminary results suggest that thecamoebians represent proxies for gauging ecosystem health, monitoring aquatic reclamation progression and developing target endpoints.

INTRODUCTION

The Alberta oil sands (AOS) are one of Canada's most economically important natural resources. Assessing and remediating potential detrimental environmental impacts to aquatic habitats resulting from sands developments in the Wood Buffalo region of northern Alberta requires innovative approaches that can follow ecological, temporal and spatial distribution of possible impacts.

During oil sands processing, large volumes of water are used, with most of it being recycled from tailings retention ponds. Over time, concentrations of dissolved constituents, mainly salts and dissolved organics, associated with oil sands operations become elevated in the OSPW/M. The waters released from tailings ("free" water) in surface zones or captured within the pore spaces of tailings (sands, fines) deposits have unique character and properties relative to natural non-OSPM impacted waters. In general, freshly produced OSPW will stress biota through elevated ionic content and presence of organic-acid constituents such as low-molecular-weight naphthenic acids, but the toxic character of the OSPW has been demonstrated to dissipate over time (Harris, 2007; Neville et al., 2011).

Ecological Indicators

Thecamoebians (testate amoebae, arcellacea) are protists (unicellular microorganisms) that comprise an important component within the microbial trophic level of the benthic community in lakes and wetlands (Patterson and Kumar, 2000; Beyens and Meisterfeld, 2001).

Species and strains are characterized by a simple sac-like decay-resistant organic test of pseudo-chitinous material that is variably agglutinated in different species (Patterson and Kumar, 2000; Scott et al., 2001). Thecamoebians are useful in environmental research as they are characterized by rapid generation times, and sensitivity to environmental conditions at the sediment/water interface and epibenthic zone (Neville et al., 2010a). Their abundant fossilized remains preserve a record of contaminate responses and changing environmental conditions over time (McCarthy et al., 1995; Boudreau et al., 2005). Unlike most microfossil groups, thecamoebians are resistant to dissolution in lower pH environments (Swindles et al., 2007).

Variation in thecamoebian community assemblages have been successfully used in investigations of paleoclimate (Boudreau et al., 2005; McCarthy et alk., 1995), sea-level change (Scott et al., 2001), and anthropogenic impact, including that of sulphide mining in acid-sensitive lakes in Ontario (Patterson et al., 1996; Reinhardt et al., 1998; Kumar and Patterson, 2000; Patterson and Kumar, 2000) and in Finland (Kauppila et al., 2006). These latter studies led us to investigate the sensitivity of thecamoebians to the by-products of oil sands production.

Wetland habitats, both constructed and opportunistic, will be important components of the reclaimed oil sands impacted landscapes, and the rate of progression from OSPM-stressed to more natural systems will be an important factor for gauging reclamation success. Simple and effective methods to monitor the early stages of remediation of these wetlands need to be developed to demonstrate a trajectory towards natural processes. This study investigates the use of benthic microbiota in assessing the effectiveness of the remediation process.

To further assess the applicability of thecamoebians as biomonitors of potential oil sands industrial impact, the project aims to determine reliable methods for discriminating between anthropogenic from natural sources and ecological impacts on natural areas surrounding oil sands operations. This could assist in evaluating whether OSPW/M emanating from oil sands operations is negatively impacting local aquatic habitats and whether reclaimed aquatic systems will perform as viable components in final lease-closure landscape. Thecamoebian populations from lakes sampled as part of this program were compared to populations found at oil sands sites where there was varying levels of stress from OSPW/M. A range in degree and timing of impacts provided various test sites that were analogous to what would be expected in aquatic reclamation options including an indication of rates of progression to target endpoints.

METHODS

Test Pond Study Sites

Thirteen surface sediment samples were collected from the Constructed Wetland Test Facility (CWTF; located at Suncor Energy Inc.; Figure 1; Table 1) in 2007; eight of those sites were re-sampled in 2008. Sediment samples for both years of study were collected using an Ekman grab sampler. The test site was comprised of four areas (Suncor CT Demonstration Study Site, Sustainable Lake South, Sustainable Lake North, and Crane Lake) that differed in construction and implementation, and each contained a series of wetlands (Figure 1; Golder Associates Ltd., 2006). Between the sample collection of Set One (2007) and Set Two (2008), modifications were made to various CWFT sites causing increased or decreased OSPW inflow and subsequent changes in their chemistry.

Natural Study Sites

Surface sediments were collected from 8 lakes in August 2010 and 54 lakes in August 2011 (Figure 2), sites were chosen to create a distal and proximal radius of natural lakes around the oil sands operation. The 2010 sample set was collected using a Glew gravity corer (Glew et al., 2001). The 2011 sample set was collected using an Ekman grab sampler.

Microfossil Analysis

Prior to thecamoebian analysis, 2 cc of sediment were passed through a 250 μm sieve to remove coarse organic debris and then a 37 μm sieve to remove fine organic and mineral detritus. The 37-250 μm aliquots were subdivided for quantitative analysis using a wet splitter. The wet aliquots were subsequently examined under an Olympus SZH10 dissecting binocular microscope (40–80X magnification) until a statistically significant number of specimens were quantified (Patterson and Fishbein, 1989). Identification of

Figure 1. Satellite photographs of the Suncor Constructed Wetlands Test Facility (CWTF) in the Athabasca Oil Sands (Neville et al., 2011).

thecamoebians followed standard reference keys (e.g. Medioli and Scott, 1983; Kumar and Dalby, 1998). Scanning electron micrograph images of common species and strains were obtained using a Tescan Vega-II XMU VP scanning electron microscope at the Carleton University SEM facility (Figure 3).

RESULTS

This study investigated 21 tailings-influenced lakes and wetlands and 53 natural lakes both up and downstream of the Athabasca Oil Sands operation. The six samples in Set One from wetlands with relatively low OSPM impact (samples 8, 9, 16, 17, 20, and 21; average naphthenic acid (NA) concentration 8 mg/L, electrical conductivity 1400 μS/cm) contained a relatively abundant (N=30/cc) and diverse (mean SDI=2.0) thecamoebian fauna dominated by difflugiid species. The seven samples in Set One from the high OSPW character wetlands (samples 2, 5, 11, 13, 14, 15 and 18; average naphthenic acid concentration 47 mg/L, conductivity 2300 μS/cm) (Figures 4 & 5) yielded a generally less-diverse (mean SDI=1.5) and typically less-abundant fauna (N=24.5/cc). These sites were typically characterized by centropyxid thecamoebians, while difflugiids are rare in these samples. The eight sites resampled in June 2008 (Set Two) (Table 1) had average conductivity and naphthenic acid concentrations of 1656 μS/cm and 24 mg/L, lower than the average values of 1850 μS/cm and 27 mg/L, respectively, reported for the 2007 (Set One) samples. Typically centorpyxid populations are ambiguous and can survive in a wide range of environments while difflugiid populations are sensitive to changes in environmental conditions, and populations will decrease as conditions deteriorate.

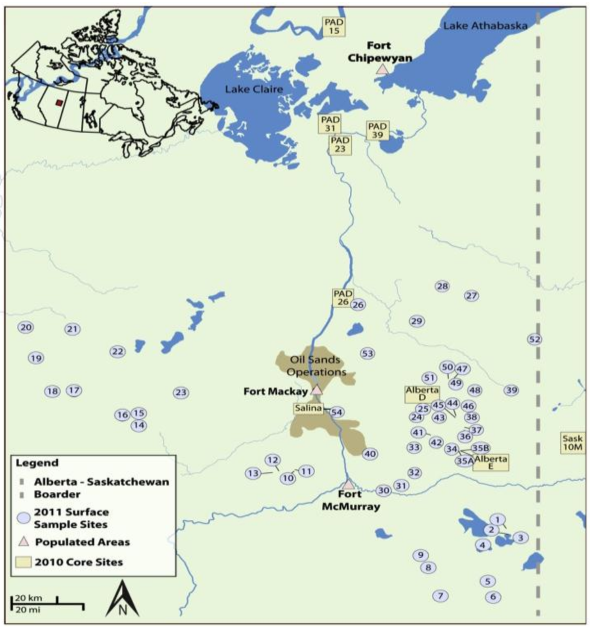

Figure 2. Map depicting the location of the natural lakes sampled surrounding the Athabasca Oil Sands operation.

An inverse relationship (r^2=0.60) between the relative abundance of difflugiid thecamoebians and conductivity was also noted (Neville et al., 2010). In addition, conductivity shows a strong relationship (r^2=0.71) with the relative abundance of centropyxids (Fig. 6).

Thecamoebian assemblages from the two largest test sites, Sustainable Lake North and Sustainable Lake South, differed markedly from each other, even though the chemical characteristics of the overlying water differed only slightly. These sites were created at the same time, as a layer of fluid fine tails (FFT) was capped by a layer of about 2.5m of fresh OSPW from an active tailings basin. After filling, they were isolated from fresh OSPW recharge. Also, these ponds were hydrologically isolated from surface water recharge, with water balances controlled by precipitation/evaporation from the basin itself and the release of OSPW from the FFT zone (Fig. 5). High concentrations of NAs and high conductivity values at both sites remained fairly constant between Set One and Set Two, but Sustainable Lake South had the more diverse and difflugiid-rich assemblage both years. In addition, a much greater change in the thecamoebian assemblage was observed at Sustainable Lake North between 2007 and 2008, with an increase in the relative abundance of difflugiid taxa from 3.1% to 23.5% of the total assemblage, and a change in SDI from 0.83 to 1.76.

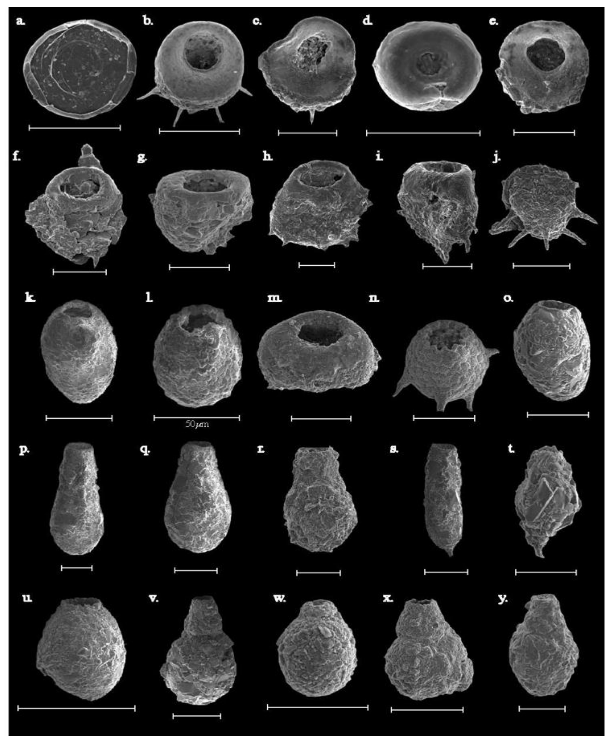

Figure 3. Examples of common thecamoebians found in Alberta.

Cluster analysis allows for grouping of sites that contain similar populations, while Principal Component Analysis (PCA) indicates the relationship between sites and measured environmental variables. Both cluster (Figure 7) and PCA (Figure 8) identified a gradual change in the thecamoebian communities from tailings influenced sites to hose found at natural lakes (non OSPW/M). The cluster analysis produces four main faunal groupings. The majority of tailings sites cluster in Groups 3 & 4 with the exception of site 14 and 14-2, which cluster in Group 2 with the

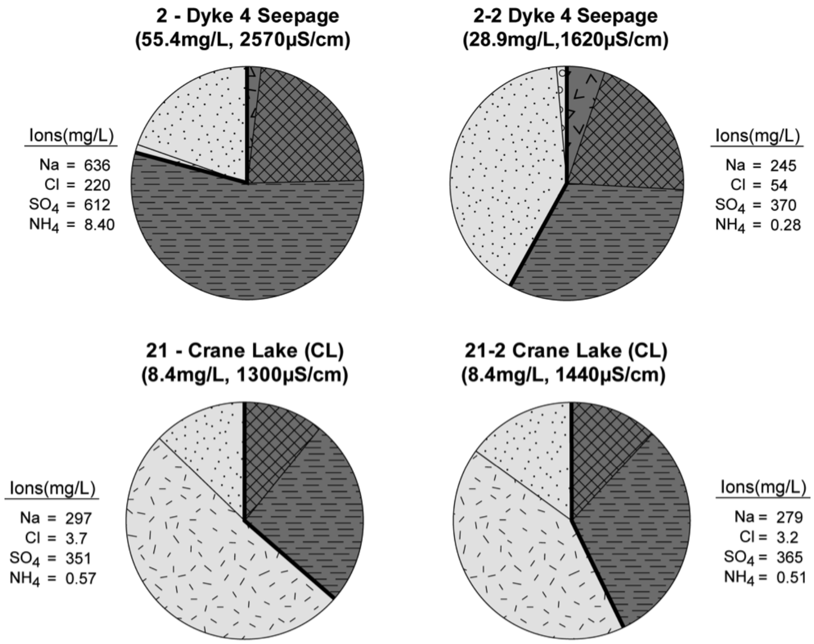

Figure 4. Pie diagrams showing the change in assemblages at sites 2 and 21 between 2007 (Set One) and 2008 (Set Two). There is virtually no change in water chemistry at site 21 and nearly identical thecamoebian assemblages are observed both years.

natural lakes. The PCA shows two main groupings; one containing the tailings influenced sites clustering on the left of the diagram, and the other containing the natural lakes clustering on the right.

DISCUSSION

Samples from the constructed wetland test facility (CWTF) at Suncor Energy Inc. demonstrated the sensitivity of thecamoebians to the by-products of mining and processing oil sands. The relative abundance of centropyxid thecamoebians showed a strong correlation with NA concentrations (r^2=0.74) and electrical conductivity (r^2=0.71) in the combined 2007 and 2008 sample sets, whereas an inverse correlation exists with difflugiid thecamoebians (Fig. 6). The resampling of eight sites in 2008 further confirmed the sensitivity of thecamoebians to by-products of oil sands mining and extraction, with the additional data slightly raising the r^2 values. The regression line created by plotting the relative abundance of difflugiid or centropyxid thecamoebians against NA concentration or conductivity (Figure 6) may be used to extrapolate how quickly the health of an aquatic ecosystem undergoing remediation is changing. Future work will establish baselines to estimate endpoints.

The constructed wetlands showing high impacts from OSPW/M contained lower diversity thecamoebian assemblages that were dominated by centropyxid-type (Figures 4 & 5). Less impacted sites contained more diverse assemblages dominated by difflugiid-type thecamoebians (Figures 4 & 5). Thecamoebian assemblages responded quickly to a deliberate reduction in the rate of OSPW/M input, with an increase in species diversity and the relative abundance of difflugiid thecamoebians. At sites with little change in water quality thecamoebian assemblages were comparable to the previous year (Figures 4 & 5).

Test sites with low OSPW character (i.e., low concentrations of salt [conductivity] and dissolved organics [mainly naphthenic acids]) contained assemblages similar to those found in natural lakes in the Boreal Forest region of Alberta (Neville et al, 2010b). These sites cluster with Group 3 (Figure 7) indicating that thecamoebian populations are similar to those found in natural non-OSPW/M impacted lakes surrounding the oil sands developments. Less impacted aquatic test sites from the active Leases result from low inputs of OSPW or natural bioremediation occurring over time. When results from these locations are plotted, they tended to cluster within the natural lakes (Group 3, Figure 7), even though their water chemistry was different than that seen in natural lakes. Thecamoebian populations found in tailings influenced lakes are strongly influenced by ion content (Figure 8), while populations in natural non-OSPW/M lakes appeared to be more influenced by temperature. In examination of the water quality properties and thecamoebian communities, it appears that the ecology of the less impacted tailings influenced lakes are developing towards and are becoming similar to the natural lakes found in northern Alberta. This encouraging evolution of a viable microbial trophic level in oil sands reclamation sites provides confidence that given sufficient time these systems should become viable aquatic habitats capable of supporting higher trophic level organisms.

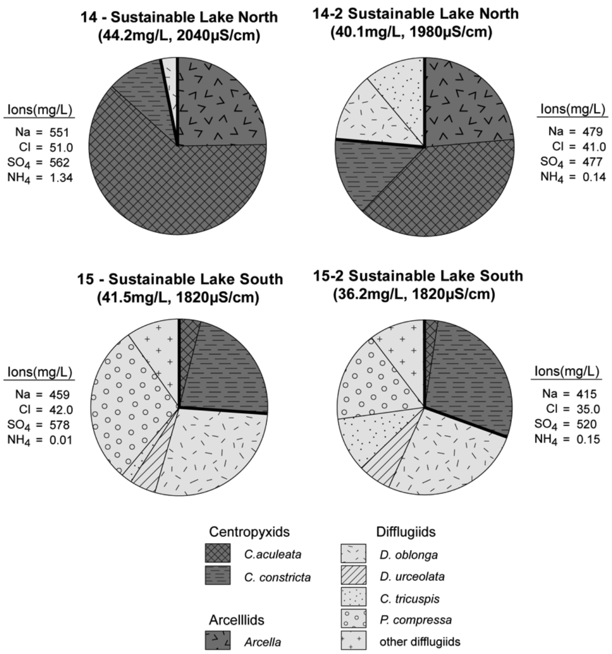

Figure 5. Diagrams showing the change in assemblages at sites 14 and 15 between 2007 (Set One) and 2008 (Set Two). Pies show assemblage differences between adjacent sites despite similar water quality.

In the comparison of the Sustainable Lake North (site 14) and Sustainable Lake South (site 15) sites from Suncor's constructed wetland facility (Figure 5), an illustration of the potential power and applicability of using thecamoebian assemblages as a biomonitoring tool for assessing reclamation management options has been presented. These adjacent small lakes of similar dimension were constructed at the same time and underwent the same seasonal changes.

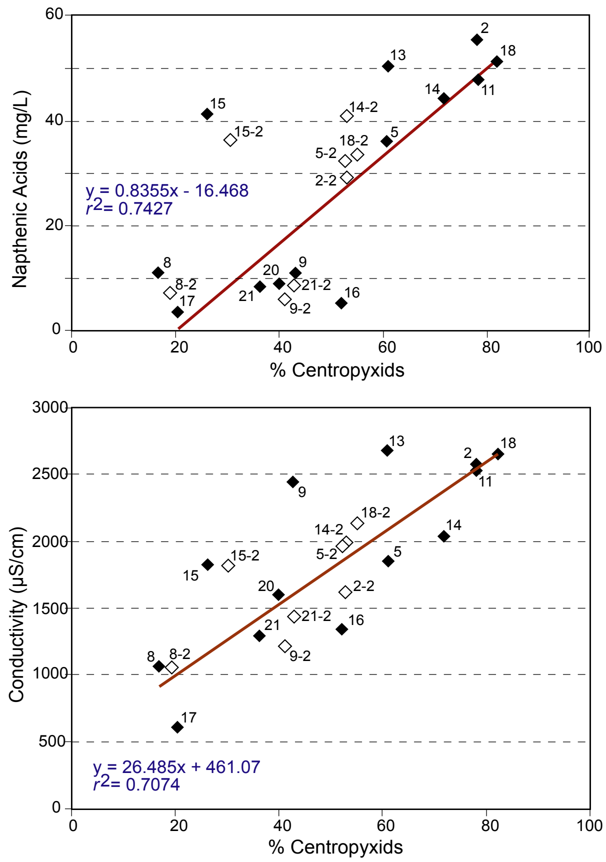

Figure 6. Graphs showing the strong correlations between the relative abundance of centropyxid thecamoebians and two of the major by-products of oil sands extraction—naphthenic acid concentrations (r^2=0.74) and electrical conductivity (r^2=0.71). The 2008 values (hollow diamonds) have shifted down the line of best fit compared to the 2007 values (solid diamonds) reflecting improvement in water quality. The enhanced water and consequent decline in the relative abundance of centropyxid thecamoebians is most apparent at sites 2 (Dyke 4 Seepage) and 18 (Jan's Pond), where they reflect the deliberately reduced inflow of oil sands process-affected water (OSPW) between 2007 and 2008.

As they evolved with time, the surface waters showed similar characteristics of OSPW. In both 2007 and 2008 studies, thecamoebian assemblages in Site 15 (15 and 15-2) were more

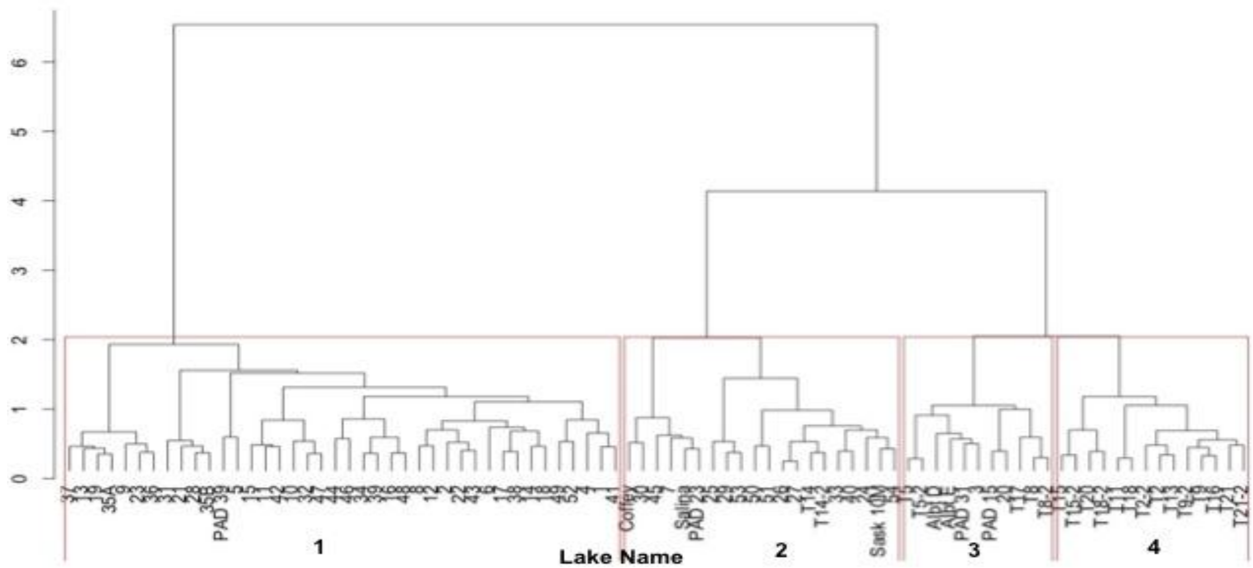

Figure 7. Cluster analysis using Wards method showing 4 main assemblage groupings. Samples collected from tailings impacted sites are prefixed with T and mainly a cluster in groups 3 & 4.

diverse and difflugiid-rich than would have been expected for a lake with such high OSPW/M influence. Sites 14 and 15 have been managed differently since their construction in 1992, with intensive nutrient-loading practiced at Sustainable Lake South but not at Sustainable Lake North. The properties of the sediment in each site also suggest that Sustainable Lake South showed higher organic content from detrital build-up on the underlying OSPM, which resulted from its higher productivity. Our analysis suggests that nutrient loading may speed up the remediation process by boosting productivity and, consequently, detrital deposition rates, even if the site remains highly impacted by OSPM. Cluster analysis (Figure 7) places the thecamoebian assemblage found in Sustainable Lake North within Group 2, which is mainly composed of natural lakes surrounding the operation. This suggests that the benthic ecology of this lake is becoming similar to that found in natural Boreal lakes and, in terms of ecology, the remediation endpoint is on the trajectory toward being met.

CONCLUSIONS

Thecamoebians are sensitive proxies of environmental quality, as demonstrated at the Suncor Energy Inc. Constructed Wetlands Test Facility (CWTF) near Fort McMurray, Alberta. They provide a relatively easy and inexpensive method for assessing remediation practices and efficacy in oil sands aquatic-reclamation systems.

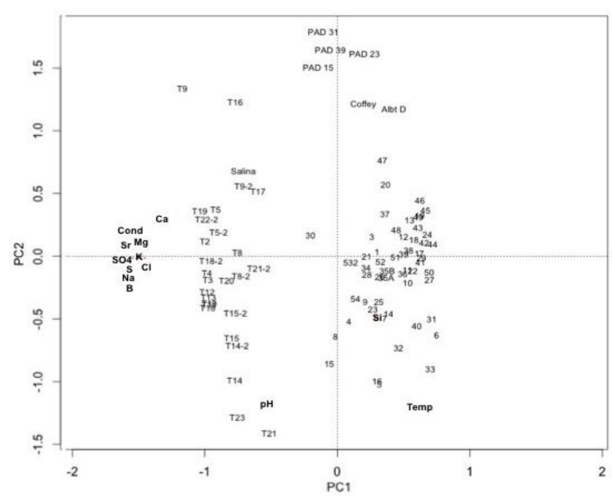

Figure 8. Ordination diagram of sites and environmental parameters created using Principal Component Analysis (PCA). Samples collected from tailings impacted sites are prefixed with T and cluster on the left half of the diagram while natural sites cluster on the right and are primarily influenced by Temperature and Si.

When thecamoebian populations in a reclaimed water body are compared to those in natural environments of the region, a goal for aquatic reclamation habitats can be defined. In OSPW/M impacted environments, the changes in numbers and composition of the thecamoebian community can be applied as an indication of ecosystem health. Over time the trajectory to natural remediation endpoints may be established.

The thecamoebian community structure at CWTF sites responded to changes in water chemistry produced by a deliberate reduction of OSPW flux. The high degree of similarity between the 2007 and 2008 sample sets where there was no marked change in water quality suggests that the use of thecamoebians as a remediation metric is both sensitive and reproducible. Chemically, the OSPW/M impacted reclamation systems are likely to be quite different from the natural lakes but the thecamoebian populations in these lakes and wetlands have still begun to establish and progress toward natural ecosystems. Currently, in some of the less OSPW/M impacted reclamation test ponds and wetlands, thecamoebian populations already resemble natural benthic Boreal communities.

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Table 1. Identification of Samples Collected at Each Site From the Suncor Constructed Wetlands Test Facility (CWTF) in 2007 (Set One) and 2008 (Set Two).

Year	Dyke 4 Seepage	Dyke 4 Reservoir	Control Reservoir	V-notch Weir	Gooseneck	Weir B	Sustainable Lake North
2007	2	5	8	9	11	13	14
2008	2-2	5-2	8-2	9-2			14-2
Year	**Sustainable Lake South**	**Sodic Wetland**	**Muskeg Wetland**	**Jan's Pond**	**Pond A**	**Crane Lake**	
2007	15	16	17	18	20	21	
2008	15-2			18-2		21-2	

Session 10

Desiccation Dewatering

FORMULATION AND SOLUTION OF A NUMERICAL MODEL FOR OIL SANDS TAILINGS FROM SLURRY TO DESICCATION

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ABSTRACT

The paper describes a numerical model that simulates physical processes that Oil Sands thickened tailings undergo from the stage of being deposited as slurry; followed by self-weight consolidation and subsequent air drying (thermal desiccation). The Oil Sands thickened tailings initially have the properties of a viscous fluid upon leaving the spigot. The Oil Sands thickened tailings then take on the properties of a soil under the process of self-weight consolidation. Self-weight consolidation and subsequent desiccation (or drying) can take place simultaneously. The consolidation and drying processes involve large volume changes. The numerical model captures all stages associated with the deposition and drying of Oil Sands thickened tailings. The soil properties take the form of nonlinear mathematical functions that account for large volume changes associated with consolidation (i.e., large-strain consolidation) and the drying process. Comparisons are made between the numerical modeling results and actual evaporation measurements on Oil Sands thickened tailings in a laboratory drying column test. Verification is illustrated in terms of total volume change and water content changes with respect to depth below the drying surface in the column of Oil Sands thickened tailings.

INTRODUCTION

A study was undertaken by Golder Associates for Total E&P Canada on the dewatering of thickened tailings (TT). The need to understand and quantify tailings dewatering mechanisms is based on the desire to move towards dry tailings management practices in response to Directive 074 introduced in 2009 by the Energy Resources Conservation Board.

The dewatering behaviour of the proposed thickened tailings was characterized through the measurement of saturated-unsaturated material properties, laboratory scale dewatering column tests, small scale field drying box experiments and numerical simulations. The scope of the laboratory testing included measurements of grain size, one-dimensional consolidation, saturated hydraulic conductivity, shrinkage curves, soil-water characteristic curves, and an estimation of unsaturated hydraulic conductivity.

A set of nine columns tests were performed to assess the dewatering and consolidation behaviour of the 2010 thickened tailings. The column tests were performed in the Golder Burnaby laboratory. The average temperature in the laboratory was 18.7^{o}C and the average relative humidity was 33.5%. The measured pan evaporation in the laboratory was 1.6 mm/day.

The main objective of the column tests was to assess the relative roles of evaporation and under drainage in the dewatering process. The columns considered a number of scenarios including evaporation only, under drainage only, and evaporation plus under drainage for single lift thicknesses of 20 cm, 40 cm, and 60 cm. (One column considered a two lift deposition sequence). The mass of the columns, settlement of the tailings, under drainage and the laboratory atmospheric conditions were measured at regular intervals during the experiment. The measured data provided the ability to quantify: i.) evaporation, ii.) under drainage, iii.) volume change, and iv.) atmospheric parameters for the computation of evaporation. The final gravimetric water content profiles were measured as the columns were dismantled. The initial and final water content profiles were compared to show the redistribution of water in response to tailings dewatering.

A numerical model was developed for the purpose of simulating the tailings dewatering process. Key considerations included software selection, interpretation of representative soil properties, and the evaluation of climatic conditions to be applied as the boundary condition at the tailings surface. A software development program was initiated using

SVFlux and the enhancements resulted in software capable of analyzing: 1) saturated/ unsaturated seepage, 2) large-strain consolidation, and 3) the ability to apply an atmospheric boundary condition at the material surface. The final result was a software solution that included the features necessary for the simulation of tailings dewatering through evaporation and under-drainage for multi-lift deposition sequences. Numerical model verification involved the back-analysis of the measured response in the columns and drying boxes.

CLASSIFICATION PROPERTIES

The tailings are 0.8 SFR (Sand to Fines ratio) and are referred to as the 2010 tailings sample. The grain-size distrubtion curve shows 55% passing the 44 μm size and the material classifies as silty sand, CL.

The specific gravity of the 0.8 SFR was 2.61. The plastic limit was 15% and the liquid limit was 46%. The clay size fraction was composed primarily of kaolinite and illite with lesser amounts of mixed layer clay minerals.

PHYSICAL REQUIREMENTS OF THE NUMERICAL MODEL

The quality of the back-analysis was based on how well the simulated evaporation, under drainage, settlement, and final gravimetric water content profiles matched actual measurements. The measured water content profiles produced signatures depending on the dominant dewatering process along the profile of the column. The simulations were able to capture crust formation at the surface of evaporation columns, self-weight consolidation for drainage only columns, and the transition between crust formation and self-weight consolidation for evaporation plus drainage columns. The ability to capture the different water content signatures depended largely on the selection of the volume change function and permeability function for the tailings.

The permeability of the tailings is known to decrease with decreasing void ratio. The permeability function must represent: 1) the reduction in under drainage as a result of decreasing void ratio at the interface between tailings and sand drains, and 2) the reduction in evaporation as a result of crust formation at the tailings surface.

VOLUME CHANGE OF OIL SANDS TAILINGS

The volume change characteristics of the tailings were characterized using a one-dimensional consolidation test to measure the change in void ratio in terms of net normal effective stress. The SWCC test was used to measure the volume change in response to changes in matric suction. The two tests represent different loading conditions as shown in Figure 1. The one-dimensional odeometer test measures the change in void ratio as a function of effective stress while the soil-water characteristic curve test measures the change in void ratio as a function of matric suction. The SWCC test involves an isotropic loading condition where a change in the stress states (i.e., matric suction) in the x-, y-, and z-direction are equal. An increase in matric suction results in an all-round shrinkage of the soil sample. The results of the SWCC test are considered to be a closer representation of the loading condition that develops near the surface of tailings during drying.

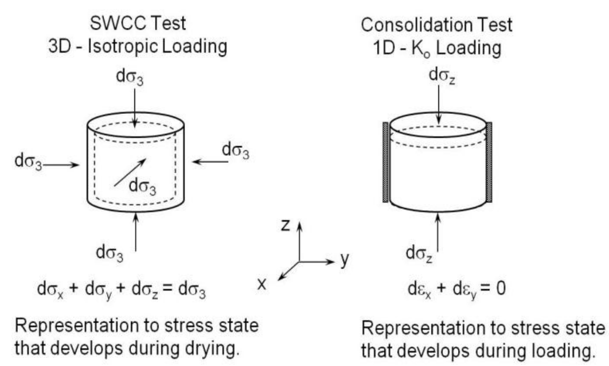

Figure 1. Comparison of 1-D and isotropic 3D loading conditions.

A comparison between the one-dimensional consolidation and SWCC test results for 2010 tailings is provided in Figure. 2. The void ratios for the one-dimensional consolidation test are plotted against normal effective stress. The void ratios measured in the SWCC test are plotted against matric suction. The data from each test has been fitted using a Weibull function. The area between the two curves Figure. 2 represents a transition zone between the two loading conditions.

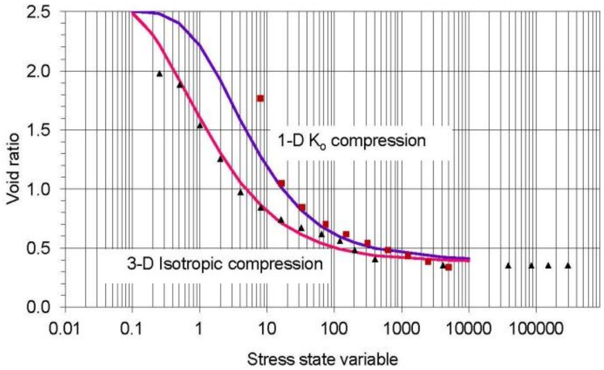

Figure 2. Oedometer (1-D) and SWCC (3-D) laboratory test results for Oil Sands tailings.

Figure 3 shows the four volume change functions used for the back-analysis of the laboratory columns. One function (No. 1 purple) represents the best-fit to the data from the one-dimensional consolidation test. A second function (No. 2 red) represents the best-fit to the data from the SWCC test. A third function (No. 3 blue), represents a situation where the tailings behave as though subject to a 1-D - K_o loading condition in the low stress range but transitions to a 3-D Isotropic loading condition in the higher stress range. The fourth function (No. 4 green), represents a situation where the tailings undergo 3-D isotropic loading condition in the higher stress range. The degree of saturation SWCC showed that the tailings had an air-entry value of approximately 300 kPa corresponding to a water content below 20%. Final measurements taken at the column experiment indicated that the minimum water content was slightly above 20%.

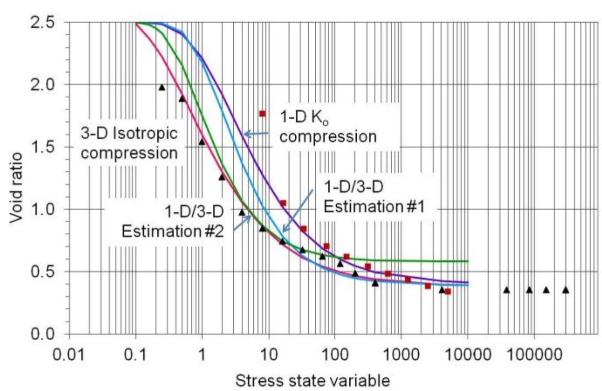

Figure 3. Stress-state versus deformation patterns used during model simulations.

Tailings Hydraulic Conductivity

The measured and best-fit hydraulic conductivity versus void ratio relationship for 2010 tailings is shown in Figure 4. The measured data has been fit using two separate power functions where each power function takes on the form shown in Eq. 1; where, k_s (m/s) is the saturated hydraulic conductivity, C (m/s) is a fitting parameter, e is the void ratio, and D (unitless) is another fitting parameter.

$$k_s = C * e^D \quad [1]$$

The general trend of the permeability function with respect to void ratio, e, was assumed to follow the approximate empirical relationship suggested by Taylor (1948).

$$k_1/k_2 = e_1^3/(1+e_1) \,/\, e_2^3/(1+e_2) \quad [2]$$

where:

e_1 = void ratio at state 1,
e_2 = void ratio at state 2,
k_1= coefficient of permeability at state 1, and
k_2 = coefficient of permeability at state 2.

The coefficients of permeability measured on all Oil Sands tailings samples appear to follow the above relationship between permeability and void ratio.

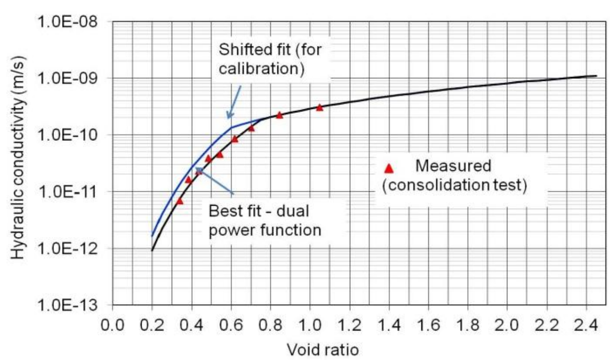

Figure 4. Hydraulic conductivity versus void ratio for Oil Sands tailings.

One power function was used to fit the data when the void ratio was less than 0.75 and another power function was used to fit the data when the void ratio was greater than 0.75. The initial void ratio of the tailings placed in the columns was approximately equal to 2.2. The first permeability measurement (from the consolidation test) was made when the void ratio of the sample was less than 1.1. A number of sensitivity simulations were run to arrive at the "best-fit" dual power function that should be carried forward through the column calibration simulations. In some cases, it appeared that increasing the permeability of tailings in the low void ratio range provided a better fit to the measured data.

LABORATORY COLUMN SIMULATIONS

The data collected during the column experiments was used to verify whether it was possible to numerically simulate tailings behaviour. Key components of the Oil Sand tailings model include the soil properties measured in the laboratory, the software formulation, and the selection of appropriate boundary conditions. The input to the software was used to back-calculate the settlement, water balance, and water content profiles measured during the column tests. The numerical simulations performed under the three de-watering conditions and lift thicknesses summarized in Table 1. (Note: Not all column tests can be presented in this paper).

Table 1. Summary of Column Dewatering Conditions and Lift Thicknesses.

De-watering Condition	Lift Thickness (cm)
Evaporation only	40
	20
Under-drainage only	40
	20
Evaporation and Under-drainage	100
	40
	20

Figure 5 illustrates the interpretation of the back-analysis results, (i.e., plotted as gravimetric water content versus "depth from top of column"). The tailings were initially placed in the columns as slurry with gravimetric water content, *w,* of approximately 83% (55% solids content, C_s). The water content was reduced and the surface of the tailings settled as the tailings were exposed to the de-watering processes. The reduction in water content and overall settlement is shown as a set of water content measurements taken along the depth of the tailings when the columns were dismantled. The water content measurements take on a specific shape depending on the de-watering conditions. The final water content profile shown in Figure 5 is for the 40 cm lift of tailings exposed to "evaporation only" conditions (i.e., no under-drainage). The water content measurement taken at the top of the column shows the final water content at the top of the lift and the overall amount of settlement (e.g., approximately 7 cm). The remainder of the water content profile takes on the form of an "*S*-shaped" curve. The decrease in water content at the top of the column is due to evaporative drying. The reduction in water content at the bottom of the lift is due to self-weight consolidation. Evaporation at the top of the column has reduced the water content of the tailings. Higher water contents remain near the middle of the lift as water moves upward.

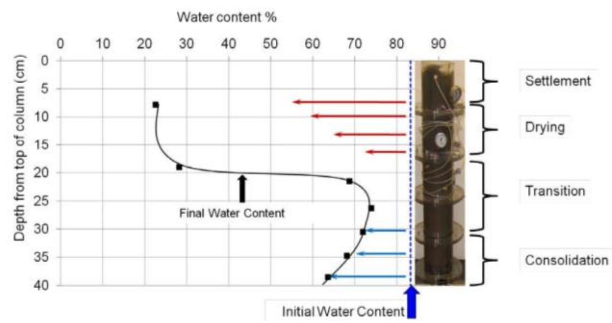

Figure 5. Illustration of measured column test results and numerical simulations.

The computed water content profiles were plotted on the same graph as the water content profile measured at the end of the experiment. Each graph contains four simulated water content profiles corresponding to differing assessments of the material properties. Generally, at least one of the simulated water content profiles provided a reasonable match to the measured data both in terms of the shape of the profile and the overall amount of settlement. The water content profile providing the best match indicates the best representation of tailings behaviour.

Following are a few considerations to bear in mind when interpreting the settlement and water content profile results:

- The undrained shear strength of the 2010 tailings was considered to be greater than 5 kPa when the water content was reduced to values less than or equal to 30%.
- The settlement versus time and final settlement was measured to the lowest point on the surface of the tailings. Differential settlement occurred to varying degrees for the different column experiments. Differential settlement could have occurred as a result of

instrumentation installed near the surface of the tailings or frictional drag along one side of the column and not along the other.

A second part of the back-analysis compared the measured and simulated overall water balance. The water balance includes water evaporating from the surface of the tailings and/or water draining out the bottom of the tailings lift. It is beyond the scope of this paper to present the water balance results.

EVAPORATION ONLY

The “evaporation only” scenario refers to a condition where water can only be removed from the surface of the tailings through evaporation.

40 cm Lift

Atmospheric evaporation occurred from the 40 cm lift continued for a period of 78 days. The measured and simulated water content profiles are presented in Figure 6. The depth of the top most water content measurement indicates that the tailings settled approximately 7.6 cm. The sharp decrease in water content near the top of the lift indicates that evaporation has resulted in the development of tailings crust. The water content of the crust is less than 30% indicating a shear strength of about 5 kPa and a reduced hydraulic conductivity. The water content profile takes on the form of an “*S*-shaped” curve. The reduction in water content at the bottom of the lift is due to self-weight consolidation. The shape of the water content profile indicates that the consolidation process is delayed when compared to the evaporation process. Higher water contents near the middle of the lift are likely due to seepage migrating from the bottom towards the top of column. The upward movement of moisture appears to be delayed by the development of the low permeable crust.

The simulated water content profiles for the four different loading conditions capture the measured behaviour to varying degrees. The “1D/3D estimation No. 2” appears to provide the best-fit to the measured water content profile and the overall amount of settlement. A loading condition with de-watering governed by evaporative drying at the top of the column and self-weight consolidation at the base of the column appears to provide simulation conditions closest to measured values.

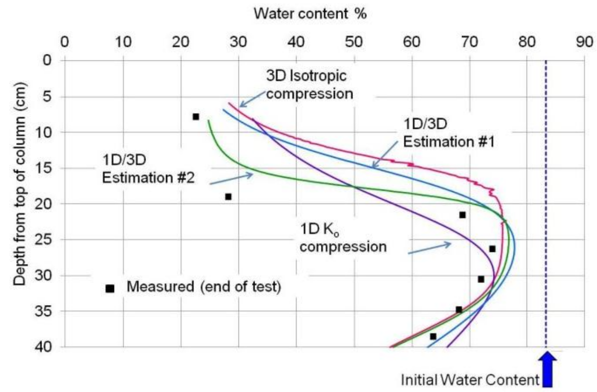

Figure 6. Comparison of measured and simulated results for 40 cm column test subjected to drying and self-weight consolidation.

20 cm Lift

The “evaporation only” 20 cm lift was exposed to laboratory atmospheric conditions for a period of 76 days. The measured and simulated water content profiles are presented in Figure 7. The depth of the top most water content measurement indicates that the tailings settled approximately 4 cm. The water content profile is nearly linear, increasing from a value of approximately 13% at the top of the lift to a value near 22% near the bottom of the lift. All of the water contents are less than 30% indicating that the undrained shear strength is higher than 5 kPa throughout the tailings lift.

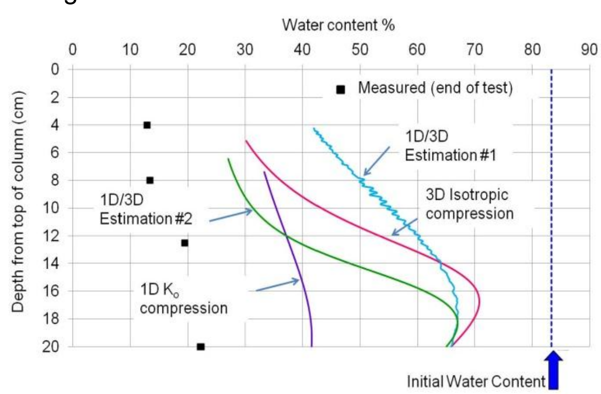

Figure 7. Comparison of measured and simulated results for 20 cm column subjected to drying and self-weight consolidation.

None of the simulated water content profiles for the four different loading conditions provide a close match to the measured data. The difference between the measured and simulated data is attributable to unrealistic boundary conditions. The simulation assumes that the tailings remain in contact with sides of the column. Unfortunately, this assumption was not maintained (See Figure 8).

Figure 8. Shrinkage of the Oil Sands tailings from the walls of the column.

It is anticipated that field behaviour would be somewhere between the simulation and laboratory column conditions. A certain amount of cracking would be expected to occur as the tailings dry but the cracks might not result in the same increase in surface area as tailings separating from a column (i.e., cracks might not penetrate as deep as 20 cm).

UNDER-DRAINAGE ONLY

The "under-drainage only" scenario refers to a condition where water can only be removed through drainage into an underlying sand layer and then out a hole in the side of the column. The top of the column was covered with tin foil to reduce evaporation to near zero values.

40 cm Lift

The "under-drainage only" 40 cm lift was allowed to drain for a period of 76 days. The measured and simulated water content profiles are presented in Figure 9. The depth of the top most water content measurement indicates that the tailings settled approximately 7.5 cm. The water content profile is nearly linear, decreasing from a value of approximately 88% at the top of the lift to a value between 42% and 29% near the bottom of the lift. The sample taken near the bottom of the tailings lift (i.e., water content of 29%) could have contained some of the sand under-drainage material and might not be completely representative of the water content in the tailings. None of the water contents are less than 30% indicating that the undrained shear strength was less than 5 kPa throughout the tailings lift. The slight increase in water content at the top of the column (i.e. from initial water content of approximately 83% to 88%) provides evidence of the self-weight consolidation process. The simulated water content profiles for the 1-D - K_o loading and the estimation No. 1 combined loading condition provided the best-fit to the measured data. The results indicate that the tailings behaviour is governed by self-weight consolidation rather than evaporation (i.e. isotropic loading). The importance of self-weight consolidation is to be expected considering the tailings were covered for the duration of the test.

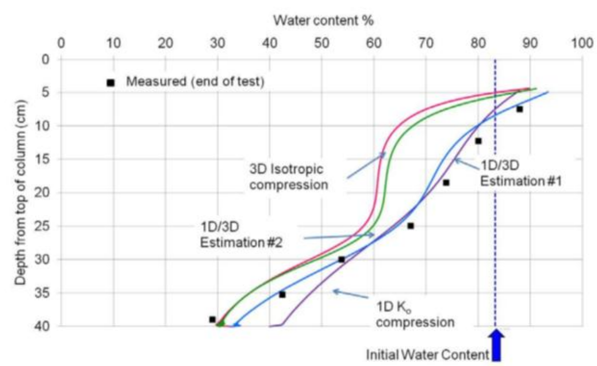

Figure 9. Comparison of measured and simulated results for 40 cm column subjected to under-drainage.

20 cm Lift

The "under-drainage only" 20 cm lift was allowed to drain for a period of 76 days. The depth of the top-most water content measurement indicated that the tailings settled approximately 3.7 cm. The water content profile was nearly linear, decreasing from a value of approximately 69% at the top of the lift to a value near 30% at the bottom of the lift. The two water contents below a depth of 20 cm were taken in the sand drainage layer. None of the water contents in the tailings are less than 30% indicating the undrained shear strength remained less than 5 kPa throughout the tailings lift. The water content at the top of the lift was less than the initial water content. The results are different from the 40 cm lift where the water content increased at the top of the lift. The results are not presented graphically for this case.

EVAPORATION AND UNDER-DRAINAGE

The evaporation and under-drainage scenario refers to a combined condition where water was removed through drainage into an underlying sand layer and through evaporation from the top of the column.

100 cm Lift

The evaporation and drainage 100 cm lift was exposed to laboratory atmospheric conditions for a period of 66 days. The measured and simulated water content profiles are presented in Figure 10. The depth of the top-most water content measurement indicated that the tailings settled approximately 10 cm. The reduction in water content at the top and bottom of the column is evidence of the "double drainage" boundary conditions and the role of evaporation and under-drainage, respectively. The sharp decrease in water content near the top of the lift indicated that evaporation had resulted in the development of a tailings crust. The water content near the surface of the crust was approximately 30% indicating the development of higher shear strength (5 kPa or greater) and lower hydraulic conductivity when compared to the middle of the column. The water content increases from approximately 33% at the base of the tailings to 77% at a depth near 75 cm. The water content was approximately 78% throughout the middle portion of the column.

The simulated water content profiles for the four different loading conditions capture the measured behaviour to varying degrees. The conditions in the column are most closely related to one-dimensional loading and provide the best match to the measured data (i.e., 1-D - K_o loading and 1-D/3-D estimation No. 1). The conditions more closely related to isotropic loading over-predict the settlement and the reduction in water content. The close relationship with one-dimensional loading would be expected considering the thickness of the lift and the fact that evaporation does not appear to significantly penetrate into the column.

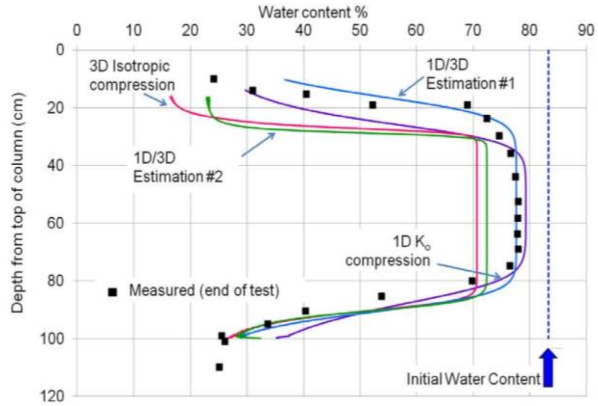

Figure 10. Comparison of measured and simulated results for 100 cm column subjected to drying and under-drainage.

40 cm Lift

The "evaporation and drainage" 40 cm lift was exposed to laboratory atmospheric conditions for a period of 72 days. The measured and simulated water content profiles are presented in Figure 11. The depth of the top-most water content measurement indicated that the tailings settled approximately 9 cm. The reduction in water content at the top and bottom of the column is evidence of the "double drainage" boundary conditions and the role of evaporation and under-drainage, respectively. The sharp decrease in water content near the top of the lift indicated that evaporation has resulted in the development of a tailings crust. The water content near the surface of the crust was between 16% and 33% indicating the development of higher shear strength (5 kPa or greater) and lower hydraulic conductivity when compared to the middle of the column. The absence of constant water content conditions in the middle portion of the lift indicated that the effects of evaporation and drainage were merging as the column thickness was reduced from 100 cm to 40 cm. The water content measurements below a depth of 40 cm were taken in the sand drainage layer.

Three of the simulated water content profiles provide a reasonable match to the measured data. The combined "1-D/3-D estimation No. 2" was selected as providing the best fit (i.e., evaporation governs at the top and consolidation governs near the bottom). It is interesting to point out that unlike the 100 cm column, the one-dimensional loading case provided the poorest fit to the measured data. The results demonstrate that isotropic loading plays a bigger role as the lift thickness decreases.

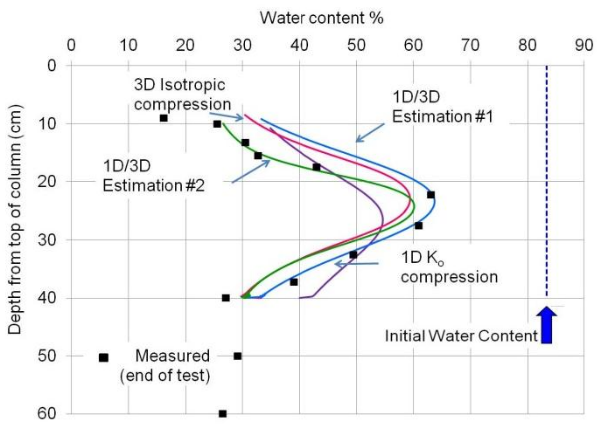

Figure 11. Comparison of measured and simulated results for 40 cm column subjected to drying and under drainage.

20 cm Lift

The "evaporation and drainage" 20 cm lift was exposed to laboratory atmospheric conditions and drainage for a period of 72 days. The measured and simulated water content profiles are not graphically presented for this case. The depth of the top-most water content measurement indicates that the tailings settled approximately 5 cm. The measured water content profile was essentially linear, increasing from a value of approximately 21% at the top of the lift to a value near 30% at the bottom of the lift. All of the water contents were less than 30% indicating the undrained shear strength was higher than 5 kPa throughout the tailings lift. It is no longer possible to identify a portion of the column that was governed by evaporation or drainage when the lift thickness was reduced to 20 cm. None of the simulated water content profiles for the four different loading conditions provided a good match to the measured data. The difference between the measured and simulated results can be attributed to a difference in boundary conditions. The simulation is strictly one-dimensional and is based on the assumption that the tailings remain in contact with sides of the column.

OBSERVATIONS FROM LABORATORY COLUMN SIMULATIONS

The numerical model appeared to be able to simulate the three dewatering processes; namely, evaporation, drainage, and the combined evaporation and drainage condition. In addition, the simulation results are similar to measured values for two out of the three lift thickness conditions (i.e. 40 cm and 100 cm). The back-analysis results provide a level of confidence considering the number of parameters used to calibrate the simulations. The model was evaluated based on the back-analysis of measured parameters including settlement, water content profiles, and water balance parameters (e.g., evaporation and under-drainage). The 20 cm lifts proved to be the most challenging to simulate because the tailings separated from the sides of the columns when evaporation was allowed.

The results of the column tests showed that evaporation was the dominant dewatering mechanism when compared against under-drainage. Single layer sand under-drains appear to be marginally effective in enhancing long-term dewatering due to the reduction in permeability at the interface between the tailings and the sand under drain. Results from the column experiment indicate that lift thicknesses in the order of 20 cm could be dewatered to the required solids content in approximately 55 days (under laboratory environment conditions).

The completion of the study has resulted in significant achievements leading towards the development of a rigorous saturated-unsaturated Oil Sand tailings model. Conventional laboratory testing protocols have been extended with the focus directed towards the characterization of tailings slurries that exhibit large volume changes. The laboratory protocol extensions are related to how tests can be conducted and tested against numerical model simulations. The numerical model proved successful in simulating the dewatering of the tailings under the examined dewatering conditions. The development and verification of the numerical model provides the basis for evaluating the practicality of various tailings management procedures.

ACKNOWLEDGEMENTS

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THE EFFECT OF FLOCCULANT ON THE GEOTECHNICAL PROPERTIES OF MATURE FINE TAILINGS: AN EXPERIMENTAL STUDY

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ABSTRACT

When oil sands tailings are deposited in a tailing pond, they settle and segregate to create a layer of stagnant water on top that is reused in oil extraction and a dense mixture of clay, silt and water on the bottom which is referred to as mature fine tailings (MFT). Dewatering of MFT is a problem due to its very high fines content and relatively low solid content. Atmospheric drying offers a possible solution to this problem. In this technique, MFT are pumped or dredged out of the pond and mixed with polymers that helps to accelerate the release of water from the fines, and then placed on a sloped drying area in a thin layer. When the released water runs off the remaining flocculated MFT will dry to the desired moisture content for removal and replacement in the mine or for subsequent lifts. This paper presents a laboratory study on the flocculated MFT. With a series of tests the basic properties and consolidation and desiccation behavior of flocculated MFT are assessed. The results were compared with original untreated MFT, which confirmed that in case flocculants are properly mixed with MFT, they have a positive effect on the dewatering rate of the MFT. However, the final volume of the consolidated flocculated MFT when exposed to atmospheric drying can be significantly larger than the untreated MFT. The results provide a useful reference for future pilot-scale tests and commercial implementation of the atmospheric drying method on flocculated MFT.

INTRODUCTION

Oil sands tailings are a byproduct from the oil extraction process used in mining operations. Normally, the produced tailings are discharged directly into a tailings pond for storage. In tailings ponds, tailings segregate relative to the grain size distribution and settle further to create a layer of stagnant water on top that is drained and re-used in bitumen extraction process and a mixture of clay, silt and water on the bottom or the central part of the pond which is referred as mature fine tailings (MFT). Dewatering MFT forms a problem for engineers due to the very high fines content (about 80%-90%, fines are defined as the particle size <44 μm) and a relatively low solid content. In order to maximize the proportion of recycled water and to reduce the volume of fine tailings, a variety of tailings dewatering options are being investigated. Among these options, atmospheric drying is being tested by several companies and considered as a promising method in disposal of fine tailings. In this method, MFT is dredged from the tailings pond and mixed with flocculants - chemical agents that help to aggregate the fines clay particles in the MFT together and accelerate the release of water from the fines - before being placed in a relative thin layer on a sloped surface. The released water runs down the sloped surface to a collection area and is reused for extraction. The remaining deposits will dry under the influence of environmental factors (radiant heat, wind, humidity, and drainage) to the desired moisture content for removal and replacement in the mine, or subsequent lifts. Under ideal conditions, a significant amount of the water is recovered leading to large volume reduction of the tailings.

Knowledge regarding the basic physical properties of the tailings and their consolidation and desiccation behaviors is necessary to understand the material behavior of the tailings and to further improve the efficiency of disposal (Qiu and Sego 1998). This paper presents experimental research on flocculated MFT which aims to ascertain the basic properties of the material and to examine its consolidation and desiccation behavior. The obtained data are compared with that of the original MFT to evaluate the effects of flocculation on the behavior of tailings The data generated will provide necessary parameters for numerical simulation with models and a useful reference for future pilot-scale tests and commercial implementation of the atmospheric drying method.

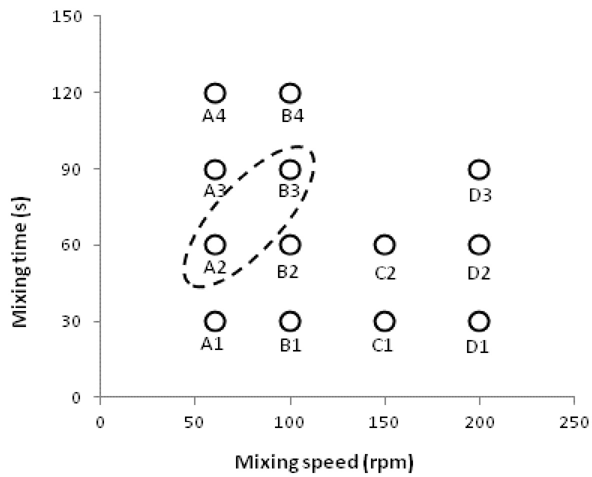
FLOCCULATION OF MATURE FINE TAILINGS

The MFT tailings used in this program originate from Shell Albian Sands Muskeg River Mine located in Fort McMurray, Canada. Several drums of tailings, 180 L each, were delivered for experimental research. The drums were agitated for about 30 minutes before the tailings were stored in 15L containers in an ambient environment away from direct sunlight.

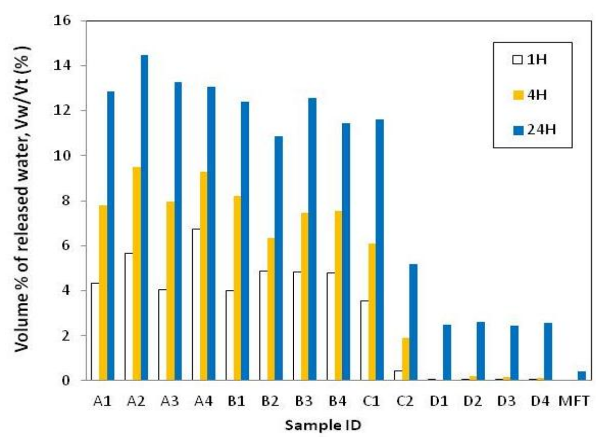
To produce the flocculated tailings for fast dewatering, one type of polymer provided by a commercial flocculent supplier was used to flocculate MFT in laboratory. The flocculent was prepared by mixing dry powder polymer with water decanted from tailings to form a solution at a concentration of 4 g/L as recommended by flocculent supplier. Flocculent solutions were then added to the MFT slurries at a dosage of 1g of dry flocculent to 1kg of dry solids. This relatively high flocculent dosage was suggested by the related research institutions based on the results of field tests. Actually, the real flocculent dosage used in engineering may vary with many factors such as the initial solid content, specific gravity, grain size distribution and pH etc. to obtain the optimum dewatering results. However, for the convenience of this research, all the tests presented in this paper were carried out on the flocculated MFT at the dosage of 1g/kg.

Several researchers (Jeeravipoolvarn, 2010; Munoz et al, 2011) found that the mixing conditions (mixing speed and mixing time) also play an important role on the results of flocculation. As stated by Munoz et al., there is a critical balance between the amount of polymer and the mixing conditions that dictate the effectiveness of the ability to release water from tailings. When the mixing conditions are ideal, water release is maximized. Munoz et al, further pointed out that the optimum mixing conditions produce large flocs with connected “macropores” which act as water channels and help to dewater the mixture. When the MFT-polymer mixture is “over mixed” there is breakdown in the flocs structure and a reduction or elimination of the water channels.

In order to evaluate the optimum mixing conditions with respect to the MFT with an initial solid content of 35%, a sands fines ratio (SFR) of 1:4 (80% fines) and a flocculent dosage of 1g/kg, a series of sedimentation tests were performed. The optimum was defined as the conditions that ensured maximum water recovery and minimized turbidity of filtrate. MFT stored in containers were agitated for several minutes with mixer before transferring to 500 ml beakers. Flocculent solutions of required volume were added to MFT when mixing started. The flocculated tailings were filled into a group of 250 ml cylinders for observation. See Figure 1a, the mixing conditions varied from 60 to 200 rpm with mixing speed and 30s to 120s of mixing time for different samples. The initial settling rate and volume of water released till 24 hours were monitored (see Figure 1b). The clarity of the filtrate was observed visually.



(a)



(b)

Figure 1. (a). Mixing conditions for different samples. (b). Observed volume percentage of released water (i.e. ratio between water volume and total sludge volume) changing with time.

It can be seen from Figure 1b that sample A2 (60 rpm×60s) was leading in settling rate and the amount of water released. On the other hand, B3 (100 rpm×90s) was superior to the samples of Group A from the aspect of turbidity of filtrate. For overall consideration, the optimal conditions for this particular bench scale experiment should fall into the area shown in Figure 1a, thus a balanced condition— mixing time is between 70 and 90 seconds coupled with a mixing speed of 70 to 90 rpm. In this paper a mixing time of 80 seconds with a mixing speed of 60 rpm was used for all experiments. We can also see that there was only 1 ml water released after 24 hours for untreated MFT, which proved the effectiveness of this type of flocculent in speeding up the settling rate. After mixing, the flocculated MFT were stored in containers and allowed to settle. After settling, the stagnant water on top was removed and the remaining tailings were used for further experiments which are described in the following chapters.

BASIC PHYSICAL PROPERTIES

A series of laboratory tests including the solid density, Atterberg limits, and particle size distribution etc. were performed to obtain the basic physical properties of original MFT and thickened tailings (TT) (Yao et al, 2010,2012). In this paper, in order to evaluate the effect of flocculent on the basic properties of MFT, Atterberg limits were re-determined for both MFT and flocculated MFT using the method described in ASTM Standard D4318 (2002). The results are presented in Table 1. The liquid limit of original MFT was 47.2%. After flocculation the liquid limits increased to 65.3%, 66.5% and 68.5% with respect to the varying flocculent dosage at 0.5g/kg, 1 g/kg and 1.5 g/kg. For untreated MFT the plastic limit was 21.5% which is close to that of flocculated tailings. These results suggest that flocculation of tailings will increase the liquid limit and an increase of the flocculant dosage results in a higher plasticity index. Comparing the MFT and flocculated MFT tested in this paper with data from earlier work by Yao et al (2010, 2012) on MFT, originating from the same stock barrel, but sampled at different times, shows that each batch of MFT can differ in basic properties, so properties from one MFT cannot be generalized for all MFT or other types of oil sand tailings.

SHRINKAGE LIMIT AND SHRINKAGE CURVE TESTS

For mine tailings that swell and shrink under the influence of environmental factors during deposition, the relationship between their changes in bulk volume and changes in water content is a necessary parameter for the geotechnical engineer to predict the tailings behavior. To obtain a quantitative indication of the amount of volume changes that occur due to sub-aerial drying of the tailings, a shrinkage limit test and a shrinkage curve test are necessary. The shrinkage limit of a soil is defined as the water content corresponding to the minimum volume that a soil can attain upon drying to zero water content. It can be used to evaluate the shrinkage potential, crack development potential, and swell potential of cohesive soils. The mercury immersion method was originally used to determine the shrinkage limit in the laboratory, however, this method is no longer considered acceptable in most countries due to health safety concerns.

Table 1. Basic Properties of the Tailings.

Property Index	MFT*	MFT	Flocculated MFT		
			0.5g/kg	1.0 g/kg	1.5g/kg
Density of solids (g/cm3)	2.31	—	—	—	—
Liquid limit (%)	57	47.2	65.3	66.5	68.5
Plastic limit (%)	27	21.5	22.9	22.7	21.1
Plasticity index (%)	30	26.4	42.4	43.8	47.4
Shrinkage limit (%)	—	14.1	—	15.3	—
Fines content (<44 μm; %)	80	—	—	—	—
Sand content (>44 μm; %)	20	—	—	—	—

*****Data from Yao et al (2012)

Another method introduced by the ASTM standard D4943 (2002) is widely accepted as an alternative to the mercury method. In this method, wax is used for the measurement of the volume of the soil specimen instead of mercury. In this research, shrinkage limit tests were performed on the tailings using a wax method, the results are shown in Table 1.

Shrinkage characteristics curve describes the relationship between bulk volume and water content which is normally expressed by void ratio (i.e. the ratio between the void volume and the solid volume) versus gravimetric water content (i.e. the ratio between the water mass and the total mass). To determine the shrinkage characteristics curve, continuous measurement of bulk volume of the soil specimen during drying is required. In this paper, the balloon method described by Tariq & Durnford (1993) was selected to determine this curve. The advantage of this method is that the setup (see figure 2) is easily constructed and the whole test can be completed for the full moisture range in a short period (3-4days) with an air pump which passes air with low pressure (100L per hour) over the sample. With this method, the balloon can be filled with MFT as slurry with a high water content before applying air flow to dry the tailing. At regular time intervals, the specimen was weighed and the volume was determined by submerging the balloon into the water and measuring the mass of water replaced by the soil using Archimedes principle. A small vacuum was applied to ensure a perfect fitting of the balloon to the soil sample before the above procedures. Figure 3 presents the shrinkage curves of flocculated MFT and non-flocculated MFT.

From the graph provided in Figure 3, three shrinkage stages can be identified based on the theories related to unstructured clay during drying (Bronswijk, 1991). Take the shrinkage curve of MFT as an example, the first stage (from A to B, in figure 2) is called normal shrinkage or basic shrinkage that is characterized by the equal decrease in water volume and in bulk soil volume. The second stage is called residual shrinkage starting from point B where the soil begins to desaturate. Generally, this point is close to the plastic limit of the soil. In this test the water content of MFT at point B is about 20% and the measured plastic limit is of 21.5%.

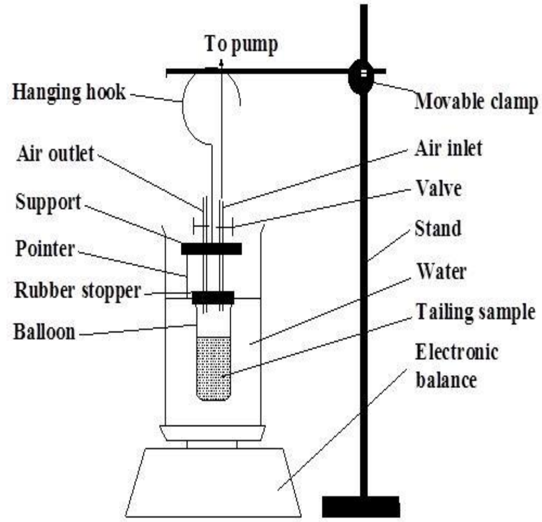


Figure 2. Setup used to determine soil shrinkage characteristic curve using Tariq & Durnford balloon method.

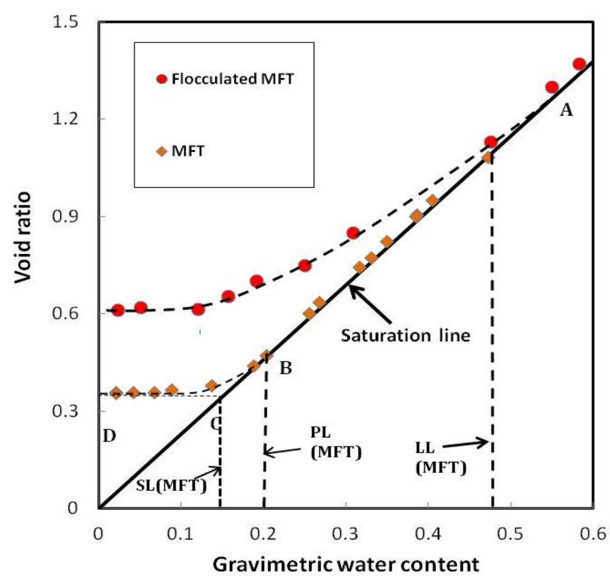


Figure 3. Shrinkage curves of the tailings determined with balloon method.

Upon further drying, point C is reached where the final stage of zero shrinkage starts. In this stage the soil particles have reached their densest configuration and the volume of the aggregates stays constant as the water volume further decreases. This can be referred to as the true shrinkage limit of the soil and the gravimetric water content appears to approximately correlate with residual soil conditions (Fredlund, 2011). The water content of MFT at point C is about 14% while the measured shrinkage limit is 14.1%. For flocculated MFT, the water content at the desaturation point is more than twice the measured plastic limit. This phenomenon is related to flocculation which changes the clay structure and properties. Finally, the comparison between the MFT with and without flocculent indicates that for the final void ratio of MFT when completely dried out was almost half the final void ratio of flocculated MFT. Apparently, flocculated MFT requires considerably more volume after complete drying than untreated MFT.

CONSOLIDATION TESTS

The purpose of the consolidation tests was to determine the consolidation behavior of tailings over the effective stress range of 1-100 kPa which is operative in the majority of tailings management facilities. One-dimensional consolidation tests were performed in accordance with the procedure described in ASTM D2435 (2004). Samples were prepared by air drying the tailings to the required moisture content in a controlled environment. Load increments that applied stresses of 2.3, 4.6, 9, 18, 36, 72 and 144 kPa were used for both MFT and flocculated MFT specimens. Each stress increment was maintained for at least 24 hours until the primary consolidation was finished. At least three specimens were tested for each type of tailings and one group of results is presented in this paper. Figure 4 provides the relationship between void ratio (*e*) and logarithm of effective stress (e-log p curve) for both tailings. The initial values of water content for flocculated MFT and MFT were 54% and 49%, which are close to the liquid limits. It can be seen that over the stress range of 4 to 150 kPa, the data points for each tailings formed a straight line. The slope of this line is denoted as the compression index c_c in soil mechanics theory which reflects the compressibility of the soil. A highly compressible soil will have large value of c_c. Other consolidation parameters such as coefficient of consolidation (c_v) and coefficient of volume compressibility (m_v) were calculated from the test results from each load increment. Moreover, the saturated hydraulic conductivity of the soil specimen from each load increment was calculated using the consolidation parameters obtained. The main results of the consolidation tests and calculated hydraulic conductivity are shown in Table 2.

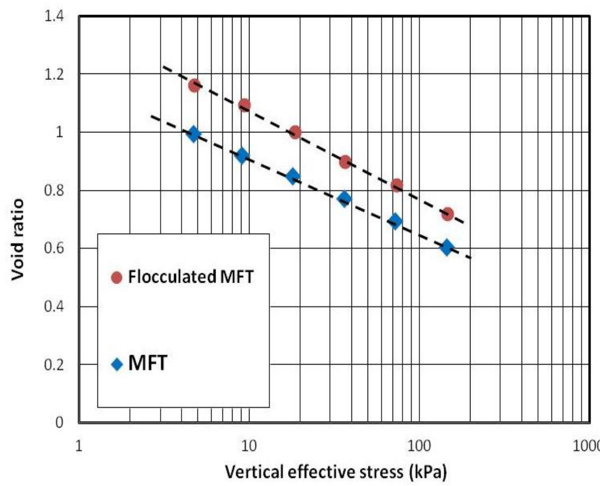


Figure 4. The compressibility of oil sands tailings.

Compared with the original MFT, the flocculated MFT have slightly higher values in the compression index c_c and an almost three times higher saturated hydraulic conductivity. The results of these tests suggest that flocculation of fine particles of MFT used in this paper, doesn't have a significant effect on the compressibility and a limited effect on the saturated hydraulic conductivity.

Table 2. Consolidation Tests and Calculated Saturated Hydraulic Conductivity of Oil Sands Tailings.

Property index	c_c	c_v (m^2/year)	m_v (m^2/MN)	k_s (m/s)
MFT	0.24-0.30	0.063-0.16	1.1-20	4.1×10^{-11}-6.9×10^{-10}
Flocculated MFT	0.27-0.32	0.07-0.16	1.4-14	4.9×10^{-11} - 5.0×10^{-10}

LABORATORY VANE SHEAR TESTS

The vane shear test is a relatively simple, quick, cost-effective geotechnical testing method used to estimate the undrained shear strength of soft to firm cohesive soils. This test can be carried out both in the field and in the laboratory. In this paper, a series of vane shear tests were performed in laboratory by following the procedure described in British Standard (BS 1377 (1990)) on the tailings specimens. Materials used for the vane tests were taken from the tailings storage which were contained in large buckets and placed in the controlled environment for air drying. At different time intervals, soil samples were taken from different positions and depths of the tailings with a sampler which is about 38mm in diameter and 80 mm in height. Besides the vane shear strength, water content and the density of each sample were also measured. Relationship between the void ratio and the undrained shear strength in each type of tailing was established through tests, shown in Figure 5.

It is well known that the undrained shear strength of a remolded clayey soil increases rapidly as the water content decreases (ref). In this test, for MFT with void ratio of 1.35 (about 60% in water content) the measured shear strength was about 1kPa, this value climbed up to 190 kPa when the void ratio decreased further to 0.48 (about 16% in water content). Comparing the undrained shear strength curves of untreated MFT and flocculated MFT, the latter curve was located slightly higher than the former one in the graph.

In other words, at the same void ratio the untreated MFT shows lower shear strength than flocculated MFT or at the same shear strength flocculated shows a higher void ratio than non-flocculated MFT.

In practice tailings produced from the extraction plant are normally adjusted to high water content slurries for the purpose of transport and deposition, however, for successful reclamation the excess water must be removed from the tailings and a minimum shear strength must be achieved. According to Directive 074, the minimum shear strength required for reclamation is 5 kPa. According to figure 5 an undrained shear strength of 5 kPa for MFT, flocculated MFT is obtained at void ratios of about 1.0, 1.1, respectively.

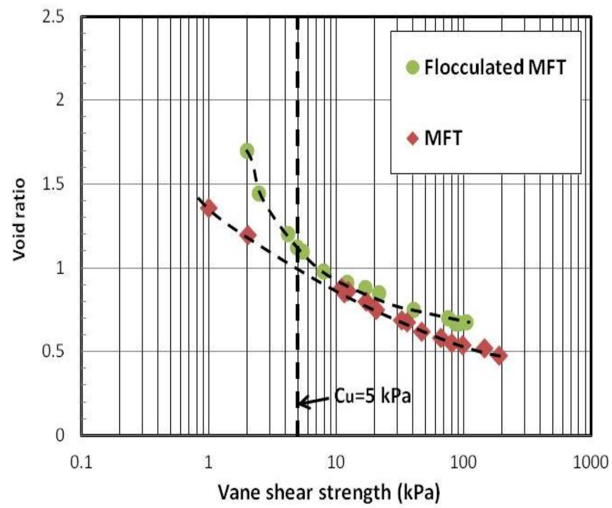


Figure 5. Vane shear strength of tailings.

WATER RETENTION CHARACTERISTIC TESTS

The water retention characteristics tests are used to determine the soil water characteristics curves (SWCC). The SWCC is an empirical relationship between soil suction stress and water content. Soil suction stress is one of the most important factors affecting the drying behavior of (unsaturated) soils. In atmospheric drying, the flocculated tailings are deposited in thin layers and allowed to desiccate and eventually desaturate under the influence of environmental factors (i.e. humidity, temperature, wind, etc.). The SWCC is used in numerical models to predict the change in volume and water content when water is evaporated from the soil surface.

The filter paper method, described in ASTM Standard D5298 (2002), was used to determine the water retention curves of fine tailings because of its advantages over other suction measurement methods. The working principle of this method is that the filter paper will come to equilibrium with the soil, and suction value of the filter paper and the soil will be the same at equilibrium. The filter paper needs to be calibrated before use. The purpose of calibration is to determine the relationship between soil suction stress (total suction or matric suction) and the measured moisture content of filter paper at equilibrium. ASTM D 5298 employs a single calibration curve that has been used to infer both total and matric suction measurements. However, Houston et al

(1994) proved that the total suction calibration curve and the matric suction calibration curve of one soil don't match thus should be determined separately. Bulut (1996, 2001) determined the calibration curves with Schleicher & Schuell No. 589-WH filter paper for both total and matric suction measurement. In this paper, the same type of filter paper was selected and the calibration curves were used. The determined water retention curves for non-flocculated MFT and flocculated MFT are shown in Figure 6. The figure shows that particularly in the low ranges of suction flocculated MFT shows significantly higher water content than MFT without flocculent. At a suction stress above 200 kPa (water content of about 25%) the SWCC of all three types of tailings are more or less equal.

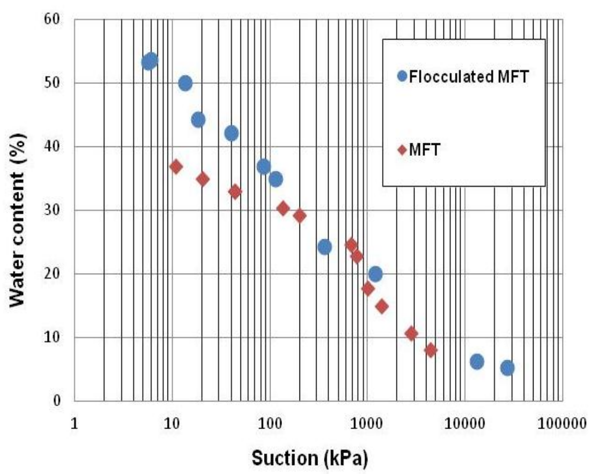


Figure 6. Water retention curves of the tailings.

DISCUSSION AND CONCLUSIONS

In this paper, the properties including basic physical properties, shrinkage characteristics, consolidation behavior, vane shear strength and soil water characteristics were compared between untreated MFT and flocculated MFT through laboratory tests. From the comparison the following conclusions could be drawn:

(1) The amount of flocculent and the mixing procedure (Mixing speed and duration) have great impact on flocculation results and thus control the dewatering behavior of the flocculated tailings. The optimum mixing condition for MFT with an initial solid content of 35% and SFR of 0.25 at the dosage of 1g/kg was determined in this paper at 70—90 seconds at a mixing rate of 70-90 rpm.

(2) Sedimentation tests on the flocculated MFT demonstrated the effectiveness of this type of flocculent in accelerating the settling rate and the release of water from the tailings.

(3) Regarding the Atterberg limits, the liquid limit of MFT increased from 47% to 67% while the plastic limit and shrinkage limit did not change much after flocculation.

(4) The shrinkage curves showed that flocculated MFT already started to desaturate at a high plastic index (i.e. a high water content compared to plastic and liquid limit.) in contrast to original MFT which started to desaturate close to the plastic limit. Also these curves showed that the void ratio of completely dried flocculated MFT was about 0.6 which is about twice the void ratio of completely dreied MFT.

(5) The one-dimensional consolidation tests showed that within the stress range of 2 to 150 kPa flocculated MFT and non-flocculated MFT tailings show similar compressibility, but flocculated MFT has a higher void ratio. The measured consolidation parameters (c_v, m_v and k_s) of the flocculated MFT are close to those of MFT.

(6) The minimum undrained shear strength of 5 kPa which is the requirement for successful reclamation is reached at a higher void ratio for flocculated MFT compared to original MFT.

(7) The SWCC shows a significantly higher water content for flocculated MFT at suction stresses below 200 kPa. Above 200 kPa differences between the three types of tailings are insignificant.

Combining the various experimental results it can be concluded that adding flocculants has a positive effect on the dewatering rate of the MFT (when properly mixed). The flocculated MFT settles faster and the permeability of the settled flocs is higher than the MFT without the flocculent. However, the high liquid limit, higher void ratio during one-dimensional compression and high water content at suction stresses below 200 kPa in the SWCC indicate that flocculated MFT binds more water that the MFT without flocculent, particularly under low stress conditions. The final volume of the consolidated flocculated MFT when exposed to atmospheric drying can be significantly larger than the MFT without flocculent. These findings seem to indicate that disposal of flocculated tailings requires larger tailing ponds. On the other hand the fact that the required shear strength is reached at a higher void ratio might allow earlier access on the reclaimed tailings to take additional measures which can further accelerate consolidation, like planting trees.

Although the results of the various tests seem to be consistent, they might differ for various types of tailings and should be validated based on field observations. For example the optimum mixing rate and duration are likely to be valid only for the mixing equipment used in our experimental program. Therefore the optimum mixing conditions should probably be redefined for different mixing equipment, sludge and flocculent types. Further investigation in the flocculation mechanism is required to explain the observed behavior. Secondly most tests were performed at stresses above 2 kPa, while significant volume change might occur in the relatively low stress range from 0-2 kPa. Large strain consolidation tests must be performed to evaluate the full consolidation and desiccation behavior as a result of atmospheric drying.

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NUMERICAL MODELING OF DRYING AND CONSOLIDATION OF FINE SEDIMENTS AND TAILINGS

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ABSTRACT

The extraction and processing of many mineral ores result in the generation of large volumes of fine-grained residue or tailings. These fine sediments are deposited as a slurry with very high water contents and lose water after deposition due to self-weight consolidation. When the surface is exposed to the atmosphere they dry due to evaporation. In thin lifts deposition both processes, consolidation and drying take place simultaneously. The understanding and a quantitative description of both processes is important for the prediction of the dewatering process of fine sediments and tailings. A numerical model, which is able to simulate one-dimensional vertical flow due to consolidation and drying by evaporation of fine sediments, is implemented in a MATLAB code.

The proper functioning of the model was verified for several combinations of boundary conditions, including a closed boundary, a constant flux boundary, a constant pressure boundary for both the top or the bottom and deposition of a fresh layer on top of a dried one. Subsequently the model was validated using data from experimental research on fine oil sand tailings. The model showed realistic outcomes for most boundary conditions and good correspondence with the observed behaviour in the drying and consolidation experiments. The paper describes the model, and the verification and validation simulations.

INTRODUCTION

Canada accommodates an enormous oil supply from oil sands. The extraction of oil from the sand creates a significant amount of slurry. The sludge, which consists of approximately 80 to 90% water, has been stored in temporarily basins. Dewatering by self-weight consolidation takes up decades. In order to accelerate the dewatering process, the use of so-called *mud farming* has been proposed. Mud farming also called sub-aerial drying represents the process where the sludge is deposited in thin layers, in which every layer loses water mainly by evaporation. Thin lift deposition is a potential solution, and involves deposition in thin lifts that are subsequently allowed to dry and gain strength. When the layer has lost a sufficient amount of water, the next layer can be deposited on top resulting in consolidation of the layers below. To determine the thickness of the applied layers and the required time to dewater the deposited layers, a model that incorporates the drying and consolidation process is required.

Both the consolidation process as the drying process of the slurry involve large volume reduction. Large strain consolidation has been described by the finite strain consolidation (Gibson et al., (1967); Schiffmann et al., (1988), to be able to model the self-weight consolidation of fine sediments. The drying process has been described by several authors mainly from the agriculture-related disciplines e.g. Reniersce (1983) who modeled the ripening of soils gained from the sea. More recently this process is described by, Wilson et al. (1994); Wilson et al., (1997), to model the atmospheric drying process of soils. The theory of the combined process of sub-areal drying, self-weight and overburden consolidation has been described by Kim et al, (1992).

This paper describes the implementation, verification and validation of this combined process in a MATLAB based computer code. For the validation simulations of experiments of the drying and consolidation process of fine oil sand tailings were used.

THE MODEL

In order to understand the calculation process of the model the basic theory as described by Kim et al, (1992) is repeated in this paragraph.

The model is based on the following correlations:

- Water ratio and void ratio (shrinkage curve);
- Water ratio and matrix potential (pF-curve);

- Water ratio and hydraulic conductivity (permeability curve).

The model basically is a flow model in which Darcy's law governs the process. After dividing a slurry layer in a finite number of sub layers the flux (v) over all modelling layers with thickness (Δz) can be calculated which gives the change in water ratio $(\Delta\vartheta)$ per unit of time (Δt). The loss of water per layer is equal to the change in water ratio per layer:

$$\frac{\Delta\vartheta}{\Delta t} = -\frac{\Delta v}{\Delta z} \qquad [1]$$

By using the shrinkage curve, see Figure ***1***, the correlation between the water ratio and the void ratio the loss of water is translated to a loss of volume and in a 1D case to settlement.

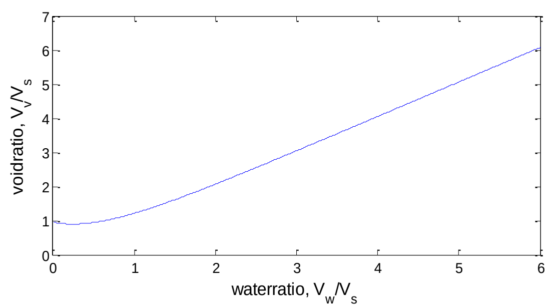


Figure 1. The Correlation Between The Water Content And Void Ratio (Taken From Kim Et Al, 1992).

The fluxes between all modelling layers are being calculated with the use of Darcy's law:

$$v = -k \cdot \nabla i \qquad [2]$$

$$\nabla i = \frac{d\phi}{dz}$$

v = water flux [cm/day]
k = vertical permeability [cm/day]
∇i = gradiënt in water pressure [-]
$d\phi$ = difference in water pressure [kPa]
dz = thickness of the modeling layer [cm]

The water pressure is built from 3 components (Philip, 1969), see Equation 3:

$$\phi = \gamma_w \cdot z + \psi + \Omega \qquad [3]$$

ϕ = water pressure [*kPa*]
z = gravitational component [*m*]
ψ = matric potential [*kPa*]
Ω = overburden component [*kPa*]
γ_w = specific weight of water [kN/m^3]

In the original paper the dimensions of the components in equation 3 are given in cm (water pressure). The component of overburden is being determined by the following formulas:

$$\Omega = \frac{de}{d\theta} P(z) \qquad [4]$$

$$\mathrm{P(z)} = P(0) + \int \gamma_b dz \qquad [5]$$

$$\gamma_b = \frac{\vartheta\gamma_w + \gamma_s}{1+e} \qquad [6]$$

$P(z)$ = total stress at depth *z* [kPa]
$P(0)$ = external load [kPa]
γ_b = bulk density [kN/m^3]
dz = thickness of modelling layers [cm]
ϑ = water ratio $\left(\frac{V_w}{V_s}\right)$
e = void ratio $\left(\frac{V_p}{V_s}\right)$
γ_s = specific weight of the grains [kN/m^3]

Kim et al, 1992 use the water retention curve as general relation between the water ratio and stress. Figure 2 presents a water retention curve (Van Genuchten, 1980), the stress strain relation that was found by Yao et al(2011) for thickened tailings, in consolidation experiments. For the saturated part they match quite well.

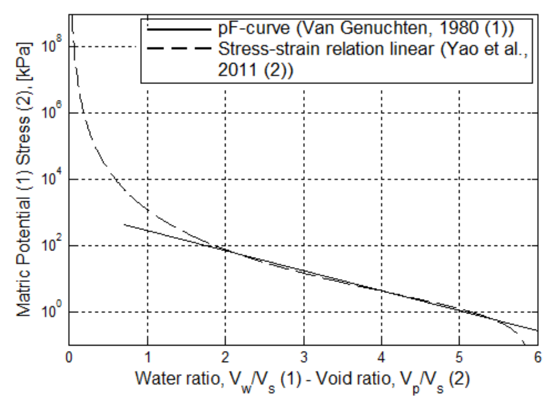


Figure 2. Correlation Between Effective Stress And Void Ratio.

The third relation is a relation between the permeability and the water ratio. As the soil becomes more compact, the permeability decreases. The permeability decreases even more when the pores become unsaturated. A similar relation is given in Figure 3.

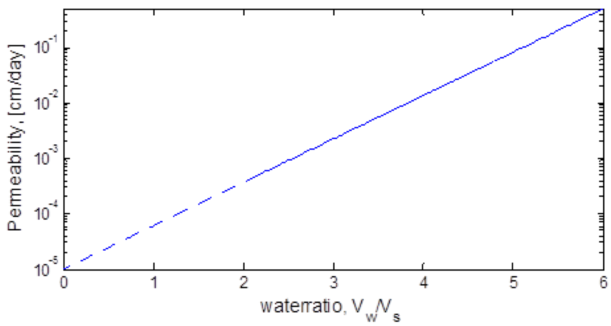


Figure 3. Permeability Which Has Been Used By Kim In Order To Model Ripening Of Marine Clay (Kim Et Al, 1991).

For one-dimensional flow the total flow equation is as written in Equation 7:

$$v = -K\left(\left\{\frac{d\psi}{d\vartheta} - \frac{d^2e}{d\vartheta^2}\left[P(0) - \int \gamma dz\right]\right\}\frac{\partial\vartheta}{\partial z} - \gamma\frac{de}{d\vartheta} + 1\right) \quad [7]$$

For saturated compaction the decrease in water equals the decrease in void ratio $\left(\frac{de}{d\vartheta} = 1 \; en \; \frac{d^2e}{d\vartheta^2} = 0\right)$, the general flow equation can be simplified by equation 8.

$$v = -K\left(\frac{d\psi}{dz} + 1 - \gamma\right) \quad [8]$$

In case of large strain systems, usually one makes use of a *material coordinate system* (Smiles and Rosenthal, 1968; Philip, 1968). In these kind of systems the regular coordinate system (dz) will be written to a so called material coordinate system (dm) where the layer thickness will be kept at a constant value.

$$dm = \frac{dz}{1+e}$$

dm = thickness of modeling layers in the *material coordinate system* [cm]
dz = thickness of the modeling layers in a regular coordinate system [cm]

The differential equation now becomes:

$$\frac{\partial\vartheta}{\partial t} = -\frac{\partial v}{\partial m} \quad [9]$$

Kim et al., 1992 reduce the general equation according one dimensional groundwater flow for non-steady state situations to Equation 10, in case without surface loading $(P(0) = 0)$:

$$\frac{\partial\vartheta}{\partial t} = \frac{\partial}{\partial m}\left[K^*\left(\frac{d\psi}{dm} + SF1 + SF2 \cdot \frac{\partial\vartheta}{\partial m}\right)\right] \quad [10]$$

With

$$SF1 = (1+e) - (\vartheta + \gamma_s)\frac{de}{d\vartheta} \quad [11]$$

$$SF2 = \frac{d^2e}{d\vartheta^2}\int_0^m (\vartheta + \gamma_s)dm \quad [12]$$

In this equation m represents the material coordinate (positive upward), $K^* = \frac{K}{1+e}$, $SF1$ is the component of gravitational and overburden potential term and $SF2$ is the component of contribution by overburden potential arising from the difference in volume shrinkage (residual zone). In these differential equations only three parameters are unknown: $e, \psi \; en \; K$. All three parameters can be written as a function of the water ratio (ϑ), Figure 1,2, and 3. These correlations depend on the type of soil and can be determined by lab testing.

BOUNDARY CONDITIONS

For both top and bottom of the model a boundary condition needs to be specified.

Top boundary condition

According to the top boundary, there are three different situations:

1. Under water consolidation (self-weight consolidation).
2. Drying by evaporation;
3. Precipitation.

Situation 1 (self-weight consolidation)

With regard to the first situation, the density of the top modelling layer does not change. There is no compaction, as there is no surface load or atmospheric suction. Zero compaction means no change in flux or matric potential $\psi = 0$. For an already consolidated layer, deposed under water, ψ is not equal to zero any more. In this case the zero-flux-boundary-condition is used. In other terms; the flux over the bottom of the top modelling layer is equal to the flux over the top of the top modelling layer. The delta flux over the top modelling layer is therefore equal to zero, see Figure 4.

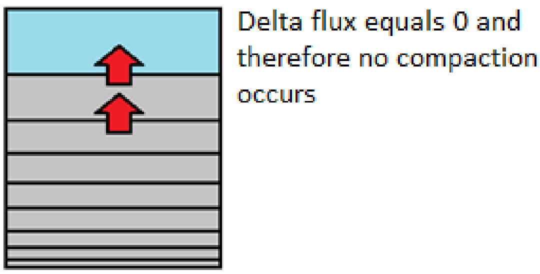


Figure 4. In Case Of Underwater Consolidation, Delta Flux Of The Top Modelling Layer Needs To Be Equal To Zero.

Situation 2 (evaporation)

Evaporation can be modelled by making use of both a flux and a matric potential (Rijniersce, 1983). By setting the flux equal to the open water evaporation and setting the matric potential to a value corresponding to the relative humidity of the atmosphere, the outflow of water at the top boundary will never exceed the open water evaporation and also will never dry out more than the given matric potential corresponding to the relative humidity, see Equation 13.

$$q_{1\to atm} = \min\left(E, k_v \cdot \frac{\psi_{atm}-\psi_1}{0{,}5\cdot\gamma_w\cdot dm}\right) \qquad [13]$$

$q_{1\to atm}$ = evaporation from the soil [*cm/day*]
E = maximum open water evaporation [*cm/day*]
ψ_{atm} = matric potential which corresponds to the atmospheric relative humidity [*kPa*]
ψ_1 = matric potential which corresponds to the top modelling layer [*kPa*]
k_v = vertical permeability of the soil [*cm/day*]
dm = thickness of the top modelling layer [*cm*]

Situation 3 (precipitation)

Precipitation can be modelled with the use of a potential or a flux.

Bottom boundary condition

According to the bottom boundary, there are three different situations:

1. The bottom boundary is closed off.
2. An open boundary which drains towards deeper layers without any constrain.
3. A boundary characterized by a fix seepage.

Situation 1 (zero flux)

The situation where no drainage or seepage occurs can be modelled by setting the bottom boundary to a flux equal to zero.

Situation 2 (drainage)

In order to model drainage, one needs to make use of a matric potential. Straight after deposing the sludge, water will drain at the bottom. This results into a water pressure equal to zero. This corresponds to a situation where the total pressure equals the effective pressure. The bulk weight of the upper lying sludge now rests on the grains at the bottom.

As long as the sludge remains saturated, the matric potential is equal to the effective stress. In order to set the right boundary condition this needs to be realized.

$$\psi_{bottom} = -\frac{\vartheta\gamma_w+\gamma_s}{1+e}\cdot z \qquad [14]$$

ψ_{bottom}= matric potential at the bottom boundary [kPa]

z = thickness of the soil layer [m]

Situation 3 (seepage)

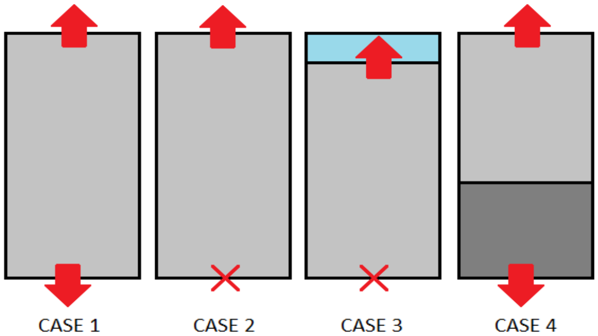
In order to model seepage, a flux is used.

$$q_{bottom\to N} = seepage \qquad [15]$$

$q_{bottom\to N}$ = flux over the bottom boundary [cm/dag]

VERIFICATION

In order to verify the model, multiple simulations have been made. The results have been verified by checking the mass balance. For the cases without evaporation (only self-weight consolidation) the expected effective stress has been compared to the calculated matric potential. The simulations are characterised by the boundary conditions in Figure 5.



1. Evaporation at the top boundary and outflow at the bottom boundary.
2. Evaporation at the top boundary and zero flux at the bottom boundary.
3. Stagnant water at the top boundary and zero flux at the bottom boundary.
4. Similar to case 2, but a second fresh layer is being deposed on an partly dried layer.

Table 1 summarizes the used boundary conditions.

Figure 5. Verification Cases.

Table 1. Boundary Conditions for Verification Cases.

Case	**Initial Water ratio**	**Height**	**Top**[1]		**Bottom**	
			Flux	Potential	Flux	Potential
	$[V_w/V_s]$	[cm]	[cm/month]	[cm]	[cm/day]	[cm]
1	5.8	170	5	-10 000	--	Min(-202, psi_n)
2	5.8	170	5	-10 000	0	--
3	5.8	170	x	0.0	0	--
4	5.8[2]	250	5	-10 000	--	Min(334, psi_n)

[1] For the evaporation two boundary conditions are being used: a matric potential corresponding to the relative humidity and the maximum open water evaporation. The model works with the smallest of the two.
[2] A fresh layer of sludge(170 cm) has been modelled on an already ripened layer (80 cm). The water ratio of the already ripened layer has been determined according case 1. It is assumed that the 2nd layer has been deposed after a period of 100 days.

Case 1

In case 1 the sludge is deposed on a dry permeable ground surface. It has been assumed that the sludge is able to drain without any resistance at the bottom. The calculation results are presented in Figures 6, 7, 8 and 9.

The sludge dewaters in both upward and downward directions in the first time steps (see Figure 9), resulting in a decrease of the water ratio simultaneously at the top and the bottom of the layer. Finally the clay has a water ratio equal to 1,5 and a matric potential of -1000 kPa. After a period of 4 years the initial layer of 170 cm is compressed to about 60 cm and the flux converges to zero. The vertical axes of Figures 6, 8 and 9 is given in calculation layers of 1 cm solids. To transfer to vertical axes in cm these values should be multiplied by 1+e, which gives initially 170 cm but changes with every calculation step in a different way over the depth.

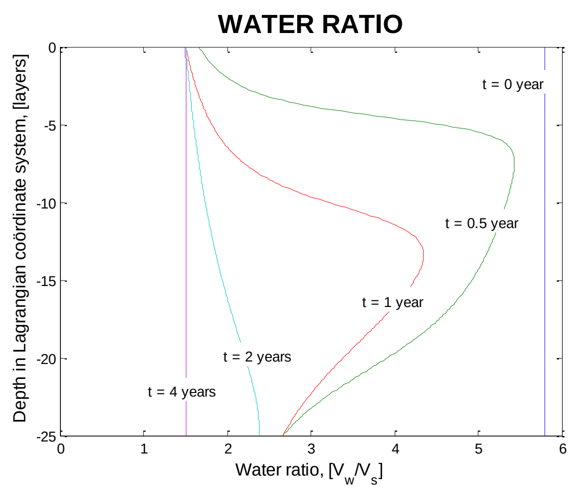


Figure 6. Water Ratio Over The Height Of The Sample.

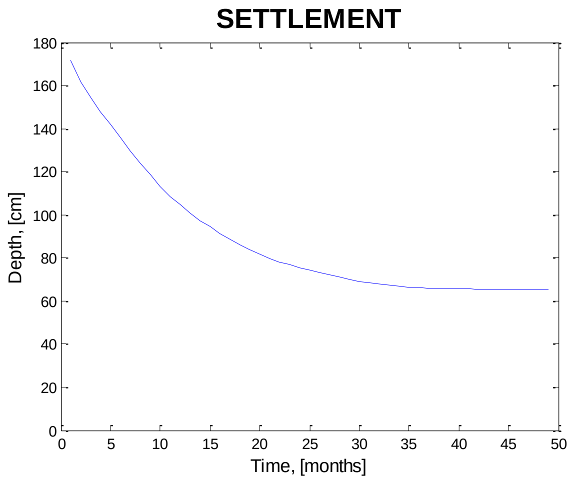


Figure 7. **Settlement In Time.**

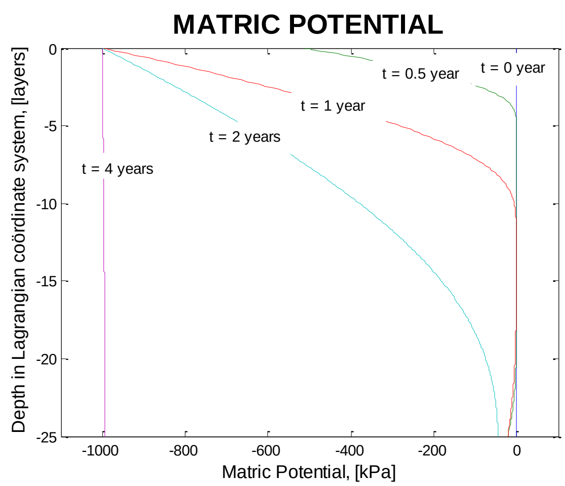


Figure 8. **Matric Potential Over The Height Of The Sample.**

Case 2

Sludge is deposed on an impermeable layer. There is evaporation at the top boundary and no flow over the bottom boundary. The height of the sample is 170 cm with an initial water ratio of 5,8. The calculated matric potential and fluxes are presented in Figures 10, 11 and 12, The settlements are nearly equal to Figures 7 as the effect of bottom drainage on the settlements is relatively small.

However the difference between an open and closed bottom can clearly be seen in water ratio, Figures 6. In case 2 water evaporates and simultaneously the sludge is consolidates under it's own weight but the water ratio does not decrease as fast as in the case with the drained bottom. The sludge dewaters only upward direction, Figure 12.

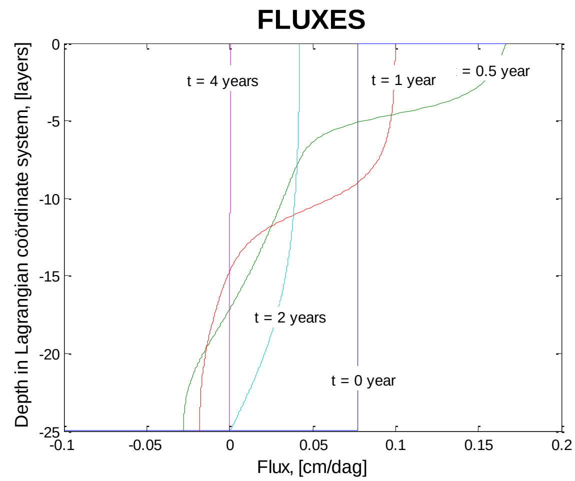


Figure 9. **Flux [Cm/Day] Over The Height Of The Sample.**

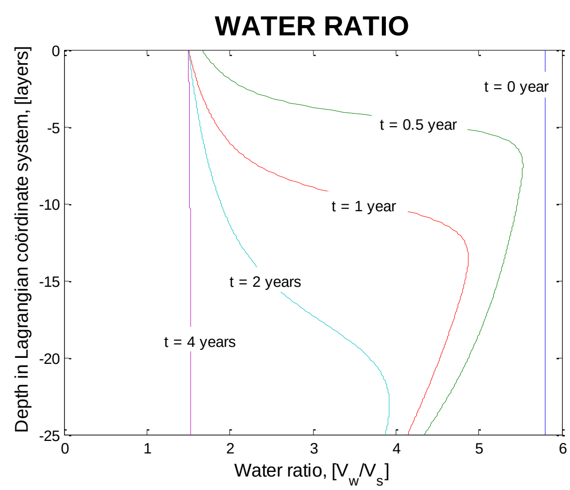


Figure 10. **Water Ratio Over The Height Of The Sample**.

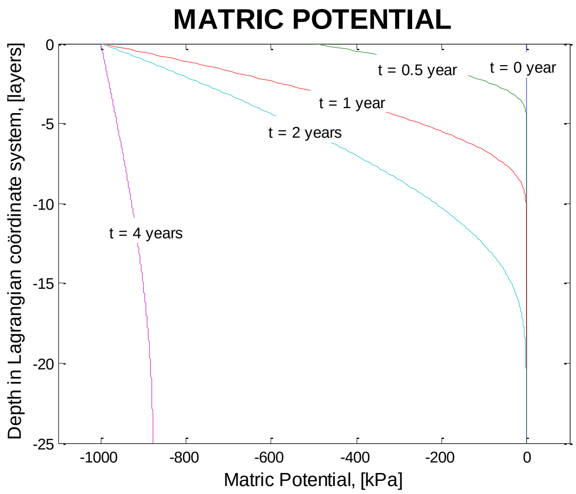


Figure 11. Matric Potential Over The Height Of The Sample.

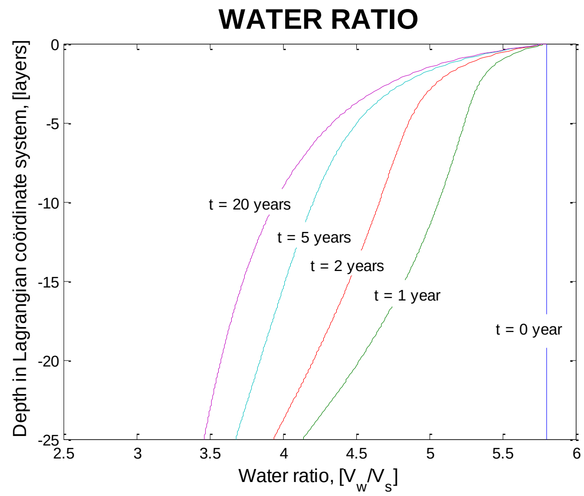


Figure13. Water Ratio Over The Height Of The Sample.

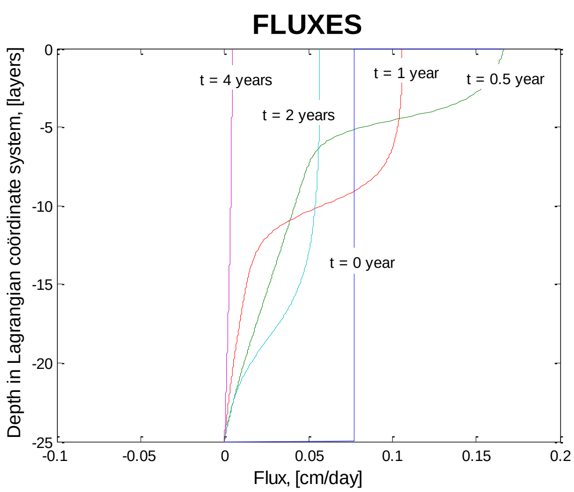


Figure 12. Flux Over The Height Of The Sample.

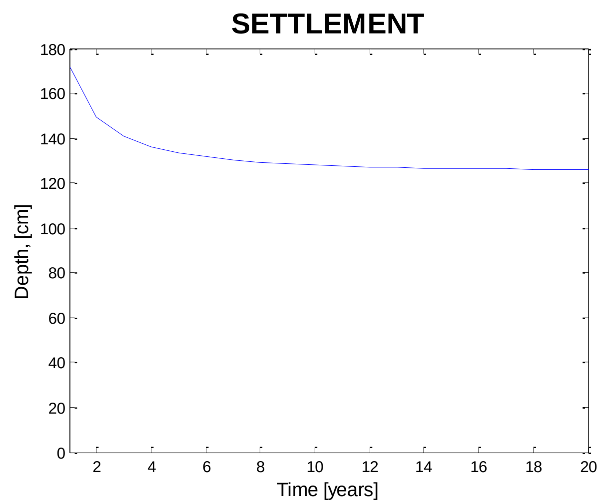


Figure 14. Settlement In Time.

Case 3

Case 3 is a situation of self-weight consolidation under water of a layer of 170 cm of slurry with the same initial water ratio of 5.8.

After a period of approximately 5 years (60 time steps) the flux is nearly equal to zero over the height of the sample and the end of settlement has practically been reached. The matric potential after 20 years equals app. 4 kPa at the bottom, which is equal to the effective stress. As γ_b= 12.21 kN/m^3 σ_v'= (12,21-10) 1,7 = 3.75 kPa

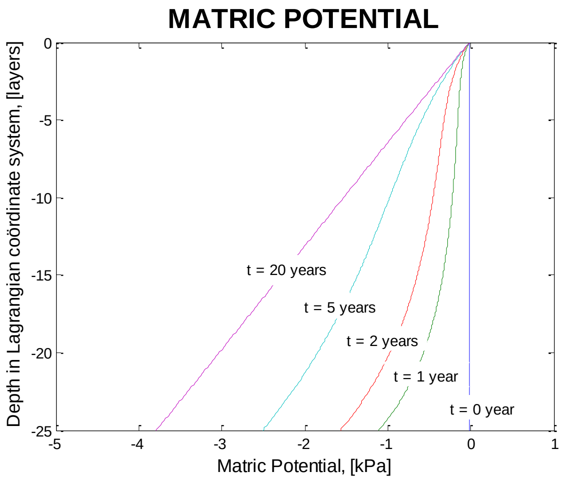


Figure 15. Matric Potential Over The Height Of The Sample.

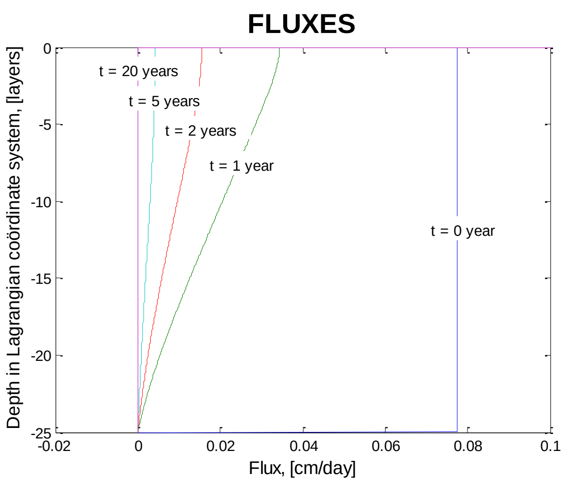


Figure 16. Flux Over The Height Of The Sample.

Case 4

Case 4 is represented by a situation where a fresh slurry layer is imposed on a drying layer (at t = 100 days). The top boundary is characterized by evaporation and the sample is able to drain at the bottom. The drying layer has been modelled according to case 1. The output has been generated at t = 100 days. Figures 17, 18 19 and 20 presents the results

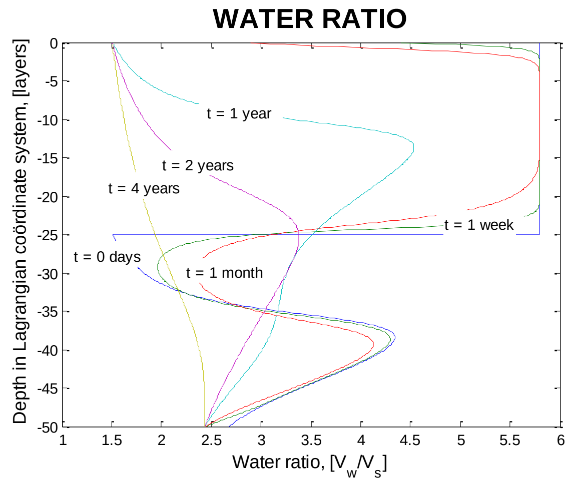


Figure 17. Water Ratio Over The Height Of The Sample.

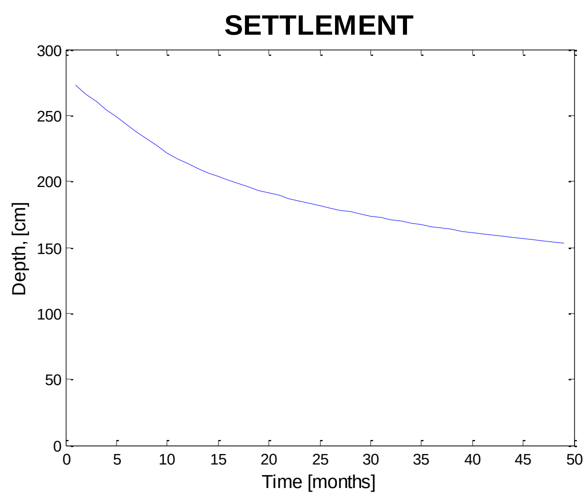


Figure 18. Settlement In Time.

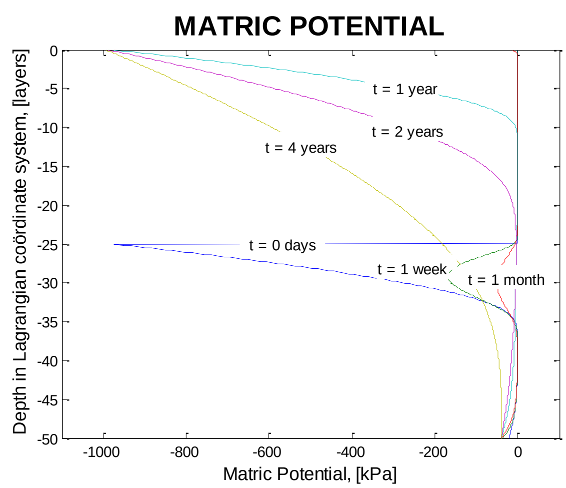


Figure 19. Matric Potential Over The Height Of The Sample.

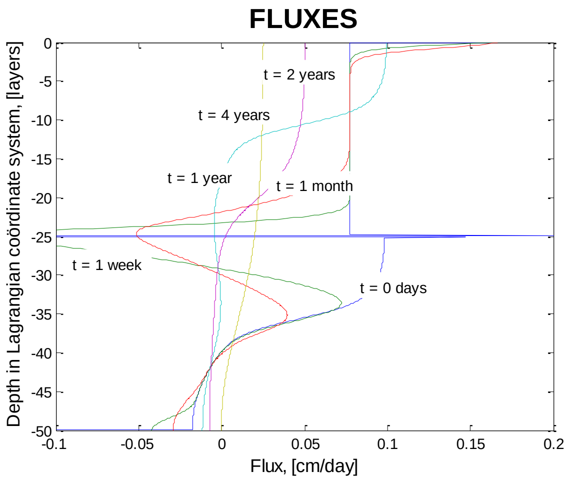


Figure 20. Flux Over The Height Of The Sample.

Figure 17 shows that the water ratio at the top of the bottom layer increases in the first time steps. The water from the imposed layer drains into the bottom layer. At the bottom of the lower layer the water ratio decreases; caused by the overburden of the imposed layer. Due to the high suction pressure in the top of the present layer, the evaporation becomes less. When the new slurry layer is being imposed, the matric potential at the top of the lower layer immediately becomes less negative due to the infiltration form the new layer.

VALIDATION

Besides verification, the model needed to be validated. Not all cases from the previous chapter have been physically modelled. Weijermars (2011) performed consolidation and sedimentation tests on thickened tailings from the Albian Sands Muskeg River Mine in Fort McMurray. In order to monitor the self-weight consolidation, a constant water head has been applied. The boundary condition are presented in table 2

Table 2. Boundary Conditions for Validation

Case	Water ratio	Height	Top[1)		Bottom		End
			Flux	Potential	Flux	Potential	
	[V_w/V_s]	[cm]	[cm/month]	[cm]	[cm/day]	[cm]	[days]
Val.	5.3	34	--	0.0	0	--	100

[1)] For the evaporation two boundary conditions are being used: a matric potential corresponding to the relative humidity and the maximum allowable open water evaporation. The model works with the smaller of the two.

Figures 21 up to 24 present the calculation results. The comparison of the settlements is shown in Figure 25.

The dots in Figure 25 represent the measured results and the lines represent the calculated results. The measured and calculated profile are very similar. The calculated settlement profile which makes use of the pF-curve and permeabilty curve come to a lesser degree of similarity than the fitted profile. The water ratio over the depth of the profile has been measured at the end of the test at t=100 days. In Figure 26 the measured and calculated waterratio at t=100 days over the depth of the sample are plotted

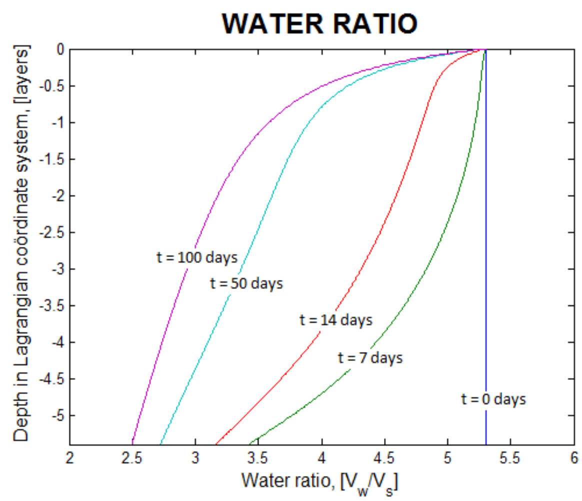


Figure 21. Water Ratio Over The Height Of The Sample.

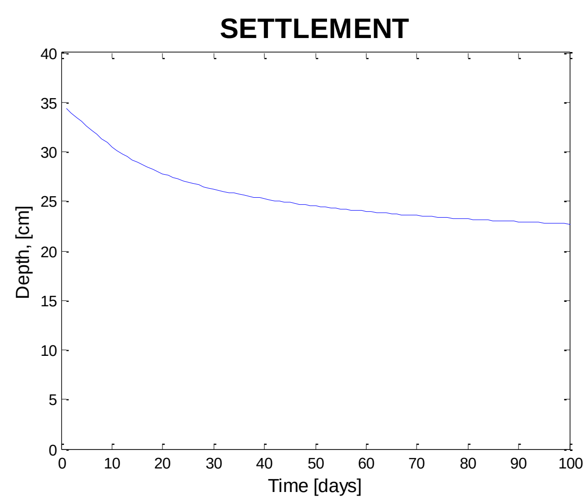


Figure 22. Settlement In Time

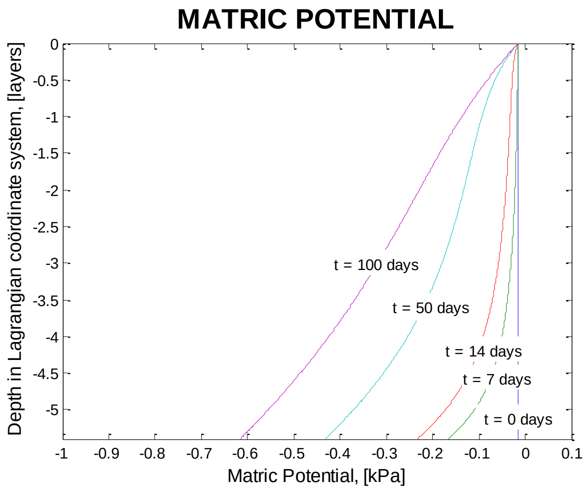


Figure 23. Matric Potential Over The Height Of The Sample.

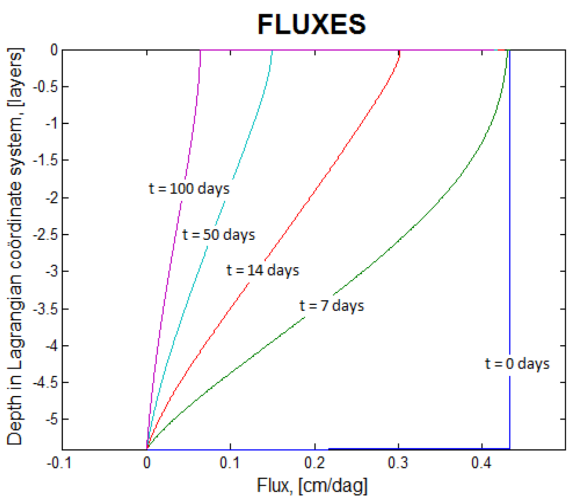


Figure 24. Flux Over The Height Of The Sample.

Both settlement profiles correspond reasonably well. The difference in both profiles can be explained by the fact that the measured profile height at t=100 was slightly less than the calculated profile height at t= 100. Therefore the measured water ratio is a somewhat less than the calculated water ratio. A possible explanation is that the chosen permeability for the calculation was too low, and also the relation was simplified (linear at log-scale). It can also be seen that the water ratio at the top and bottom of the sample correspond exactly, this means the water retention curve curve (pF-curve) was estimated quite well.

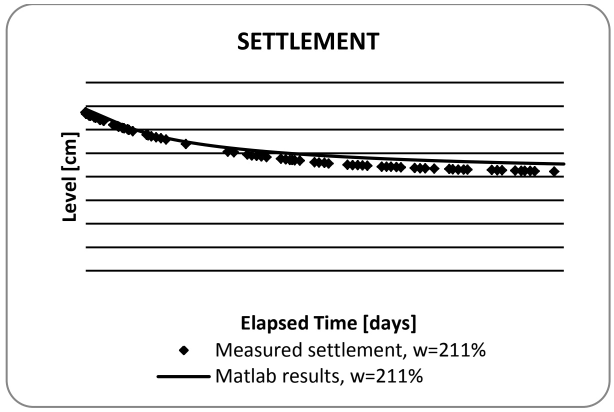


Figure 25. Calculated And Measured Settlement Of Under Water Consolidation Of Thickened Tailings.

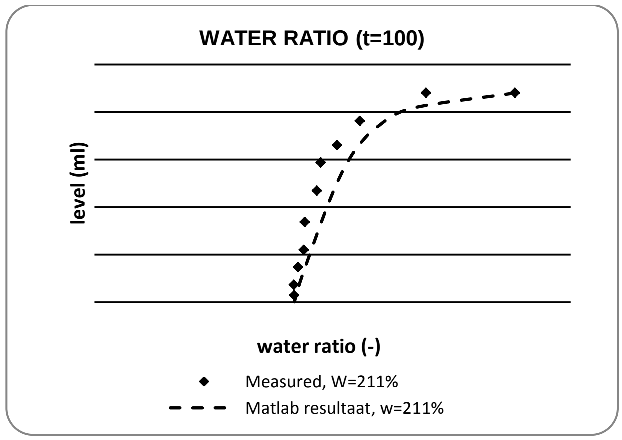


Figure 26. The Measured And Calculated Water Ratio.

CONCLUSION

A numerical model, which is able to simulate one-dimensional vertical flow due to consolidation and drying by evaporation of fine sediments, was successfully implemented in a MATLAB code.

The model calculates per time step the settlement, water ratio, the matrix potential and the flux over the depth of the sample. By a number of verification calculations it was shown that model gives consistent results. Validations using the results of laboratory tests could be simulated rather well.

During the development of the model some interesting findings were made: *i* the pF-curve can be interpreted in the same way as Terzaghi's stress-strain relation. For the fully saturated part the matric potential equals the effective stress.

ii This makes it possible to apply a matric potential at the bottom boundary of the sample, which equals the effective stress (case 1 and 3). This same applies for the upper boundary condition. If there is no load or evaporation, the matric potential at the top boundary can be set at zero. *iii* By the application of an overburden component, the pore water will flow upwards as a consequence of self-weight consolidation.

With the model it was possible to simulate successfully subsequent deposited layers of slurry which is promising for the prediction of the drying process in thin lifts. For the time being the model uses only one stress strain relation for drying and rewetting. This short coming can be avoided by using different retention curves for drying and rewetting.

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TOWARDS THE IMPROVEMENT IN GEOTECHNICAL PERFORMANCE OF ATMOSPHERIC FINES DRYING (AFD) DEPOSITS AT SHELL'S MUSKEG RIVER MINE

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ABSTRACT

Atmospheric Fines Drying (AFD) technology has been piloted at Shell Canada's Muskeg River Mine (MRM) near Fort McMurray, Alberta, since 2010 and is currently being implemented on a commercial scale. AFD technology, which involves the use of a chemical flocculent to bind fine particles in Mature Fines Tailings (MFT) stream followed by bed drying, is used within a suite of technologies to capture fines and form soft deposits from tailings streams to comply with Energy Resource Conservation Board Directive 074 (ERCB, 2009). Emergence of the AFD process stems from earlier success of thin-layer drying of MFT and research showing the potential benefits of flocculation in the process (Wells, 2011)

Two years of observation and monitoring of AFD operation at MRM reveals the inherent challenge in producing thin layer (<30 cm thick) deposits when flocculated MFT is produced to specification and placed on reasonable grade slopes (<3.5%) without the use of mechanical spreading. Thicker deposits (up to ~50 cm) were able to achieve target undrained shear strength of 5 kPa within a month. Progressively thicker lifts required significantly more time to dewater and reach target strength and had stratified shear strength profiles within the lift, with shallow depths below the surface "crust", having lower strength values compared to deeper within the profile.

The current paper details the improvement in understanding of the influence of cell geometry and slurry rheology on the flow behavior and uniformity of cell footprint coverage. In addition, an attempt at quantifying the mechanisms of water loss for the deposit is reported as well as the results from the evaluation of the geotechnical performance, namely shear strength gain, of the deposit.

INTRODUCTION

AFD technology implementation at Shell entails MFT harvesting from a barge located at the External Tailings Facility (ETF), slurry transport via pipeline to the drying beds, flocculent addition and mixing in-line in the discharge pipe to the bed, and material conditioning via submerged pipe outlet which results in a "rising bed of flocs". The general process stages are shown in Figure 1.

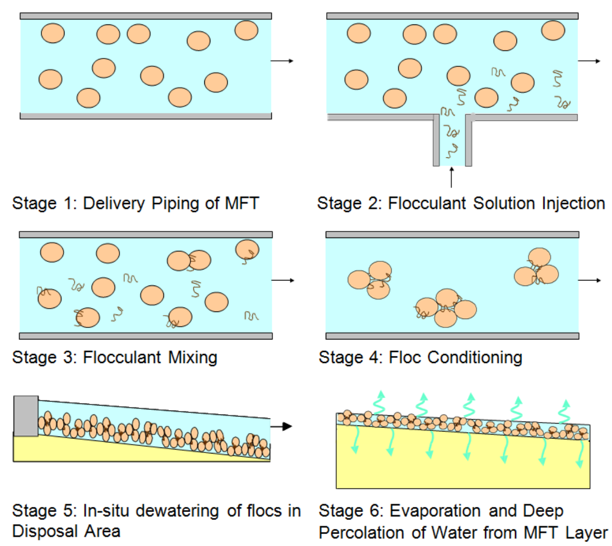

Figure 1. Stages of Shell's AFD technology.

The focus of the program reported in the current paper was to collect data in support of assessment of stages 5 and 6 of the process. The flocculation stages were developed through the work of House et al (2010). The deposited slurry typically contains 65 to 70% water (mining water content; weight of water over total weight). A significant amount of water is released in Step 5 as flocculated solids settle out and relatively clear water runs off. The deposit dries and densifies further through additional runoff, evaporation, and percolation to the drying bed's foundation. The current paper reports observations on how the cell geometry and discharge configurations affect the flow behavior, efficiencies of drying in Stages 5 and 6 of Figure 1, and the resulting impacts on shear strength gain of the deposits.

DEPOSITIONAL STRATEGY

The depositional strategy implemented included submerged pipe discharge accomplished via an earthen spill box or submergence under previously deposited material. Unrestricted pipe discharge (pipe laid flat on the cell base) was used in Phase 1 cells for the initial lift followed by submerged pipe discharge for subsequent lifts. Submerged discharge via earthen spill box was used for Phase 2 cells. Cell geometry and discharge piping details for Phases 1 and 2 are summarized in Table 1.

Table 1. Cell Geometry and Discharge Piping.

Cell type	Base slope (%)	Discharge location
Tall Trapezoid (Phase 1)	0.5 – 1.5	Cross-slope; side of cell
Rectangle (Phase 2)	2 – 3.5	Up-slope; top of cell

DEPOSIT FLOW BEHAVIOUR RESULTS

The deposit surface topography resulting from initial pours into a Phase 1 cell 2A and Phase 2 cell 8, constructed with 1% and 3% base slope, respectively, are shown in Figures 2 and 3. The product material rheology was relatively consistent for the two deposits, with average yield stress of 188 Pa and 216 Pa and slump height of 4.5 and 4.8 cm measured for cell 2A and cell 8, respectively. The pouring was from the locations depicted on the figures with black boxes and terminated when the material reached the downslope end (bottom) of the cell. The labels indicate the depth of material in meters. Surface topography was obtained from 3D scanning that was performed approximately 2 days following pour completion.

The deposit coverage was more uniform in areal extent and depth in the Phase 2 cell which had top slope discharge, higher cell base slope and earthen discharge box; all improvements implemented for Phase 2 based on observations from monitoring Phase 1 deposits in 2010. It was difficult to effectively utilize the far cross slope portion of the Phase 1 cells as shown in Figure 2.

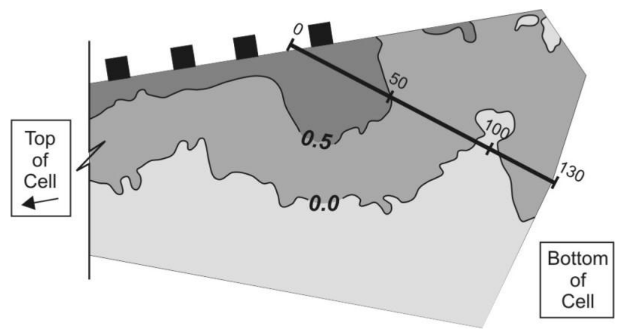

Figure 2. Phase 1 Cell 2A initial deposit coverage.

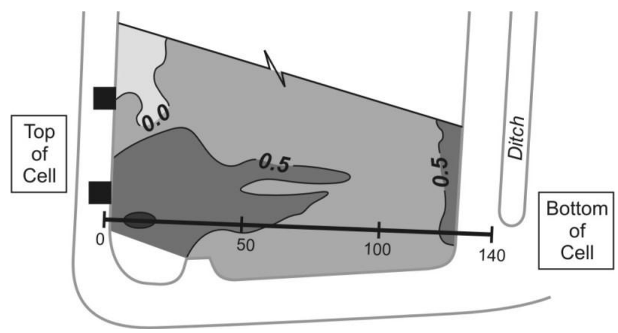

Figure 3. Phase 2 cell 8 initial deposit coverage.

The resulting slopes at the surface of the respective deposits are shown in Figures 4 and 5. The material "stacked" near the discharge points of the flatter sloped cell 2A and thinned at the downslope end. The resulting surface slope was over two times the base slope. Conversely, the material surface slope in cell 8 was relatively similar to the base slope resulting in a more uniform deposit depth; on average 0.5 meters thick.

AFD technology is premised on dewatering thin lifts of flocculated MFT in order to maximize evaporative drying. Common lift thickness targets are on the order of 30 centimeters (cm). Achieving uniform lifts as thin as 30 cm for flocculated material with high quality (high yield stress, low slump) at a commercial scale is very challenging. The balance between thin lifts and product quality considering cell geometry is illustrated in Figure 6, which shows the equilibrium flow depth of material that would be expected based on a yield stress of 200 Pa at varying bed slopes.

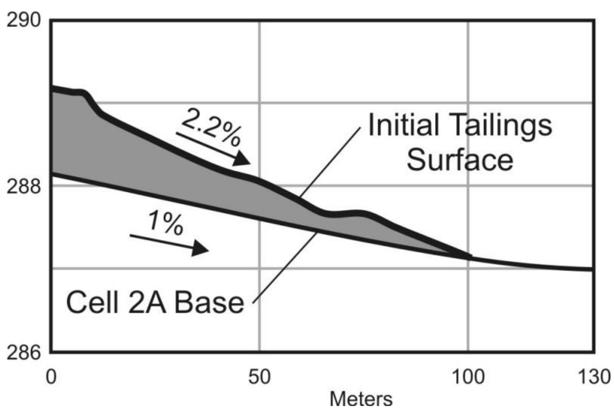

Figure 4. Cell 2A initial deposit profile.

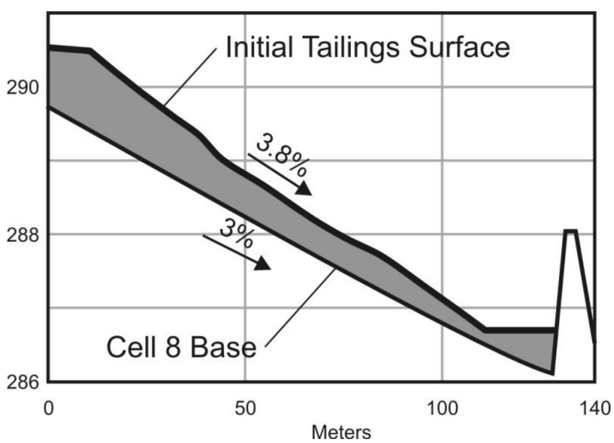

Figure 5. Cell 8 initial deposit profile.

The equilibrium flow depth (ho) is calculated as;

ho= τy/p g sin(a)

Where τy = yield stress, p = bulk density of the material, g = gravitational acceleration, and a = angle of the bed slope.

The vertical red arrows in Figure 6 depict the depth discrepancy between the target and the calculated depth for the Phase 1 and Phase 2 cells. As shown on Figure 6, achieving 0.3 meter lift thickness without compromising material quality (pouring low yield stress, high slump) requires very large slope beds (>5%). Large drying bed slopes are impractical from an economic standpoint. Hence, the results of the flow behavior of deposits at Shell's MRM site demonstrated that relatively uniform 0.5 meter thick lifts are achievable with reasonable bed slopes and high-quality AFD material.

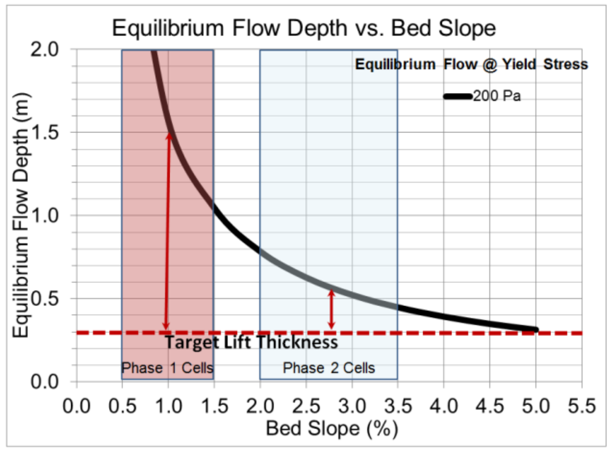

Figure 6. Slurry flow depth and bed slope.

DEPOSIT WATER LOSS

The mechanisms of water loss from the deposits were assessed using multiple methods as discussed below. Quantitative results are provided in ranges based on the individual cell results and method of assessment. Due to the large scale of the demonstration and limited number of monitoring points and thus, limited areal coverage of observation, the results should be considered indicative and not absolute. Smaller, more controlled, field demonstration cells were built and being poured in 2012 so that the deposit water loss mechanisms can be more accurately assessed.

Initial Water Release (~2 days post deposition)

Initial water release is the runoff that accumulated at the downslope end of the cell and discharged to the perimeter release water ditch. Quantification of initial water release included approaches that utilized operators deposition logs, *in situ* instrumentation, and material sampling. Operator's deposition logs included the volume of material poured into each cell, average density, and solids content of the feed MFT stream, and release water pumping rates from the ditch. Total Pressure Cells (TPCs) were installed at the base of the Phase 2 cells and measurements were used to estimate the height of tailings. Material sampling was performed ~2 days after deposition and analyzed for water content using the halogen moisture analysis (HMA) procedure.

Many of the pour events were for prolonged periods, with pouring spanning over 3 or more consecutive days. More "holistic" approaches were used to quantify performance for these extended pour regimes and more "instantaneous" or point measurement approaches were used to quantify performance of single day, end of deposition period, stacks.

Table 2. Initial Water Release Method Results.

Method for Overall Pour Performance	Water Content Reduction by HMA (%)
TPC-Total Pour	5 - 26
Deposition Log	6 - 10
Method for Single Day Pour Performance	Water Content Reduction by HMA (%)
TPC Peak Pressure	2 - 4
Material Sampling	14 – 35

An assessment of initial dewatering indicates that the water content decreased between 2% and 35% and results vary both spatially and temporally. For the overall deposit, the deposition log method was more consistent, but may underestimate the reduction since some release water was trapped at the toe of the cells and not accounted for. The larger variability in the TPC-Total Pour method for the overall deposit and for both of the short term methods is attributed to limits in areal coverage of the instrumentation and sampling, but offers insight into the variability of the deposit performance across a single cell or among different cells.

In general, the water content decrease attributable to initial dewatering is approximately 10% based on the method results.

Water content change (from the initial water release) compared with the product rheological parameters (yield stress and slump) for the samples are shown in Figures 7 and 8, respectively.

The spread of the data notwithstanding, the maximum observed initial dewatering occurs within the following ranges of the parameters:

- yield stress greater than 175
- slump less than 5.8 cm

The maximum water release observed related to yield stress here conforms to the yield stress range 3 (from ~350 Pa conditioned material to 125 Pa floc breakdown) (Wells 2011, see Figure 1) which the author suggested to be the "target design base for deposition".

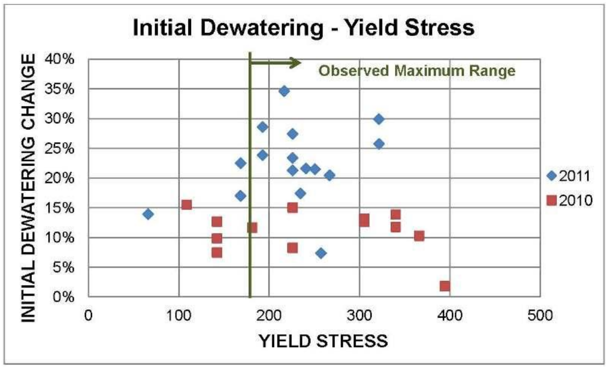

Figure 7. Initial dewatering sample results as a function of material yield stress.

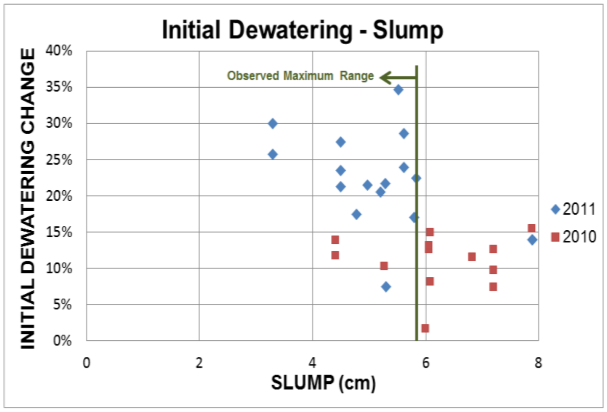

Figure 8. Initial dewatering sample results as a function of material slump.

Additional Drainage (>2 days post deposition)

Additional drainage, or delayed release water, is the water that was observed as water flowing in small surface channels 2 or more days after the end of deposition. The drainage is illustrated in Figure 9, which is a deposit surface 11 days after the end of deposition. The drainage is presumed to be water released during early stage of self-weight consolidation of the deposit.

The chart shown in Figure 10 displays the change in deposit water content with time for the same deposit shown in Figure 9. The water content reduction is interpreted from volume change determined from deposit scanning. The deposited material contained 68% initial water content and dewatered at a relatively constant rate for the first

10-14 days to ~47% water content. This resulting water content is approximately the saturation water content for the deposit, which was determined to be ~44% according to profile sampling conducted on day 8 after the end of deposition.

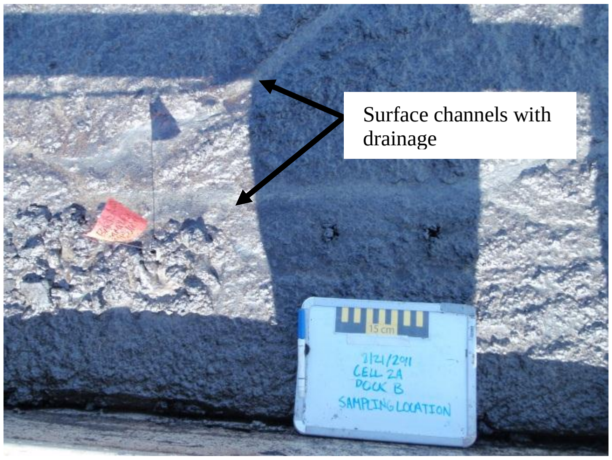

Figure 9. Deposit surface 11 days post deposition at Cell 2A Dock B.

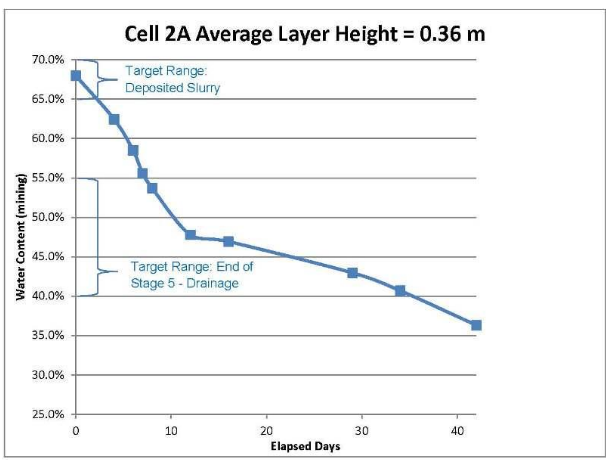

Figure 10. Deposit water content change with time estimated from 3D scanning.

CONCEPTUAL WATER BALANCE MODEL

The conceptual water balance model is presented as Figure 11 and includes initial dewatering to account for ~10% solids content increase based on the previously discussed analysis. The model also includes actual evaporation (AE) at a rate of 0.75 PE (potential evaporation), and percolation at 10 mm/month. The basis for the evaporation and percolation rates utilized for the water balance is explained in the following sections.

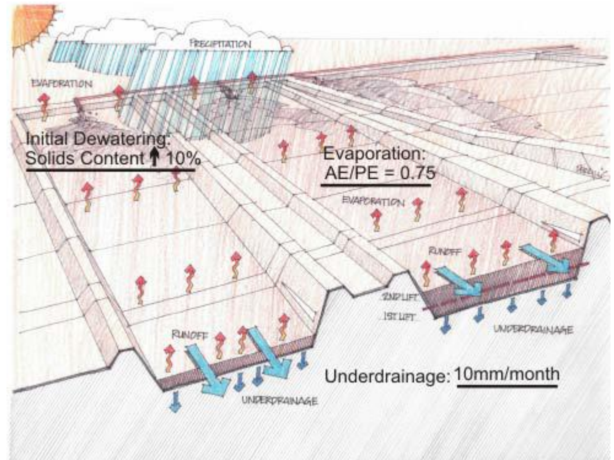

Figure 11. Conceptual model with updated dewatering parameters obtained from program.

Evaporation

Evaporation is the loss of water from the AFD deposit to the air. Evaporation from a tailings surface is a function of the vapor pressure gradient between the tailings surface and the ambient atmosphere. For saturated tailings deposit, evaporation rate is at the maximum (Potential) and solely dependent on climatic conditions. The potential evaporation in 2011 and longer term (over 100 years) for the site is shown in Figure12.

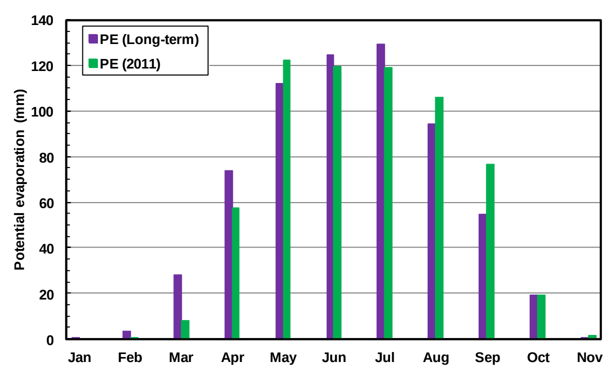

Figure 12. Potential evaporation (PE) in 2011 in comparison with long-term averages at the site.

Actual evaporation (AE) is influenced by the hydraulic properties of the tailings (i.e., tailings moisture retention curve and unsaturated hydraulic conductivity function) in addition to climatic parameters. AE was calculated using two approaches, namely; water balance with measured tailings settlement, precipitation (negligible percolation) and modified Wilson-Penman equation (Wilson, 1990) with measured temperature and pore-water pressure at the tailings surface. The PE, AE, and AE/PE ratio for the AFD deposits and monitoring period are shown in Table 3 below:

Table 3. Actual and Potential Evaporation Calculated in 2011 at the Site.

Month	PE in 2011 (mm)	AE in 2011 (mm)	AE/PE ratio
April	57.4	45.9	0.8[a]
May	122.6	85.8	0.7[a]
June	119.5	71.7	0.6[b]
July	118.8	71.3	0.6[b]
August	106.3	85.0	0.8[b]
September	76.6	61.3	0.8[b]
October	19	19	1.0[b]

[a] Data were estimated for monthly AE/PE ratios.

[b] Data were calculated from field measurements.

As shown in Figure 12, PE in May 2011 is slightly larger than PE obtained from the long-term climate data, while PE values in June and July 2011 are slightly smaller than the long-term PE values. This leads to the calculated AE in May 2011 being slightly larger than AE in June and July 2011 (Table 3) based on the designated AE/PE ratios. AE would be the largest in July based on the long-term PE and the same deposition conditions.

From an operational standpoint, tailings deposition thickness should be adjusted based on climatic variability in each month in order to maximize the potential for evaporative drying. Generally, thinner deposits in April, August, and September and thicker deposits in May through July can achieve the required undrained shear strength when PE is efficiently utilized.

Percolation to Foundation Soil/Desiccated Deposits

Percolation to drying bed foundation or previously deposited layers results when water can gravity drain from the fresh deposit into underlying material in response to hydraulic gradient. Two approaches were used to estimate percolation rate for the deposits; water balance method and Darcy's law. Field infiltrometer tests were conducted on the foundation material (lean oil sand) to determine the permeability of the soil and results ranged between 0.1 to 238 mm/hr with a geomean value of 7.2 mm/hr. The permeability of the older material layers were not measured, but was estimated to be approximately 3.6×10^{-2} mm/hr based on calculated void ratio (typically near 1 on average) and previous consolidation testing of flocculated MFT which yielded rates of 0.7×10^{-2} and 1.1×10^{-2} mm/hr at void ratios of 0.7 and 0.8, respectively. Piezometers were also installed throughout the tailings layers and foundation soils to determine hydraulic gradients.

Presented in Figure 13 is the estimated percolation through an older deposit in cell 3B. The initial spike in percolation rate occurred when the fresh deposit is placed and then decreases as the hydraulic head is reduced. The cycle is repeated at approximately 110 days when another fresh lift of material was placed.

In general, the range of percolation is estimated between 5 and 25 mm/month with an average of 10 mm/month. Percolation through older AFD deposits is expected to be on the lower end of the range due the lower permeability of the material; an important aspect when considering a multiple lift depositional drying scheme.

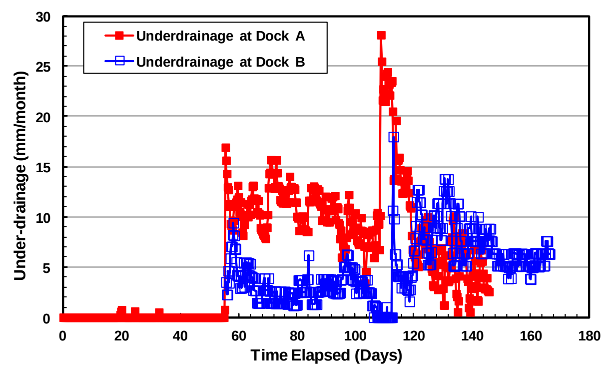

Figure 13. Percolation through underlying AFD deposits.

In the later stages of deposit drying, evaporation is dominant compared to percolation, accounting for about 4 times as much water loss as percolation. This ratio is based on estimated site AE of 480 mm/year (75% of PE at about 640 mm/year) and percolation of 120 mm/yr.

DEPOSIT STRENGTH GAIN RESULTS

The deposit begins gaining strength shortly after deposition. The initial stages of strength gain are attributable to the flocculation mechanism (causing the aggregation of fines particles) as the deposit transitions from slurry phase to a more “soil-like” phase. This deduction is based on the fact that the conventional mechanisms of shear strength gain (e.g. settlement, consolidation and desiccation) would be expected to be negligible shortly after deposition. Subsequent densification and strength gain of the deposits is attributable to a combination of settlement, desiccation and consolidation, in addition to flocculation.

Figure 14 below depicts one of the thicker deposits poured in 2011 which was just under a meter at the monitoring location. The plot shows time series of *in situ* peak shear strength measured by hand vane at different profile depths throughout the deposit. The deposit has an initial strength of approximately 1 to 3 kPa and then gains strength with time. The strength gain is stratified; with material at depths (near cell foundation) gaining strength more rapidly compared to material closer to the deposit surface.

The measurements close to the surface samples were taken below the “crust” which was the top desiccated portion of the deposit (shown in Figure 15) that was several to tens of centimeters in thickness.

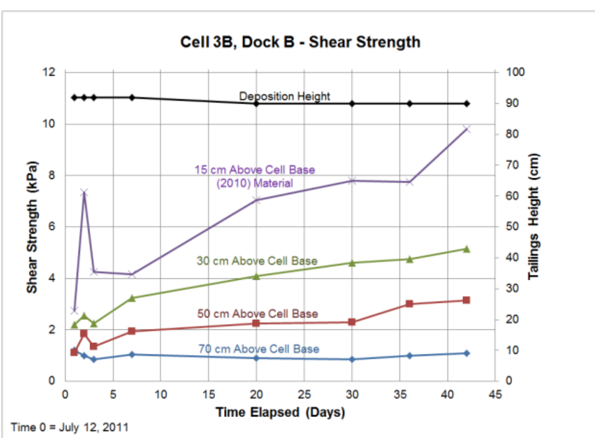

Figure 14. Shear strength gain over time at different profile depths for a 90 cm thick deposit.

The stratified strength profiles are consistent with the density profiles presented in Figure 16. Material densities were determined by portioning sections of the material retrieved using a piston-core sampler. The density profiles also depict a trend of increasing values with time consistent with the strength plots previously shown in Figure 14.

Figure 15. Deposit surface at 30 days at Cell 3B Dock B.

The time taken for the deposits monitored to achieve the target peak undrained shear strength of 5 kPa is shown in Figure 17 as function of deposit thickness. The plot includes deposits ranging in initial thickness of between 25 and 85 cm; layers over 90 cm either did not reach strength target within the program duration or access for testing was lost. In general, the data trend is logarithmic with a “knee” in the curve at about 60 cm deposit thickness and 30 days of age, followed by a flattening of the curve from 60 to about 80 cm deposit thickness from 30 to 90 days.

It is important to note that the data in Figure 17 do not distinguish between variables such as material quality, frequency of pours, cell slope, subgrade material, and climatic conditions. The potentially biased measurements are for layers that covered residual 2010 AFD material left near the monitoring point and were less than the thickness of the residual material, thus imposing a potential boundary condition issue. The potential outlier pocket of data below 45 cm layer thickness as identified in the figure are made up of material that was off specification (<200 Pa yield stress) or data was not available.

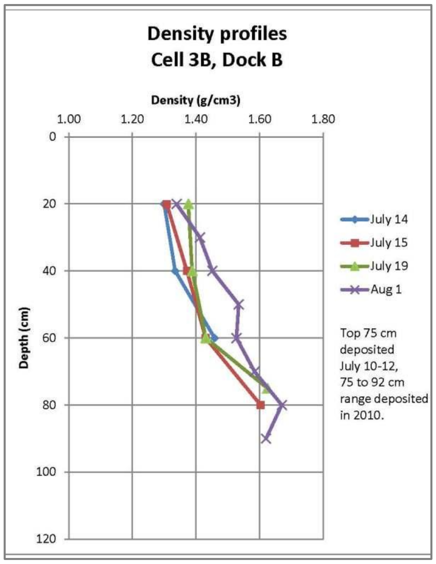

Figure 16. Density profiles of the 90cm-thick AFD deposit at several times over a 17-day period after deposition.

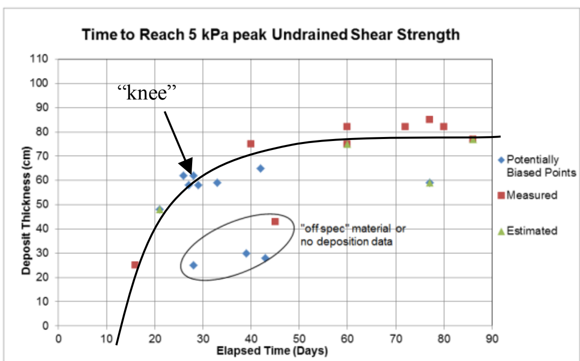

Figure 17. Time for deposits with different lift thickness to reach 5 kPa peak undrained shear strength target.

From an operational standpoint, this type of relationship can help guide deposition schedules based on available footprint. For example, if the cell capacity is used up in 30 days of pouring at planned rates, then targeting lift thickness of 60 cm or less provides opportunity to meet strength targets prior to subsequent deposition events.

DEPOSIT SHEAR STRENGTH GAIN AND WATER CONTENT RELATIONSHIP

The plot in Figure 18 depicts the profiles of water content within the same deposit as previously presented in Figures 14 and 16. Relatively lower water contents were recorded at depths within the deposit compared to shallow depths, consistent with higher densification (lower degrees of saturation) at depths.

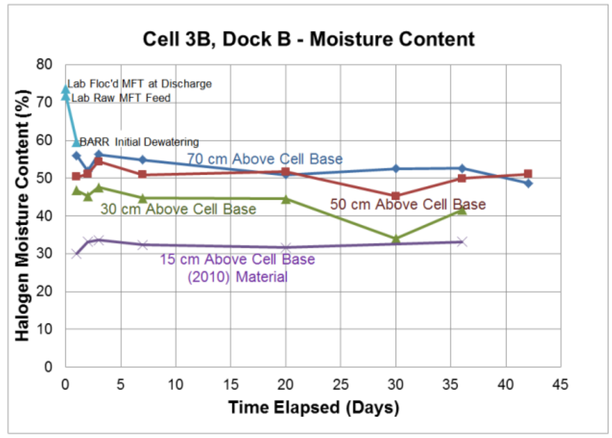

Figure 18. 90 cm thick deposit water content loss performance at several height intervals.

Figure 19 displays all of the 2010 and 2011 AFD material data with corollary strength and water content measurements (471 data points). Similar to Figure 17, the data do not distinguish between variables such as frequency of deposition events, cell slope, subgrade material, climatic conditions, and post deposition treatment.

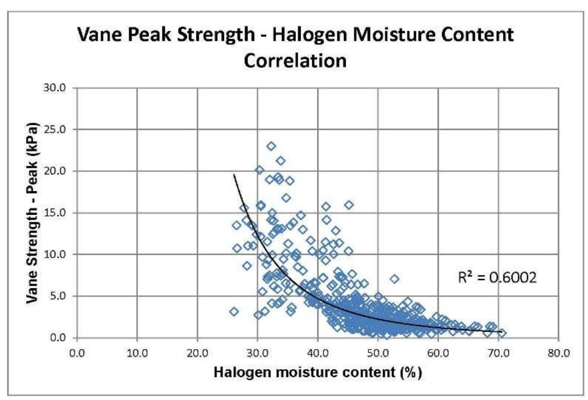

Figure 19. Strength and water content data correlation for all monitored deposits.

The regression line shows relatively poor coefficient of determination (R^2) suggesting the relationship is not appropriate for prediction of strength solely on water content. However, a trend can be seen to exist between shear strength and water content with the presented relationship being similar to those of other tailings, such as those depicted in Dimitrova (2011).

Plotting of strength versus liquidity index (Figure 20), a common approach for conventional soils in geotechnical application, shows relatively similar scatter for the AFD material.

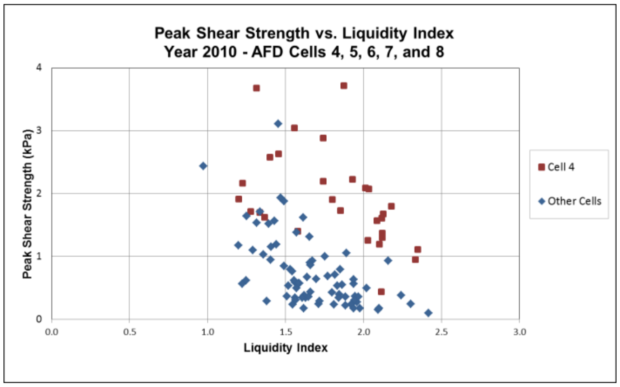

Figure 20. Strength and liquidity index correlation.

A general observation regarding the peak strength points in Figure 20 is that they are located higher up in the plot than conventional soils and MFT (Masala, 2010), especially the Cell 4 data set. The shear strengths for the LI>1 range are unusually high, which indicates that the major contribution to shear strength comes from the flocculated structure and not from density as in typical soils (non-cemented). Detailed investigation of the mechanisms of shear strength contribution from flocculation during the AFD process is outstanding and only general knowledge can be relied upon for interpretation of the current data at this time.

CONCLUSION

The understanding of water loss mechanisms and their relative contribution to deposit dewatering and shear strength gain is important for improving the design and operation of the AFD technology at the commercial-scale. Various elements of deposition (e.g. cell geometry, base slope, discharge point arrangement, frequency and length of deposition) as well as the rheology of the flocculated MFT all contribute to deposit's footprint coverage, uniformity of thickness and geotechnical performance.

A conceptual model for dewatering of flocculated MFT was presented based on two years of data and observation. Three distinctive stages of dewatering for the AFD process are identified as: the initial water release due to flocculation; additional drainage or delayed dewatering due to self consolidation (water lost from the tailings surface to runoff or evaporation); and evaporative drying. Deposits' surface slopes, runoff water management, climatic conditions, and underlying material's boundary conditions all contribute to dewatering rates for the various mechanisms.

The interplay of so many parameters that affect the resulting deposit thickness and dewatering rates makes the prediction of the performance of the AFD technology to a high degree of certainty challenging. However, in general, thin (< ~30 cm) lifts dewater faster and need less time to meet shear strength targets. Thicker deposits exhibit stratification and slower shear strength gain compared to shallower lifts. Song et al. (2011) estimated the total deposit thickness that can be evaporatively dried using thin lifts on a yearly basis to be approximately 144 cm with 80% probability of exceedance. Thicker lifts show promise as a potential variant depositional strategy leveraging self-weight consolidation. However, multiple lifts may be necessary to further consolidate the upper portions of single thicker lifts that may not reach 5 kPa undrained shear strength on their own within a year.

Future technology development work aimed at optimizing the deposition and management of flocculated MFT is warranted in order to improve the geotechnical performance of resulting deposits. A multi-year technical program aimed at optimizing the MFT tonnage processed per unit footprint to target undrained shear strength using the AFD technology is currently underway at Shell's MRM site.

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BENCH-SCALE DRYING OF MULTI-LAYERED THICKENED TSRU TAILINGS

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ABSTRACT

Atmospheric drying technologies as practiced by Shell and Suncor are promising options to manage oil sands fine tailings. While single-layer tailings deposits have been shown to dry and gain strength effectively, there are questions as to how the strength and water content of the dried layers will change when successive fresh wet tailings slurries are deposited.

A bench-scale testing program was conducted at Shell Canada Energy's Muskeg River Mine in northern Alberta to study multi-lift drying of thickened paraffinic froth treatment or tailings solvent recovery unit (TSRU) tailings. The tests were conducted under controlled conditions in boxes with varied drainage conditions to evaluate changes in geotechnical parameters as additional lifts were placed and dried. The tailings were placed in lifts of 50 cm. Instruments were deployed in each lift to monitor soil matric suction, water content and pore-water pressure. Additional analysis included vane shear tests and hand-held tensiometer measurements.

This study found that after placement of a second lift, the first lift lost little strength, and quickly recovered to exceed the prior strength. Soil matric suction returned to its prior values, as did water content, although the drier upper portion of a lift may not return to its prior minimum water content. Second and subsequent lifts gained matric suction and strength after placement more quickly than had the first lift. These findings apply to thickened TSRU tailings, and may have been strongly influenced by partial decant water removal prior to lift placement.

INTRODUCTION

A number of oil sands operators are considering thin-lift drying or variants on that approach as a management strategy for fine tailings. This technology is now being used at a commercial scale by Shell, and has been used for several years by Suncor. There are challenges in treating significant volumes of tailings using this technology. These include: devoting sufficient land area for thin-lift drying; installing the necessary tailings transfer, treatment, and deployment infrastructure; and finding the balance in layout and operational practices for efficient cost-effective tailings placement, drying, and removal or reclamation. Economic thin-lift fines drying likely requires placement of tailings in multi-lifts to optimize the value of the infrastructure and footprint needed for this technology and to minimize material handling.

Placement of new lifts redistributes water to underlying lifts and at the same time substantially cuts off evaporation from the covered lift. Understanding drying and strength gain of just-placed lifts, and wetting and strength loss in underlying lifts, is critical to developing an effective drying program. Multi-lift placement will not be viable if the addition of new lifts compromises the strength of underlying lifts, the ability to operate equipment as needed in the thin-lift drying area, or the ability to reclaim the deposit in a reasonable time frame.

A group of oil sands companies is exploring the possibility of managing their TSRU tailings using thin-lift drying. As part of their 2011 pilot program to test thickening of TSRU tailings and deposition of the thickened material, a companion program to monitor the effects of placing multi-lifts of thickened TSRU tailings in laboratory columns was developed. The companies sponsoring this work were Shell Canada Energy (Shell), Imperial Oil Resources Limited (Imperial), Total E&P Canada Ltd. (Total), and Teck Resources Limited (Teck). The work was performed by Barr Engineering Co. (Barr) and O'Kane Consultants Inc. (O'Kane). This paper presents a brief summary of selected data and key findings of the multi-lift, bench-scale testing program conducted at Shell's Muskeg River Mine (MRM) in northern Alberta. The field program, also carried out by the same operating companies, is described in a separate paper: TSRU Tailings Pilot-Scale Deposition Trials in 2011.

MULTI-LIFT TEST PROGRAM

To understand better how sequential layers of tailings might behave, a bench-scale test program was carried out. The program was designed based on placing 50 cm lifts of tailings, using in situ monitoring, and drying under controlled conditions. The tests used large Plexiglas boxes so that lifts could be observed during and after placement and could be accessed for additional monitoring using hand-held instruments. Figure 1 provides the conceptual layout of the test box. A lift of tailings was placed in the box and dried for a period, after which the next lift was placed. In order to accelerate drying of the tailings, air was continually circulated into the box after each lift was placed.

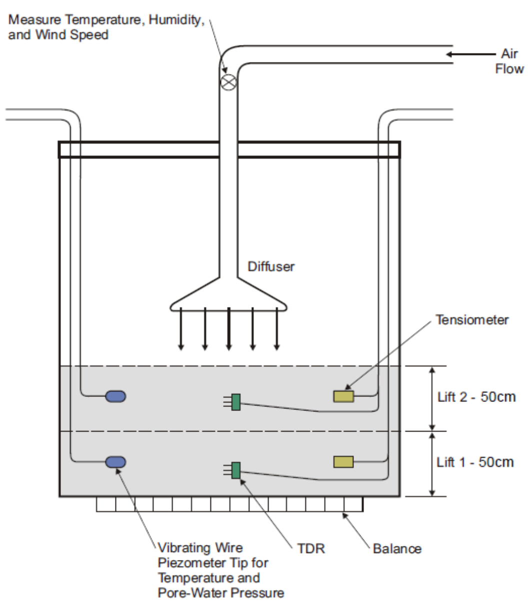

Figure 1. Test Box Layout Concept.

The program included four boxes:

- A box with two lifts of TSRU tailings and no under-drainage;
- A box with two lifts of TSRU tailings and under-drainage at the base of the box;
- A box with three lifts of TSRU tailings and drainage below each lift; and
- A control box with just water, to measure potential evaporation (PE).

The results from the box with two lifts and under-drainage are reported here to illustrate the results of this multi-lift experiment. Where significant observations from the other boxes help to provide further understanding, they are also mentioned. Deposition under sub-zero temperatures was not studied.

Tailings Supply

In the TSRU field program a slipstream of raw TSRU tailings from the Shell MRM TSRU tailings discharge pipeline were pumped to a one-meter-diameter Outotec thickener, flocculated using approximately 100 grams/tonne SNF A3338, and thickened to 42-48% solids by weight (mineral plus bitumen).

Thickened tailings were produced in late October, but the bench drying test program was not authorized until mid-November, so twelve drums of the thickened tailings at about 42% solids were stored for this experiment until the test systems could be set up. The tailings were stored in drums for seven weeks prior to placement of the first lift, and nine weeks for the second lift. To avoid breaking the flocculated structure, the material in the drums was not mixed during storage. Rather, the tailings were allowed to settle, and the accumulated water was decanted just prior to placing the tailings in the boxes.

The TSRU tailings produced by the pilot-scale thickener had a well-graded particle size distribution as illustrated in Figure 2, with a sand-to-fines ratio (44 μm) of around 1.

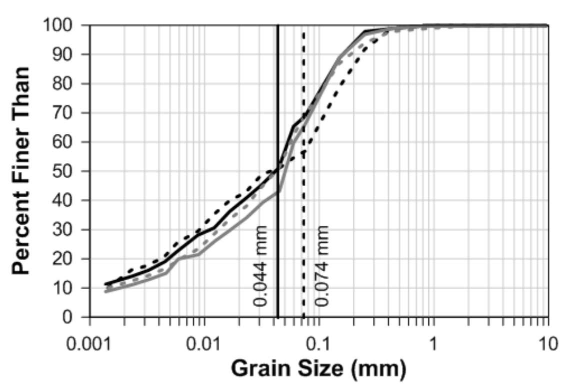

Figure 2. Particle Size Distributions for Four Thickened TSRU Samples from the Field Program Deposits.

Test Box and Instrumentation Layout

Figure 3 shows an assembled test box. The test box was 0.75 m wide, 0.75 m long and 1.3 m high. A removable hood containing the air drying system rested on top of the box. Air flow was controlled with a valve in the ducting and was distributed into the test box using a diffuser. The position of the diffuser was adjustable to maintain a uniform height from the tailings surface upon placement of each lift. Temperature and relative humidity were also measured during the drying process. There was no attempt to simulate the effects of solar radiation.

Figure 3. Test box assembly.

Monitoring Parameters

The program was designed to measure: (1) water redistribution; (2) strength gain or loss; and (3) settling or consolidation. The box was monitored for the following parameters:

- pore-water pressure and temperature using vibrating wire push-in piezometers (VWP);
- water content using time-domain reflectometry (TDR) ;
- matric suction (negative pore pressure) using tensiometers; and
- quantity of water leaving the system using a scale that continuously measured the weight of the box.

Rather than a single set of instruments per lift as suggested by Figure 1, instruments were placed near the upper and lower boundaries as well as the near middle of each lift. This provided a vertical profile within the lift and also illuminated changes across lift boundaries.

Non-continuous measurements were collected using hand-held instruments and small samples taken from the lifts. Shear strength was measured by vane shear, soil matric suction was measured with a hand-held tensiometer, and moisture content was measured from samples analyzed on-site in a halogen moisture analyzer. The holes left by sampling were re-filled with tailings. Every two to three days, the box lid was removed for several hours so that strength gain, soil matric suction, and moisture content could be measured. Strength gain was an important factor in deciding when to place the next lift. The objective was to reach a minimum peak shear strength of 20 kPa before placement of a subsequent lift.

After completion of the drying program, the box was taken apart. Samples were collected to profile the deposit. The deconstruction sampling included oil/water/solids analysis, density measurement, and soil matric suction.

Lift Placement

The thickened TSRU tailings were scooped from the drums and carefully placed in the test box in a lift of 0.5 m over a 16 cm sand drainage layer in the base of the box. The sand had been saturated and allowed to drain before placement of the first lift. After 2 weeks of drying, the second lift was placed to a 0.5 m height above the first lift, and the added mass was recorded. After a lift was placed, expressed water that accumulated at the surface was drawn off and weighed. This was considered to simulate runoff conditions. Removal of the expressed water also saved the time needed to evaporate the water, which accelerated the testing. Similarly, water that percolated to the drainage layer was drawn off using a valved tube at the bottom of the box and weighed. This was considered to represent percolation or drainage to the underlying soils. Actual evaporation (AE) was calculated by deducting drainage and runoff from the total box weight loss.

Care was taken in placing the lifts to minimize stirring or mixing that might break down the flocculated structure of the thickened tailings. This preserved the soil structure and permeability of the settled tailings from the drums.

TEST RESULTS

Evaporation

Evaporation from slurries such as these saturated tailings characteristically proceeds in stages, illustrated in Figure 4. Stage 1 drying is the period when AE and PE rates are approximately equal. During Stage 2, a dry surface zone develops that reduces water transfer and the rate of evaporation. (Wilson et al., 1994; Simms et al., 2009; Cargill, 1985). When Stage 2 drying occurs, AE becomes a progressively lower proportion of PE.

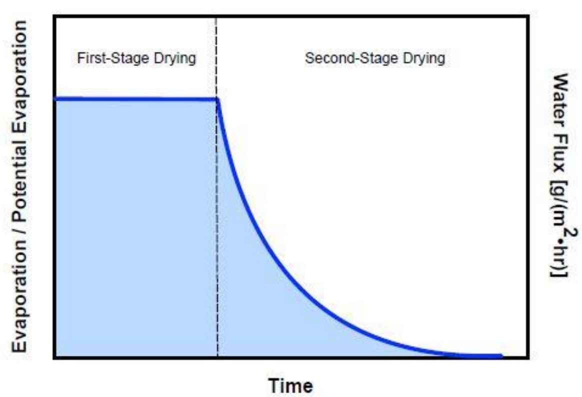

Figure 4. Stages of Drying.

One test box containing water only was used to measure PE. The average wind speed was 5.7 m/s, producing typically 6 mm/day PE. The PE rate was expected to be the same in all boxes because the air supply was tuned to provide uniform wind speed in all boxes. In fact, there was some variation in PE because some areas of the room were warmer than others, due to the somewhat localized heating system used. In addition, details of the air exhaust from the boxes were different between the TSRU tailings boxes and the water box. In consequence, some tailings AE rates exceeded the PE measured for the water-filled box.

This effect is seen in Figure 5, which shows that the AE measured in the two-lift drained tailings box exceeded the PE measured in the water box for Lift 1.

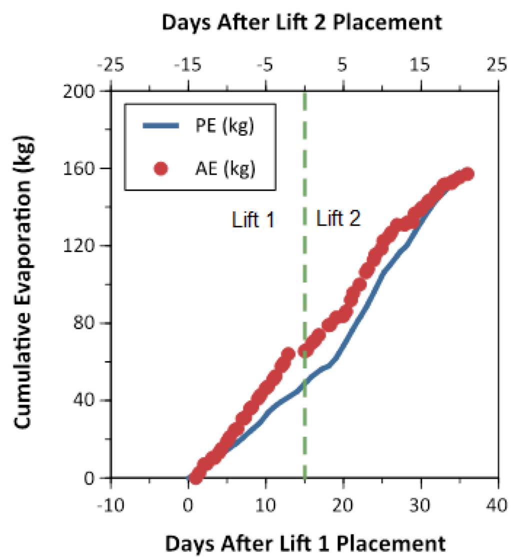

Figure 5. Potential and Actual Evaporation: Lifts 1 and 2.

Adjustments to better match exhaust systems between tailings boxes and the water box were made for Lift 2, with the result that PE was equal to or greater than AE. It can be seen in Figure 6 that for Lift 2, about 12 days after placement, the slope of the AE line is less than the PE slope, which indicates that the lift transitioned to Stage 2 drying at that time.

Total Water Loss

Figure 7 presents the water loss and tailings settlement, but does not account for the water decanted before lift placement. For the material placed in the test box, the runoff and drainage were about equal, and plot on top of each other in Figure 7. Runoff and drainage appear to be relatively quick, ending within 3 days in this experiment. In the first lift, the evaporation loss was 119 mm which accounted for around 74% of the post-deposition water loss.

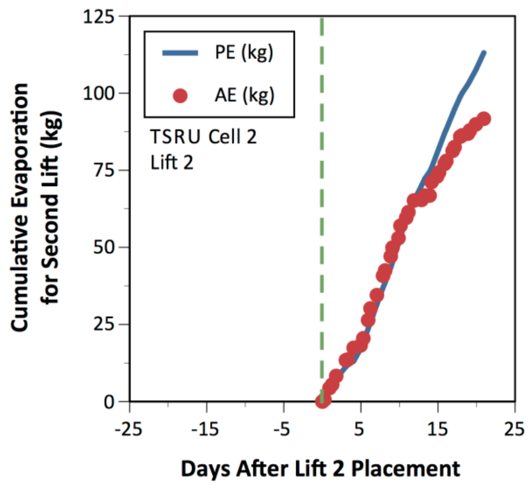

Figure 6. Potential and Actual Evaporation: Lift 2.

Unlike Lift 1, no runoff (supernatant) water was observed from Lift 2. The addition of the second lift re-wetted the underlying first lift but did not saturate it or result in drainage through the underlying layer.

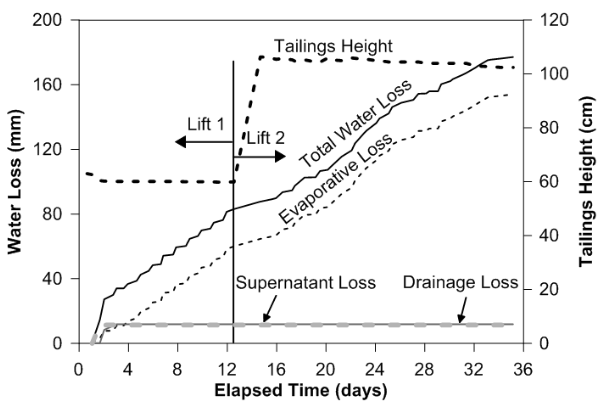

Figure 7. Cumulative Water Loss, Tailings Height.

Water loss, including water decanted from the drums, is summarized in Table 1.

A significant amount of water (a little over 50 percent of total water loss) was removed from the tailings prior to placement in the box. Had this water not been decanted, it would likely have appeared as either runoff or drainage. When the decanted water volume is combined with the runoff and drainage measured for Lift 1, runoff and drainage constitute about two-thirds of the total water loss. Evaporation, though it was about three-quarters of the water loss measured in the box for Lift 1, accounted for only about one-third of the total water loss for these TSRU tailings. However, if the water had not been decanted prior to placing the lifts, Stage 2 evaporation may have been prolonged, thereby increasing the contribution from evaporation. The proportions for Lift 2 run a little more in favor of evaporation, but runoff and seepage/redistribution are still about 60 percent of the Lift 2 water loss.

Table 1. Water Loss Summary.

	Lift 1		Lift 2	
	(mm)	(%)	(mm)	(%)
Decant Water Loss in Drums	199	55	228	53
Water Loss in Test Box				
a. Evaporation Loss	119	33	170	39
b. Runoff	20	6	-	-
c. Drainage	22	6	-	-
d. Redistributed to underlying layer	-	-	36	8
Total Water Loss	360	100	434	100

Most of the TSRU tailings settlement took place in the drums during storage. The tailings settled to about 75 percent of their original volume, the balance being decanted as clear water. Settlement of the first lift after placement in the box was 3 cm (from an initial Lift 1 height of 47 cm). Lift 2 settled by 4 cm from an initial height of 46 cm. Had fresh tailings been placed in the box, total settlement might have been close to the sum of that observed in the drums and the box, or about 30 percent.

Water Content

The TDRs continuously monitored the in situ volumetric water content, as shown in Figure 8. The water content data show the expected pattern: the top of Lift 1 became drier and began drying earlier than the bottom. Lift 2 showed the same general pattern, but the period of initial rapid drying occurred earlier for Lift 2 than for Lift 1. Addition of the second lift increased the water content of the first lift, most strongly affecting the relatively dry top portion of Lift 1.

Lift 1 then resumed losing water, and from day 28 on (from 14 days after Lift 2 placement) the top of Lift 1 had a water content practically indistinguishable from the bottom of Lift 2. The bottom of Lift 1 gained a small amount of water from placement of Lift 2, yet by day 28 (14 days after Lift 2 placement) became drier than before placing Lift 2. This demonstrates that the wetting effect of the added lift is temporary. It appears that a lower lift can, with time, recover satisfactorily, although the drier upper portion of a lift may not return to its prior minimum water content. Testing with fresh tailings would provide greater confidence in this conclusion. Previous laboratory and numerical modeling work with thickened gold tailings seem to support this preliminary conclusion (Dunmola, 2012).

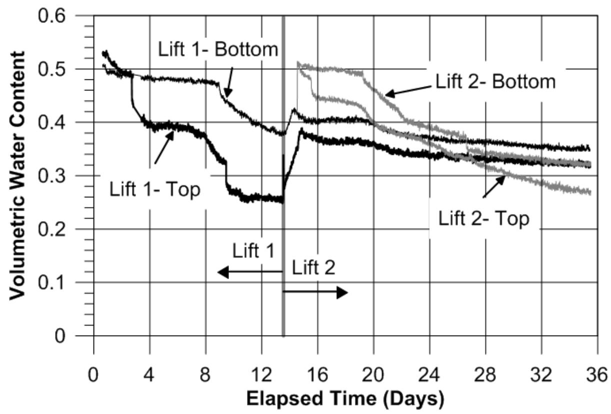

Figure 8. Water Content by TDR.

An important factor in this experiment was the dryness and matric suction of the underlying lift, giving it the capacity to accept some water from above. Table 1 provides an estimate of the downward deportment of water from Lift 2 to Lift 1 based on the change in water content recorded by TDR. The 36 mm deportment is of comparable magnitude to the runoff plus drainage from Lift 1. This amount of downward water deportment did not resaturate lift 1 because no water drained from Lift 1 to the sand under-drainage layer. Fresh tailings might have transferred more water to Lift 1 and potentially to the drainage layer.

Soil Matric Suction

Soil matric suction was measured by a hand-held tensiometer and in situ tensiometers. Figure 9 shows the matric suction (kPa) versus height above box bottom for Lift 1 (lower portion of the figure) and for Lift 2 continuous down into Lift 1. Generally, the matric suction within the lift decreased with depth, consistent with the moisture content profile. As expected, matric suction increased rapidly and significantly as the lift lost water.

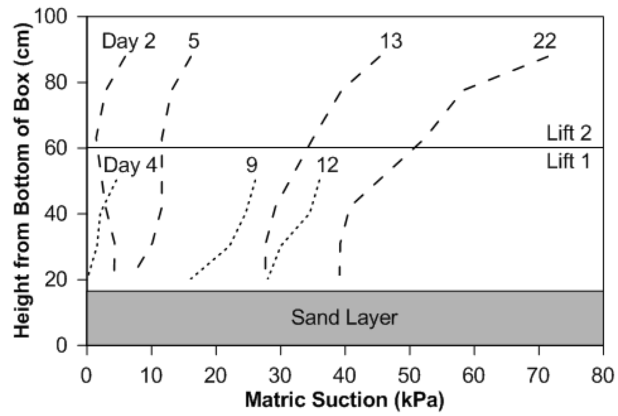

Figure 9. Soil Matric Suction by Tensiometer.

Before the placement of Lift 2, the matric suction profile for Lift 1 was nearly all above 30 kPa (see the Lift 1 day 12 profile in Figure 9). Upon addition of Lift 2, the soil matric suction in Lift 1 dropped to less than 5 kPa, (see the Lift 2 day 2 profile in Figure 9). This loss in matric suction due to re-wetting was reversed nearly fully in Lift 1 (compare Lift 2 day 13 to Lift 1 day 12 in Figure 9), consistent with the water content levels (at least for the bottom TDR) reaching their previous minimum values. Thus, both water content and matric suction recovered from the effect of a fresh lift re-saturating a previously-desiccated lift.

Shear Strength

Figure 10 presents the shear strength profile for both lifts in the box, measured by vane shear tests. The peak shear strength of the first lift exceeded 20 kPa before placement of the second lift. With the placement of the second lift, the underlying lift lost some strength, but the loss was modest. By day 6 after placement of Lift 2 the first lift had regained and exceeded the shear strength previously achieved. The proportionate loss was smaller and the recovery was more rapid than that observed for water content and matric suction.

The observed strength loss would likely have been sustained longer and may have been greater had the second lift been fresh undecanted tailings. However, observations from the field testing (reported separately) suggest that the deposit still regained its strength relatively quickly, even with fresh tailings as the added lift.

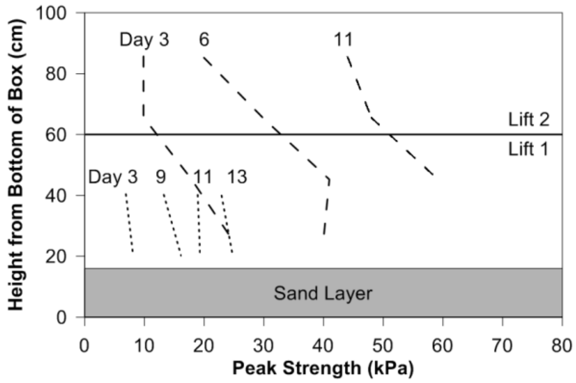

Figure 10. Shear Strength Profile.

Possible explanations for this strength recovery include: 1) the flocculated TSRU tailings exhibited high enough permeability that there was little ponding of water at the surface (none in the second lift in the box tests, and little in the field tests), so water loss from the deposit is prompt and matric suction recovers quickly; 2) the load added by the second lift compacts the underlying material; and, 3) there is evidently a strength benefit from the effects of flocculation, which would be retained in the immediately underlying lift. The first and third factors noted above would have been compromised if the flocculated structure had been lost due to storage or handling. The combined observations from the field work and the box tests suggest that strength recovery may occur timely and consistently for thickened TSRU tailings of the character tested here.

The second lift developed shear strength more rapidly than the first lift, and to a magnitude disproportionately larger than its water loss or matric suction gain. As seen in Figure 10, the entire deposit (Lifts 1 and 2) reached peak strengths of 40 to 60 kPa by 11 days after placement of the second lift. This was still within the period of Stage 1 drying for Lift 2, which ended at about 12 days. These strengths are 2 to 3 times the Lift 1 strengths at 11 days after initial placement.

These findings are broadly consistent with qualitative observations from the field pilot test with thickened TSRU tailings (reported elsewhere), and in fact also with in-line flocculated mature fine tailings. Similar observations from surface deposits of thickened gold tailings have also been reported (Dunmola 2012).

It is also worth noting that thickened TSRU tailings gained strength during storage in the drums, prior to placement in the test boxes. Peak shear strengths of 2 to 4 kPa were reported for vane shear tests performed on the material in the drums the day prior to placement in the boxes. These tests were performed prior to decanting the supernatant from the drums. This shows these tailings achieved a measure of strength from flocculation and consolidation even without runoff, drainage, and drying.

End of Program Testing

Figure 11 presents the density, void ratio, and degree of saturation profiles at the end of the test during box deconstruction. The profiles are consistent with the data collected during the course of the experiment. The material was less dense and drier at the surface of the box, increasing in density and degree of saturation with depth. The void ratio was relatively constant with depth, and at about 1.1 is on the loose side, as would be expected for a material with a considerable proportion of fines, that has settled under self-weight. The degree of saturation profile shows a similar overall pattern as the TDR moisture data. The surface of the top lift was less saturated and saturation increased with depth.

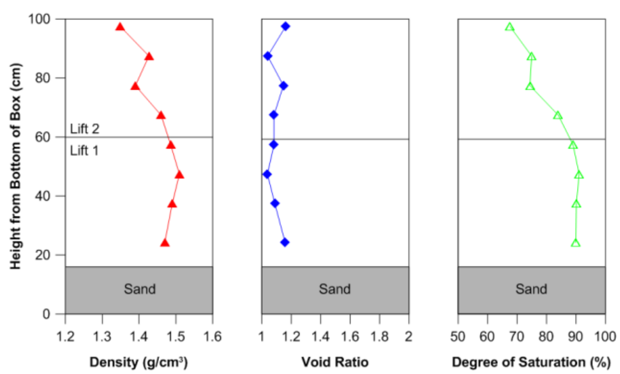

Figure 11. Deconstruction: Density, Void Ratio, and Degree of Saturation Profiles (Two-Lift Box).

During testing of the box with 3 lifts and drainage between each lift, the degree of saturation increased in each successive lower lift, as seen in Figure 12. The high saturation in the bottom lift occurred despite the under-drainage in the box.

This suggests there may be an issue with high water content in the underlying lifts as a multi-lift tailings program is implemented. The water content conditions in the three-lift box were also influenced by the capillary barrier effect of the sand drainage layers between lifts. Yet, the evidence from the three-lift experiment showed acceptable strengths in the wetter lower lifts. In fact, in all boxes the strength in the first-placed lift by day 12 after Lift 2 (and 3) placement was in excess of 40 kPa.

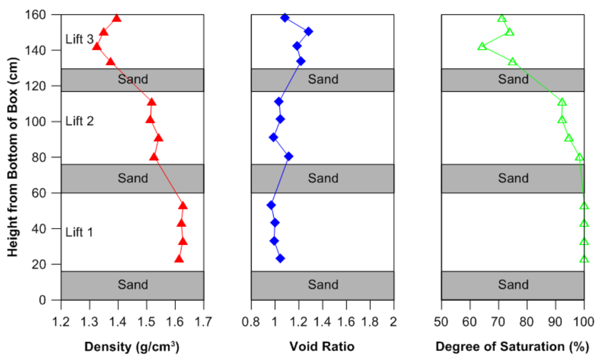

Figure 12. Deconstruction: Density, Void Ratio, and Degree of Saturation Profiles (Three-Lift Box).

CONCLUSIONS

The thickened TSRU multi-lift box testing points to two important preliminary conclusions, which should be confirmed by follow-on work using freshly thickened TSRU tailings and more than three lifts:

1. The first lift recovered satisfactorily from the re-wetting effect, resulting from the placement of a second lift. Strength in the first lift recovered and exceeded the previously-recorded values. Soil matric suction returned to prior values, as did water content, although the drier upper portion of a lift may not return to its prior minimum water content.

2. The second lift developed matric suction and gained strength more rapidly than the first lift. This is largely attributable to the degree of dryness achieved in the first lift. While this is in part an artifact of the water removal associated with drum storage of the tailings, several factors suggest this may also be true even if the experiments had been run with freshly thickened tailings. Those factors include the favorable drainage properties of thickened TSRU tailings, their rapid water release and matric suction gain, and their rapid strength gain.

Other important observations from the experiment include:

3. There was significant water loss from the thickened TSRU tailings by surface runoff or downward drainage (decant from drums, plus runoff and drainage in the box). These tailings benefitted from significant dewatering due to consolidation, enhanced by good under-drainage. Strength exceeded 5 kPa within days after that initial volume of water was lost and strength gain continued at a fairly rapid pace. This means that there is relatively little reliance on evaporation for initial strength gain with these tailings. This would suggest thin-lift placement of thickened TSRU tailings could be less dependent on good drying weather than is usual in conventional thin-lift fines drying.

4. As indicated by the rapid moisture loss, matric suction increase, and strength gain of thickened TSRU, it is likely that additional lifts can be placed at relatively short time intervals compared to experience with thin-lift drying for other oil sands tailings such as mature fine tailings. The timing of additional lifts would be dictated primarily by the strength and dryness of the underlying lift.

5. The current study did not observe much drainage reporting to the sand drainage layer underlying the thickened TSRU lifts. The effect of the under-drainage would likely have been more significant if the decant water had not been removed in the storage drum. Thicker deposits might also have increased the significance of the drainage layers.

6. Stage 2 drying was not evident during the 14 days of Lift 1. Although Lift 2 reached Stage 2 drying, it had achieved much higher strength than Lift 1 prior to exiting Stage 1 drying. This suggests that the TSRU material can reach target strengths

with relatively high water contents, before Stage 2 drying begins. This is consistent with observations with other flocculation-based technologies for dewatering mature fine tailings (Beier et al, 2012).

7. It is vital to preserve the flocculated structure of the thickened TSRU tailings to realize the beneficial effects reported here. This is most critical for the upper lifts of a deposit, where rapid water release is an essential factor in achieving the moisture loss, matric suction increase, and strength gain as observed in the current study.

8. Testing deposition under sub-zero temperatures and the effect of freeze/thaw climatic cycles is recommended.

A larger-scale program would be necessary to determine if and how the findings in this study may be applied to thickened TSRU tailings management at a commercial scale.

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