Development of a Tailings Management Simulation and Technology Evaluation Tool

by

Nicholas Alvin Beier

A thesis submitted in partial fulfillment of the requirements for the degree of

Doctor of Philosophy

in

Geo-Environmental Engineering

Department of Civil and Environmental Engineering
University of Alberta

ABSTRACT

This research aims to assist in the assessment of tailings management technologies through the development of a dynamic simulation model. The developed model (TMSim) incorporates the mine plan, various stages of dewatering including classification, pre- and post-deposition dewatering, and an impoundment material balance including tailings, process water, construction material and capping materials. Through simulations of a simple metal mine operation and a complex oil sands operation, TMSim demonstrated it can evaluate technologies and mine plans and diagnose potential drawbacks and strengths. This information can then be used to strategically guide and support technology development and resource expenditure.

During the evaluation of oil sands fine tailings technologies, it was found that the use of chemical amendments (flocculants) to augment dewatering and strength gain may present some challenges. Chemically-amended fine tailings can have low storage efficiencies and these deposits may exhibit sensitive, metastable behavior upon deposition. Additionally, flocculation of fine tailings may actually hinder the self-weight consolidation process through the development of an apparent pre-consolidation pressure.

The TMSim modelling tool was utilized to simulate a model oil sands mine based on the Syncrude Aurora North mine utilizing composite tailings (CT) technology. Model inputs and functions were based on publicly available sources of information. Based on the mass balance agreement with the Syncrude tailings

plan, the TMSim model was established to be an effective quantitative tool that can be used in the evaluation of technologies for oil sands mining operations.

A tailings plan was compiled using a novel dewatering technology, cross flow filtration (CFF), as the core tailings technology. The CFF tailings plan was then evaluated with the TMSim model and the model oil sands mine plan. The cross flow filtration process provides an opportunity to deposit high density tailings stacks requiring minimal containment. Two thirds of the yearly process water demand can also be satisfied by immediate recycle from the CFF process resulting in lower green house gas production. Additionally, if fluid fine tailings (FFT) spiking is incorporated, existing inventories of FFT can be consumed and stored in the pore space of the CFF tailings.

Preface

The research in this thesis was completed by Nicholas Beier. Portions of Chapter 2 of this thesis have been published in the 2008 Canadian Geotechnical Conference and 2009 Tailings and Mine Waste Conference. Mr. Beier was responsible for the literature review and manuscript composition. M. Alostaz assisted with the freeze-thaw dewatering summary in the manuscript. Co-authors provided editing comments only.

Portions of Chapter 3 and 4 were previously published in the Tailings and Mine Waste 2009 Conference, 2012 International Oil Sands Tailings Conference, and the 5th International Young Geotechnical Engineers Conference (2013). Co-authors provided editing comments only.

Versions of Chapter 6 were previously published in the 2012 Canadian Geotechnical Conference and the Canadian Geotechnical Journal. Data was collected and graciously provided by Shell Canada Energy (A. Dunmola). Mr. Beier conducted the data analysis, presentation of data and wrote the manuscript. Co-authors provided editing comments only.

A Portion of Chapter 8 was previously published in the 2014 International Oil Sands Tailings Conference.

References:

Chapter 2

Beier, N. and Sego, D. (2008). Dewatering of Oil Sands Tailings Using Cross Flow Filtration. 61st Canadian Geotechnical Conference and 9th joint IAH-CNC Groundwater Specialty Conference, Edmonton, Alberta, September 22-24, 2008. 8 pp.

Beier, N., Alostaz, M. and Sego, D. 2009. Natural Dewatering Strategies for Oil Sands Fine Tailings. Tailings and Mine Waste 2009 Conference, Banff, Alberta, November 1-4, 2009. 845-858.

Beier, N., Sego, D., and Morgenstern, N. 2009. Dynamic Simulation of tailings Management Options. Tailings and Mine Waste 2009 Conference, Banff, Alberta, November 1-4, 2009. 441-450.

Chapter 3.

Beier, N., Sego, D., and Morgenstern, N. 2009. Dynamic Simulation of tailings Management Options. Tailings and Mine Waste 2009 Conference, Banff, Alberta, November 1-4, 2009. 441-450.

Beier, N. and Sego, D. 2013. Tailings Management in Canada's Oil Sands. 5th International Young Geotechnical Engineers Conference, Paris, France, 31 August – 1 September 2013, pp. 4.

Chapter 4.

Beier, N., Sego, D., and Morgenstern, N. 2012. Simulation of Tailings Management Options. 3rd International Oil Sands Tailings Conference, Edmonton, Ab. Dec 5-8, 2012. 9 pp.

Chapter 6.

Beier, N.A., Wilson, G.W., Dunmola, A., and Sego, D.C. 2012. Geotechnical Aspects of Flocculation-based Technologies for Dewatering Mature Fine Tailings. 65th Canadian Geotechnical Conference, Winnipeg, Manitoba, September 30 – October 3, 2012. 8 pp.

Beier, N.A., Wilson, G.W., Dunmola, A., and Sego, D.C. 2013. Impact of flocculation-based dewatering on the shear strength of oil sands fine tailings. Canadian Geotechnical Journal, 2013, 50(9) 1001-1007.

Chapter 8.

Beier, N., Sego, D., and Morgenstern, N. 2014. Assessing a novel technology using a tailings management simulation model. 4th International Oil Sands Tailings Conference, Edmonton, Ab. Dec 7-10, 2014. 10 pp

Acknowledgments

I would like to extend a sincere thank-you to my supervisors, Dr. Dave Sego and Dr. N. Morgenstern for their patience, guidance, technical insight and financial support over the course of my studies. The opportunities they have afforded me, facilitated my growth both technically and professionally.

I would also like to thank Alan Fair and the COSIA Tailings EPA for the opportunity to be involved with Tailings Roadmap project and for the many technical discussions related to oil sands tailings technologies and management. Thank you to Nathan Deisman for all your assistance with the numerical modeling and code writing. I would also like to thank the many graduate students I have worked with over years for their assistance in the lab and debates about tailings including Sorta Amarebh, Reza Nik, Silawat Jeeravipoolvarn, Nam Pham and Chenxi Zhang. Christine Hereygers, thank you for your technical assistance in the lab and at the OSTRF and making yourself always available to help. Thank you also Dr Scott, Dr. Wilson and Dr. Kabwe for insight and data on tailings management and behavior. I was also a pleasure working with Sally Petaske and the rest of the Geotechnial Centre staff.

This research would not have been possible without financial support from Alberta Innovates Technology Futures (formerly Alberta Research Council), NSERC, and the OSTRF. Their support is greatly appreciated. Additonally, the software and support provided by GoldSim and FSConsol by Gord Pollock were invaluable to the completion of the thesis.

Finally, I would like to thank my family, especially my wife, Kristina and our children for their patience, love and support throughout my studies.

Table of Contents

List of Tables

List of Figures

List of Symbols and Acronyms

AERI	Alberta Energy Research Institute
AFD	Atmospheric fines drying
AI-EES	Alberta Innovates- Energy and Environment Solutions
AMD	Acid mine drainage
$A_{catchment}$	Area of the run off catchment
A_f	Area of filter surface
A_p	Area of discharge pipe
AEPN	Aurora East Pit North
ASB	Aurora settling basin
$A_{storage}$	Total impoundment area
A_{tail}	Surface area of tailings layer
A_{water}	Area of pond water
BAW	Beach above water
B_o	Beaching characteristic parameter of tailings
bbl	barrel
BBW	Beach below water
BT	Coarse beach deposit
c'	Mohr-Coulomb cohesion coefficient
c_1	Kynch suspension concentration
C_{dredge}	Clay content of dredged tailings
CFF	Cross flow filtration
$C_{i\ pond}$	Concentration of chemical species, "i"
Ckc	Kozeny Carmen constant
CNRL	Canadian Natural Resources Limited
COF	Cyclone overflow
CT	Composite or consolidated tailings
C_{tails}	Clay content of tailings
CTMC	Consortium of Tailings Management Consultants
CUF	Cyclone underflow
Cv	Coefficient of Consolidation
C_w	Solids content by mass (%)
D_{50H}	Mean beach particle size
DDA	Dedicated disposal area
d_{frozen}	Depth of tailings frozen
DSM	Dynamic simulation modeling
d_{thaw}	Depth of thaw
E	Extraction efficiency
ERCB	Energy Resource and Conservation Board
e	Void ratio
e_{fines}	Fines void ratio
e_m	Void ratio where grain to grain contact starts
e_0	Void ratio prior to deposition
Ep	Potential evaporation rate
EPL	End pit lakes

ETF	External Tailings Facility
F	Fines content
F_{COF}	Fines content of cyclone overflow
F_{CUF}	Fines content of cyclone underflow
FC%	Beach fines capture
F_{ORE}	Fines content of ore
FSA	Design filter surface area
F_{treat_tail}	Fines content of Stage 2 dewatered tailings
FFT	Fluids fine tailings
G	the ratio of the specific weight of the grains to the specific weight of the water
h_{aw}	Elevation above pond water level
H_{cap}	Thickness of capping material
h_f	Suspension thickness
H_{sed_tail}	suspension/sediment interface height
$H_{storage}$	Maximum dyke height
H_{tail}	Height of tailings
H_{tail_bbw}	fictitious discharge height for the tailings below the water level
H_{water}	Thickness of water cap
i	Beach slope (%)
iBAW	Tailings slope above water surface
iBBW	Tailings slope below water surface
ILTT	*In-line* thickened tailings
I_L	Liquidity Index
I_p	Plasticity index
JPM	Jackpine Mine
k	Saturated hydraulic conductivity
KC	Kozeny Carmen method
MFT	Mature fine tailings
Mm^3	Million cubic metres
MRM	Muskeg River Mine
m	Time since deposition
m_v	Coefficient of volume compressibility
n_0	Blight's dimensionless constant
NPV	Net present value
NST	Non-segregating tailings
OB	Overburden
OB_{split}	Fraction of overburden suitable for use as construction material
P_f	Filtrate pressure
PSD	Particle size distribution
PSV	Primary separation vessel
PT	Paste tailings
Q	Pipe flow rate
$Q_{basal\_seepage}$	Rate of seepage to the base of a tailings deposit
Q_{beach}	Rate of solids reporting to beach
Q_{cap}	Rate of material reporting to the cap

$Q_{chemical}$	Rate of chemical addition
Q_{COF}	Rate of cyclone overflow
Q_{CUF}	Rate of cyclone underflow
$Q_{concentrate}$	amount of concentrate extracted from the ore
$Q_{consol\_release}$	Rate of water released during consolidaiton
Q_{consol_tail}	Rate of tailings consolidation
Q_{dredge}	Rate of dredging for reprocessing tailings
$Q_{dyke\_seepage}$	Seepage through a dyke
Q_{env_water}	Water liberated during environmental dewatering
Q_{make_up}	Water required to fulfill the extraction process water demand
Q_{misc}	Miscellaneous process water flows
Q_{mill_losses}	Water lost in mill due to spillage
Q_{ore_feed}	Ore feed rate
Q_{OB_const}	Construction overburden rate
Q_{OB_waste}	Waste overburden rate
$Q_{overburden}$	Overburden removal rate
Q_{pond_evap}	Evaporation from process water pond
$Q_{precipitation}$	Precipitation rate
$Q_{process\_water}$	the process water demand rate
$Q_{reclaim}$	Rate of water reclaim to the extraction process
$Q_{release}$	Sedimentation and consolidation water release rate
$Q_{residual\_conc}$	mass of concentrate (i.e. bitumen) that reports to the tailings stream
Q_{reject}	Reject rate
Q_{run_off}	Rate of water runoff from beaching
Q_{sed_tail}	Rate of tailings undergoing sedimentation
Q_{sed_water}	Water liberated during sedimentation
$Q_{seepage\_return}$	Seepage collection return rate
Q_{tails}	Rate of tailings production
Q_{tail_water}	Water in the tailings stream
$Q_{treat\_concentrate}$	Recovered concentrate from Stage 2 dewatering
Q_{treat_tails}	Rate of dewatered tailings from Stage 2 dewatering process
Q_{treat_water}	Rate of water liberated from Stage 2 dewatering
r	radius
S	Sand content
Ss	Specific surface
SCT	Straight coarse tailings
SFR	Sand to fines ratio
Sg	Specific gravity
SR_F	Specific resistance to filtration
S_t	Sensitivity
S_u	Undrained peak shear strength
S_{ur}	Undrained residual shear strength
S_{uR}	Undrained remoulded shear strength
SWCC	Soil water characteristic curve
t	time

TFT Thin fine tailings
TMS Tailings management system
TRO Tailings Reduction Operations
TSRU Tailings solvent recovery unit
TT Thickened tailings
u Excess pore pressure
UDF User defined function
VBA Visual Basic for Applications
$Vol_{freeboard}$ Available free board volume
Vol_{frozen} Volume of tailings frozen
Vol_{slurry} volume of tailings slurry per unit mass of tailings solids
$Vol_{storage}$ Volume of available storage
Vol_{water_cap} Water cap volume
Vs Particle settling velocity
w Water content as % (mass of water divided by mass of solids (sand, fines and bitumen)
w_L Liquid limit
w_P Plastic limit
w_S Shrinkage limit
WT Whole tailings
X_{chem} tonne of chemical required per tonne of clay
Xd x-coordinate of tailings discharge location
Yd y-coordinate of tailings discharge location
Z_f Water/suspension interface height
$\beta(e)$ interaction coefficient
ε_{th} Thaw strain
γ' Submerged unit weight
γ_w Unit weight of water
φ' Mohr-Coulomb friction angle
ρ_{dbeach} Beach dry density
ρ_p Particle density
ρ_f Suspension density
ρ_{sed} Sediemtn density
ρ_w Fluid density
σ'_p Pre-consolidation pressure
σ'_v Vertical effective stress
μ_w Fluid viscosity or dynamic viscosity of water

1 INTRODUCTION

Tailings are a by-product of the mineral and hydrocarbon extraction process at mining operations. These extraction tailings are typically low density, high water content, segregating slurries. Without processing and dewatering, tailings may require fluid containment upon deposition and exhibit poor dewatering and strength gain behavior (Sobkowicz and Morgenstern 2009; Vick 1990). Consequently, raw extraction tailings have proven to be troublesome for mining operations to meet closure and reclamation goals. For example, in the oil sands industry, a low density, high fines content tailings called mature fine tailings (MFT) or fluid fine talings (FFT) is formed after deposition of whole tailings. In future references to oil sands tailings within this thesis, fines denote the material finer than 45 μm and will include silt, clay and residual bitumen. All other reference to fines will represent the geotechnical fines definition of material finer than 75 μm. Additionally, the coarse fraction of oil sand tailings represents the material greater than 45 μm. The MFT requires long term containment and further dewatering is expected to take decades (Sobkowicz and Morgenstern 2009). On average, approximately 0.25 m^3 of MFT and 1 m^3 of sand are produced for every barrel of crude oil produced from an oil sands mine. To date, there is an estimated 850 million m^3 of MFT stored at the operating mine sites (Fair and Beier 2012). The oil sands region is currently dominated by a wet landscape with several, large above grade containment structures storing fluid tailings and process water.

In lieu of the oil sands industry's past tailings management practices, continual accumulation of fine tailings and the associated risks to reclamation activities, Alberta's Energy Resource Conservation Board (ERCB) elected to regulate oil sand fine tailings through performance criterion (Directive 74). The aim of the Directive is to reduce fluid tailings accumulation by capturing the fines in dedicated disposal areas (DDAs) and create trafficable surfaces for progressive reclamation. The Directive requires a minimum undrained shear strength (S_U) of 5 kPa for tailings material deposited during the previous year. Additionally, five years after deposition, a trafficable surface must be achieved with a minimum S_U of 10 kPa.

In order for tailings deposits to meet reclamation, closure, and regulatory objectives, the deposits should develop strength at a rate sufficient to allow for reclamation activities and develop a low compressibility to minimize long term settlement. Therefore, tailings must undergo significant dewatering after they are formed. During dewatering, tailings will undergo changes in strength, saturated hydraulic conductivity, and compressibility of several orders of magnitude. With sufficient dewatering, tailings can develop the required long term stiffness and strength (50 to 100 kPa) to support reclamation activities (Boswell and Sobkowicz 2010).

There are three stages of dewatering tailings may undergo before they are deemed suitable for reclamation (Boswell and Sobkowicz 2010). The first stage involves mechanical classification (i.e. hydrocyclones) or natural classification during deposition of the tailings stream. Hydrocyclones may be used to classify tailings slurries into two streams, a low density, fine grained overflow and a coarser, dense underflow. Often, the coarse under flow is used as a structural component of tailings impoundment structures. Tailings may also undergo natural segregation upon deposition. The coarse fraction of the tailings will settle and form a beach deposit and the fine grained runoff will form a low density slurry within the impoundment. The second stage of dewatering includes various mechanical, chemical and electrical methods such as thickeners, centrifuges, and filters and may employ chemical flocculant addition to enhance dewatering. For fine grained tailings, these technologies can be used to dewater the tailings to near, but still wet of their liquid limit. For coarse grained tailings, these technologies will decrease the tailings water content to near the desaturation point. For gold and copper tailings, the desaturation point may occur at a volumetric water content of near 45% (Qiu and Sego 2001). Upon deposition following Stage 2 dewatering, the tailings deposits will typically have strengths of a few hundred pascals (Boswell and Sobkowicz 2010). A summary of the Stage 1 and 2 dewatering technologies and the approximate strength behavior of the resulting tailings deposit is presented in Figure 1.1. This relationship is influenced by the particle size, shape, mineralogy, as well as pore fluid chemistry of the tailings, therefore is provided for comparative purposes. The final stage of dewatering following deposition (Stage 3) includes time dependent and environmental dewatering processes such as sedimentation/consolidation freeze/thaw dewatering, desiccation and evapotranspiration. Deposition and deposit management strategies can be employed to maximize Stage 3 dewatering in order to achieve the strength required to meet reclamation targets.

Management of tailings also includes the construction and operation of tailings storage facilities (i.e impoundments). Impoundments may be constructed from the tailings (hydrocyclone underflow or segregated beach deposits), from other mine waste or naturally occurring materials. Mined out pits may also be used as tailings impoundments. The construction of the impoundments must be coordinated with the deposition and storage requirements of the tailings and associated process water to ensure sufficient capacity is available. The required capacity of the impoundment is a function of the tailings dewatering processes (described above), the interaction with the environment (i.e. seepage, precipitation, evaporation), and process water demands from the extraction process in addition to other miscellaneous site water. Mine operators manage tailings and impoundments through the implementation of a tailings management system (TMS) that incorporates all aspects of the tailings dewatering and their associated storage facilities. The various dewatering and deposition options available for tailings management are summarized in Figure 1.2.

As mine operators look to alternative technologies and management process to reduce their inventory of fine tailings, expedite the reclamation process and meet regulatory requirements, there is an increasing need for technology evaluation/screening tools. The underlying processes in a TMS are typically modeled separately using complex, analytical tools. There are few models or approaches available to the public that incorporate the tailings management process as a whole system over the life of the mine. Existing publically available tailings and mine planning models also do not take into account the dynamic nature of the tailings management process and rely solely on final tailings volumes or static tailings forecast models. These models or tools are not well suited for evaluating multiple technologies or management strategies.

A recent joint industry-government initiative (Oil Sands Tailings Technology Development Roadmap, Tailings Roadmap; CTMC 2012) developed a foundation for assessing tailings technologies but the process was based on qualitative assessments. The Tailings Roadmap provided a snap shot of the state of tailings technologies applicable to oil sands in general. However, site specific assessments are still required to evaluate a technology for a particular mining operation. Future evaluation of technologies in general or for specific sites using the Tailings Roadmap approach may require significant effort.

1.1 OBJECTIVE

There currently is no publically available simulation/assessment model for evaluating tailings technologies and management options quickly and efficiently. Therefore, a simulation model is needed that incorporates the mine plan, various stages of dewatering including classification, pre- and post-deposition dewatering, and an impoundment material balance included tailings, process water, construction material and capping materials. The objective of this research program is to develop a tailings management simulation model (tool) that can be used to evaluate tailings technologies and incorporate the above processes. The simulation model should guide the tailings planner/operator/regulator through the process of tailings management to attain a practical, economical, and environmentally sound solution. The model will simulate the tailings system over time, demonstrate various outcomes by alternating management practices, and may also be used to conduct sensitivity analyses. Essentially, the simulation model will be a what-if tool to experiment with various operating strategies or design alternatives to support technology assessment, scenario-analysis, fore-sighting and mine planning (Scaffo-Migliaro 2007; Halog and Chan 2008). The simulation model will also be used to strategically guide further research and resource expenditure in the development of new tailings technologies and management strategies. Given the complexity intricacies of tailings management systems, simplifications and assumptions will be required in the development of the model. Therefore, the simulation model should not be regarded as a final design tool, but rather a planning or evaluation tool used to assist in the design making process. Additionally, this research will not include methods for managing sulphide bearing ore bodies and associated wastes including acid mine drainage.

1.2 OUTLINE OF THESIS

This thesis is organized into nine chapters. A brief introduction for each chapter is provided below including how the chapter contributes to the objective of the research program.

Chapter 2 defines the intricacies of tailings management at mining operations. It outlines the physical, chemical and natural processes that tailings undergo during their management at typical mining operations. Specific dewatering technologies and various depositional and containment options are also reviewed.

Chapter 3 presents the development of the tailings management simulation model (TMSim) developed using publically available data and tailings plans. The simulation model utilizes a multitude of process-based, empirical and qualitative formulations. User inputs (data and functions) can be static or defined as functions of time. Stochastic model inputs can also be utilized in TMSim, however, they were not included in this research program. The model incorporates the major components of a tailings management system such as the extraction plant, tailings dewatering (by segregation, chemical or physical processes, or naturally), the impoundment (containment, water cap etc.) and the environment.

In Chapter 4, the TMSim implementation is detailed using a dynamic systems approach. An object orientated, systems dynamic modeling software called

GoldSim was used as the "simulation engine" for the model. The model was then validated using experimental, analytical and numerical data sets. A metal mine scenario was also simulated to demonstrate the applicability of the model to a real mine scenario.

Chapter 5 provides a summary of the historical tailings management practices and challenges in the oil sand industry and the proposed future tailings management plans based on publically available sources of information. Due to a delay by the Government of Alberta in publically releasing the tailings management plans, the information presented only includes data that was available at the time of writing.

Chapter 6 highlights some of the geotechnical issues and challenges associated with flocculation-based technologies for dewatering oil sand fine tailings. The discussion will focus on the undrained strength development, compressibility behavior and storage implications based on publically available data and data provided by Shell Canada Energy's Muskeg River Mine. The analysis emphasizes the potential implications of adopting flocculation-based technologies on tailings and mine plans.

In Chapter 7, the applicability of the TMSim model to an oil sands mine is presented. A model oil sands mine scenario based on Syncrude's Aurora north mine site is evaluated using the TMSim model. Simulations were conducted utilizing composite tailings (CT) as the main tailings management technology.

In Chapter 8, several simulations were conducted with the TMSim model to demonstrate its utility as an evaluation tool for novel tailings dewatering technologies. A novel tailing dewatering technology (crossflow filtration) was evaluated as a potential candidate for managing oil sand tailings.

Finally, Chapter 9 provides a summary of the conclusions and observations developed during the course of the research program.

1.3 FIGURES

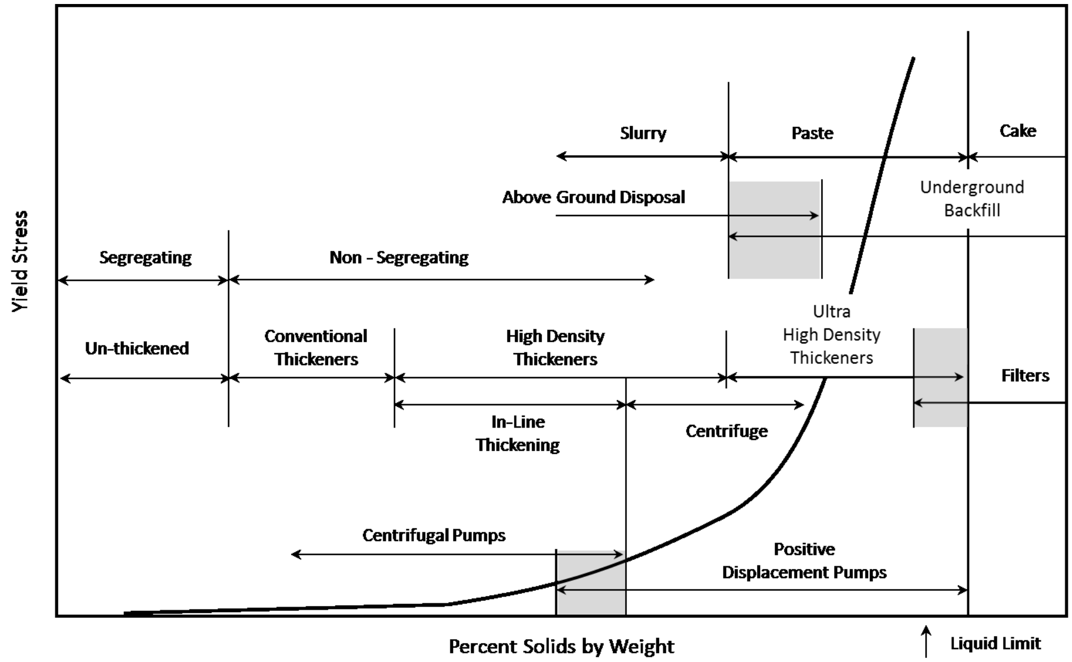


Figure 1.1. Continuum of Stage 1 and 2 dewatering methods and tailing behaviour. (modified from Jewell and Fourie 2006).

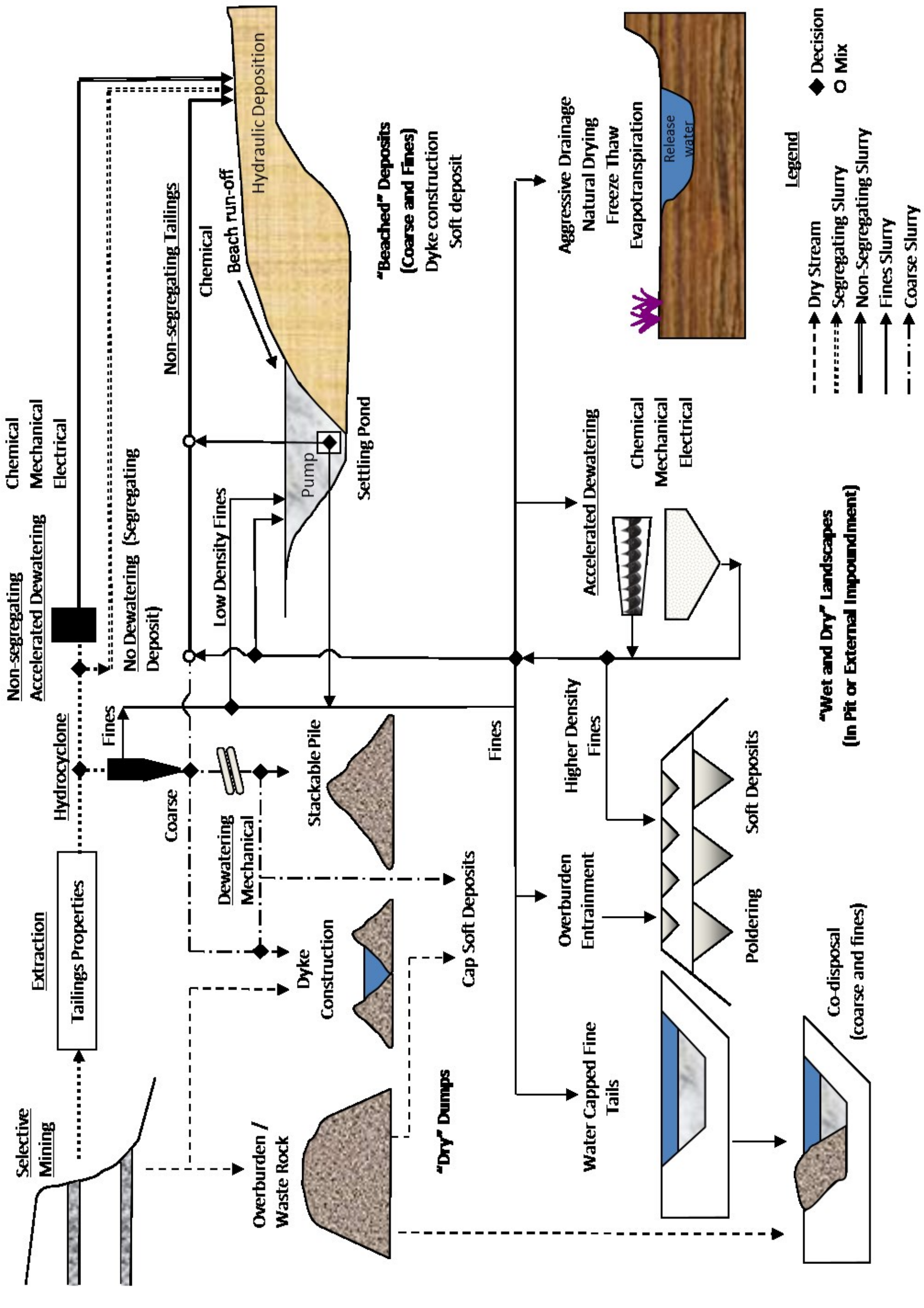


Figure 1.2. Mine Waste Management Strategies (Modified from Sheeran 1993).

1.4 REFERENCES

Boswell, J.E.C. and Sobkowicz, J.C. 2010. Environmental assistance for tailings disposal. 63rd Canadian Geotechnical Conference, Calgary, Alberta, 12-16 September 2010, pp. 665-669

Consortium of Tailings Management Consultants (CTMC), 2012. Oil sands tailings technology deployment roadmaps: Project report. Report prepared for AI-EES and the Oil Sands Tailing Consortium (OSTC). June 29, 2012. 5 Volumes.

Fair A.E. and Beier, N.A. 2012. Keynote Paper: Collaboration in Canada's Oil Sands: Fluid Fine Tailings Management. In Proceedings of the 3rd International Oil Sands Tailings Conference, Edmonton, Alberta, 3-5 December 2012, pp. 3-12.

Halog, A. and Chan, A. 2008. Developing a dynamic systems model for the sustainable development of the Canadian oil sands industry. International journal of Environmental Technology and Management, **8**(1): 3-22.

Jewell, R.J. and Fourie, A.B. 2006. Paste and thickened tailings – A guide. 2nd Edition. Australian Centre for Geomechanics, Nedlands, Western Australia, Australia. Pp. 257.

Qiu, Y. and Sego, D.C. 2001. Laboratory properties of mine tailings. Canadian Geotechnical Journal **38**: 183-190.

Scaffo-Migliaro, A. 2007. Modeling and simulation of an at-face slurry process for oil sands. MSc thesis, Department of Mechanical Engineering, Univeristy of Alberta, Edmonton, Alberta

Sheeran, D. 1993. An improved understanding of fine tailings structure and behavior. Fine Tailings Symposium, Environment Canada, Edmonton, Alberta, 4-7 April 1993, 11 pp.

Sobkowicz, J.C. and Morgenstern, N.R. 2009. A geotechnical perspective on oil sands tailings. 13th International Conference on Tailings and Mine Waste, Banff, Alberta, 1-4 November 2009. University of Alberta, pp. xvii-xli.

Vick, S.G. 1990. Planning Design and Analysis of Tailings Dams. BiTech Publishers Ltd. Pp. 369.

2 TAILINGS BEHAVIOUR AND MANAGEMENT

2.1 INTRODUCTION

Our modern world relies on mineral commodities such as coal, copper, diamonds, gold, iron, platinum, and zinc to name a few. Mining and mineral processing to produce these commodities ultimately lead to the production of waste by-products including waste rock and a finer grained slurry called "tailings". Management of these tailings and waste rock results in environmental challenges and financial burdens for operators. The volume of tailings generated and associated environmental hazards for a particular mine depend upon the individual ore bodies and the physical/chemical extraction processes. For example, annual tailings production at the Kidd Creek copper-zinc mine is 2.92 million tonnes (Fitton 2007) while a typical oil sands mine operating at 200,000 barrels per day could produce up to 235 million tonnes a year (Allen 2008).

Mine operators manage these tailings through the implementation of a TMS. In a recent technical publication, the Australian Government (2007) defined tailings management as "managing tailings over their life cycle, including their production, transport, placement, storage, and the closure and rehabilitation of the tailings storage facility." Therefore, one can see the management of tailings consists of several components including tailings treatment such as dewatering, transport to and construction of geotechnically stable impoundments, water recovery and recycle, effluent treatment, and restoration of the site (Ritcey 1989).

There are several potential management options for mine waste tailings as evident by the numerous case studies in the literature. The following sections will review the development of the waste streams encountered at typical mining operations and their subsequent management. The intent is to provide an overview of the various tailings management technologies including the physical, chemical and natural processes that tailings undergo rather than detail specific case studies.

2.2 MINE WASTE STREAMS

There are generally two major waste streams at most mining operations: a dry stream and a wet stream. The dry stream consists of chunks of blast rock, overburden (OB), or lean ore. Wet streams normally consist of a finer grained slurry called "tailings" comprised of sand to clay size ground rock or minerals, water, and residual process chemicals (Morgenstern and Scott 1995). Sludge arising from process water treatment may also contribute to the volume of wet streams to be managed at a mining operation (Ritcey 1989). Mechanical segregation or depositional methods may further separate the wet tailings into a coarse (sand) stream and a fines (silt/clays) stream. There are several options available to the mine operator for management of these streams either together or separately as illustrated in Figure 2.1 and described below. The following overview will not include methods for managing sulphide bearing ore bodies and associated wastes including acid mine drainage.

2.2.1 Dry Streams

Dry wastes, identified with dashed lines in Figure 2.1, may be heaped in non-impounding "dry" dumps or used as a construction material for external tailings impoundment dykes (Vick 1990). The material may be transported via truck or conveyor to the dumps or dykes (Morgenstern and Scott 1995). Co-disposal of the coarse waste rocks and tailings slurries in the same surface impoundment or in-pit may also be possible (Bussiere 2007). Through co-disposal, the simultaneously or alternately deposited mixture may be modified such that the final deposit characteristics are superior as compared to each of the separate streams (Wilson et al. 2002).

2.2.2 Wet Streams

2.2.2.1 Tailings Formation and Composition

Prior to understanding the behaviour and management of mine waste tailings, one must first understand how these various tailings streams are formed. Upon discovery of a recoverable ore body, operators can utilize open pit mining, underground mining, heap leaching or in situ extraction to remove the ore from the ground (Bussiere 2007). Then, the valuable component of the ore must be separated from the parent material or gangue. Crushing and grinding of the ore is required to release and expose the valuable component (mineral, metal, bitumen, etc.). The final gradation of crushed ore will depend upon the degree of grinding and parent material (amount of clay, silt, sand present). Typically, more valuable minerals require finer grinding. After grinding, the valuable component of the ore must be separated from the parent material. Concentration of the mineral is achieved by gravity or magnetic separation, floatation, chemical leaching using solvents or releasing agents, or by heat (Vick 1990). The separation process results in two separate streams: a concentrated valuable mineral and a residue devoid of the target mineral. This residue, typically in a slurry form is called "tailings". Typical grain size distributions of various tailings streams are presented in Figure 2.2. For hard rock mines, tailings slurries typically have particle sizes between 2 and 1000 microns (1 mm) (Bussiere 2007; Blight 2010). Tailings slurries from oil sands depend highly on the parent ore material and range from coarse sand (2 mm) to ultra fine clays (< 2 micron) (FTFC 1995). The various extraction processes discussed above influence both the gradation and physiochemical behaviour of the tailings that will ultimately impact the behaviour of the tailings during processing, transport, deposition, and dewatering.

2.2.3 Wet Tailings Management

Wet tailings streams are typically transported hydraulically to the disposal site via pipelines. The physical and chemical characteristics of the tailings depend upon the parent ore body, mining method and extraction process (Vick 1990). Tailings from hard rock mines are generally composed of sand to silt sized particles whereas tailings from oil sands mines consist of sand particles to very fine clay particles (Morgenstern and Scott 1995; Bussiere 2007). Conventional tailings disposal practices utilize hydraulic deposition techniques along or near the external impoundment dyke crest from a single point or multiple locations. Tailings slurries may be discharged below the pond water level (sub-aqueous) or in thin layers on the beach above the water level (sub-aerial). Segregation may occur during sub-aerial deposition resulting in coarser (or denser) particles concentrating near the deposition point or beach zone and finer particles flowing towards or into the pond (Blight 2000). Typically, additional processing is required to render tailings slurries non-segregating. Accelerated dewatering via mechanical (i.e. paste thickeners, high pressure filters) or electrical methods and/or chemical additives may decrease the segregating potential of the tailings slurries resulting in a more homogeneous deposit (Jewell and Fourie 2006; Sobkowicz and Morgenstern 2009).

If sufficient dry waste is not available for the external impoundment dyke, the coarse fraction from the tailings stream may be used (Vick 1990). Segregating slurries may be deposited on the beaches or in constructed cells allowing the coarse fraction to settle and form the dyke while the fines flow to the pond. Hydrocyclones may also be used to mechanically segregate the tailings slurry into a coarse fraction (mainly sand size) and a fine fraction (silts and clays). Coarse cyclone underflow could then be used for dyke construction. The finer grained cyclone overflow may be impounded and allowed to settle or be further processed (Vick 1990; Morgenstern and Scott 1995). Management and reclamation of the segregated fine and coarse fractions may be accomplished separately or in combination.

Coarse tailings or cyclone underflow not utilized for beaches or dyke construction may be mechanically dewatered with high pressure filters and dumped into stackable piles (Davies and Rice 2001). The filtered tailings must be transported via truck or conveyor and form an unsaturated, dense, stable stacks (“dry stack”).

Managing the fines fraction is a significant challenge to the mine operator. Fine tailings inevitably will have high moisture content and low saturated hydraulic conductivity thereby decreasing the consolidation rates (volume reduction) and strength gain (Morgenstern and Scott 1995). Impounded fine tailings, either in-pit or in the external impoundment, may be transferred to an engineered containment cell in-pit and capped with a layer of water to form aquatic landforms referred to as end pit lakes (EPL) in the oil sands industry (FTFC 1995; Fair and Beier 2012). In this scenario, the fines would naturally dewater by self weight consolidation under the water cap. Since lakes and wetlands are typically an integral component of the original terrain, wet landscapes are fundamental to a final reclamation strategy. However, since the fine tailings have negligible shear strength, long term containment in above ground impoundments may prove to be a challenging issue.

A wet landscape alone is not sufficient to manage the potential massive inventory of fine tailings. Re-combination of the fines and coarse fraction may provide an opportunity to consume the inventory of fines and develop a stable landscape in a timely manner (Matthews et al. 2002). Fines from thickened cyclone overflow or dredged from the settling basins may be recombined with the coarse cyclone underflow, and with the use of a chemical binder (coagulant), rendered non-segregating. Under self weight consolidation, the resulting mixture termed "non-segregating tailings" (NST) or CT should dewater naturally to form a dewatered, stable deposit. Balancing the use of coarse tailings as a construction material to provide containment with the deposition of NST is a challenge with this option.

To achieve a "dry" final landscape, aggressive fines dewatering techniques, beyond natural self-weight consolidation in water capped deposits, may be required. Fine tailings from cyclone overflow or dredged from settling basins may be dewatered through several natural or mechanical methods. Mixing of the fine tailings with sufficient dry, clay-shale overburden may result in a soft clay deposit. This new deposit could be strong enough to support a reclamation layer directly or be incorporated into a geotechnically stable land mass (Lord et al. 1993; Morgenstern and Scott, 1995). Accelerated mechanical/chemical dewatering of fines may be achieved with high density/high rate thickeners or paste thickeners (Lord and Liu 1998; Bussiere 2007), high pressure/vacuum filters such as drums, stacked plates, or belt filter presses (Bussiere 2007; Xu et al. 2008); or centrifuges including the horizontal solid bowl scroll or filtering basket types (Lahaie 2008; Mikula et al. 2008; Nik et al. 2008). Fines from accelerated dewatering could be incorporated into a geotechnically stable land mass, poldered within overburden or coarse sand deposits, or deposited in-pit. These deposits may be strong enough to support a reclamation layer directly within a short time frame. Electrokinetic methods utilizing electro-osmosis may also be used to aggressively sediment and dewater fine clay slurries (Mohamedelhassan 2008).

Natural, aggressive dewatering of fines can be accomplished through actively managing release water from an existing deposit or re-handling and subsequent strategic deposition and management of the fine tailings. The degree of dewatering is highly dependent on the climatic conditions at the mine. Aggressive drainage can be invoked in fine tailing deposits by removing free water with perimeter ditching and decant structures. Once exposed to the atmosphere, the fine tailings may desiccate forming a crust that may be able to support reclamation materials (Carrier et al. 1987; Lahaie 2008). In northern climates, removal of surface water also exposes the fine tailings to freezing conditions. Yearly freezing/thawing and wetting/drying has been shown to cause moisture reduction and potential strength increases in the tailings. The increase in strength and solids content (C_w) has the potential to provide a surface layer capable of supporting re-vegetation and reclamation efforts (Stahl and Sego 1995; Beier et al. 2009). Dewatering and strength gain of these fine tailings deposits may also be enhanced by promoting evapotranspiration via suitable plant species (Carrier et al. 1987; Silva 1999).

Optimal sub-aerial deposition of fine tailings into an impoundment or in-pit may also lead to an increase in density and strength through desiccation and freeze/thaw consolidation. Deposition in arid climates or summer months will allow the tailings to desiccate during depositional cycles reducing the total volume of tailings (Burns et al. 1993; Qiu and Sego 2006). Thin layer deposition of fine tailings under freezing conditions may result in significant dewatering and strength gain (Dawson and Sego 1993; Proskin 1998; Dawson et al. 1999; Wells and Riley 2007). Benefits of volume reduction due to freeze/thaw may only be realized if the released water is managed and stored separately from the fines deposit. Both methods may result in higher tailings placement densities capable of supporting re-vegetation and reclamation efforts.

2.3 Tailings Dewatering

After extraction and separation, some mills will dewater the tailings slurry to recycle water and chemical reagents back to the mill. In cases where the tailings are not dewatered prior to deposition, the slurry may segregate (fine and coarse particles) upon deposition (Vick 1990; Blight 2010). Typically, additional processing is required to render tailings slurries non-segregating and improve the depositional behaviour of the tailings. Accelerated dewatering via mechanical (i.e. high density thickeners, paste or ultra-high density thickeners, high pressure filters, and centrifuges) or electrical methods and/or chemical additives may decrease the segregating potential of the tailings slurries resulting in a more homogeneous deposit. Alternatively, the operator may choose to dewater the coarse component of a tailings stream and separate the fine fraction from the coarse fraction. This separation may be achieved with a hydrocyclone.

The behavior of tailings slurries change significantly as water is removed. The curve on Figure 2.3 represents the relative strength (yield or shear) of tailings as the solids content increases or the extent of dewatering increases. Figure 2.3 is for comparative purposes only because the curvature and asymptote of the strength – solids content relationship varies among different tailings streams therefore the axes are dimensionless. This variation arises from the impact of particle size, shape, mineralogy, as well as pore fluid chemistry (Jewell and Fourie 2006).

2.3.1 Hydrocyclones

Hydrocyclones (cyclones) are utilized to dewater the coarse component of a tailings stream and to separate the fine fraction from the coarse fraction. Cyclone underflow from tailings streams can often be used as an alternative construction material for tailings embankments when suitable natural soils are not available. Reasonably clean sand can be produced from the underflow of hydrocyclones from most mill tailings with less than 60% fines (<75 μm) by weight (Vick 1990). The cycloned sand is typically produced at or very near the embankment thereby significantly reducing trucking costs. Embankments from cycloned sand typically have high effective strength and a reduced phreatic surface in their downstream zones.

A hydrocyclone is a continuously operating classifying device that functions on centrifugal separation principles to accelerate the settling rate of particles (Wills 2006). The feed slurry enters the hydrocyclone and fluid pressure creates a rotational motion forcing a vortex. The rotational flow imposes centrifugal forces onto the particles in the slurry and they migrate at varying rates, depending on size and density, to the outer wall. The coarse particles spiral downward and exit the apex as underflow. Water moves towards the centre of the hydrocyclone along with the finer particles whose particle size does not produce net outward motion. The slurry of water and fine particles leave the hydrocyclone through the vortex finder as overflow (Flintoff et al. 1987; Vick 1990; Wills 2006).

The performance of the hydrocyclone or "partition curve" (weight fraction of each particle size that reports to the underflow or overflow) is dependent on the dimensions of the cyclone, the feed solids content, the operating pressure, and the specific gravity of the feed slurry (Flintoff et al. 1987; Vick 1990).

According to Mittal and Morgenstern (1975) and Vick (1990) a desirable underflow would have less than 20% fines (preferably less than 12%, where fines are defined as geotechnical fines [<75μm]) in the sand to allow sufficient difference in saturated hydraulic conductivity between the sand and fine tailing overflow. The lower the percent fines in the underflow the faster water will drain from underflow sand. High water content in the sand embankment may lead to a high phreatic surface and instability. The percent solids of the sand should be close to 65% or better to reduce the amount of mechanical compaction required. At about 70 % solids the tailings should attain an angle of repose of 3:1 to 4:1 (Vick 1990). To meet the embankment raising (construction) schedule, large quantities of sand are needed from the hydrocyclone underflow. Therefore the mass rate of sand in the underflow should also be maximized. However, the amount of fines in the underflow and amount of sand recovered in the underflow are inversely related (Vick 1990; Wills 2006). There may have to be a compromise between quality and quantity of the underflow.

2.3.2 Thickeners

Unit operations for dewatering tailings slurries often utilize gravity sedimentation to separate the solid fraction from the liquid. Essentially, a slurry flows into a simple settling vessel whereby the solids settle under gravitational acceleration to the bottom and are removed as a thickened underflow while the released water is collected at the top of the vessel as overflow. The separation process may be classified as clarifying or thickening (Jewell and Fourie 2006). The goal of clarification is to produce a high quality overflow with minimal solids. Clarification is typically used to recycle water and process chemicals to the mill with no or little control on the underflow solids. Thickening, on the other hand, aims to deliver high solids content underflow. The resident time of the solids in a thickener is such that a bed of solids develops a structural network that in turn increases the solids content via compression (Usher and Scales 2005). The quality of the overflow is not a crucial factor during thickening.

Thickening is typically described by three phases as outlined in Figure 2.4 (Jewel and Fourie 2005; Usher and Scales 2005). In the first phase, particles undergo free settling whereby particle separation is sufficient where particle interaction is minimal. As the particle separation reduces, the particles undergo hindered settling. In this stage particles are constrained and settle as a mass. Finally, a structural network forms in the compression zone. The settling rate is limited by the structural network as well as the compressive force between particles and fluid above the particles. Thickeners can operate in batch or continuous mode. To increase the throughput or solids content of the underflow, designers can optimize the rate of sedimentation and compression by addition of chemical flocculants and modifying the shape of the thickener (Figure 2.5). As evident in Figure 2.3, conventional thickeners produce very low solids content underflow. With improved geometry and chemical addition high rate thickeners can increase the underflow solids significantly ($50 < C_w < 70\%$) and ultra-high rate thickeners can produce and underflow that can be classified as a paste ($70 < C_w < 85\%$). The efficiency of a thickener is dependent on the following factors (Jewel and Fourie 2006; Usher and Scales 2005):

- Feed solids content;
- Particle size distribution (PSD) and shape;
- Specific gravity of solids and fluid;
- Flocculant type and addition method;
- Temperature and viscosity of fluid;
- Raking or movement of particle bed.

The above properties will influence the thickener in two manners. During initial sedimentation, the up-flow velocity of the water must be less than the settling velocity of the solids particles to achieve a clear supernatant. The critical velocity is called the rise rate and is used to size the diameter of the thickener. The final underflow density is dependent on the residence time of the solids bed within the thickener. The bed residence time is dependent on slurry properties, flocculant addition, geometry and raking efficiency. In most high rate and paste thickeners the hindered settling zone is significantly smaller than the free settling zone and compression zone and is therefore ignored.

In the oil sands industry, thickeners are generally used to thicken cyclone overflow tailings to create underflow solids contents ranging from 6% to 40% (Jeeravipoolvarn 2010). They have also been investigated to thicken MFT and other finer grained tailings streams originating from the extraction process such as froth tailings or floatation tailings (Yuan and Lahaie 2009; Shaw et al. 2010; Longo et al. 2011; Boxill and Hooshiar, 2012). From the numerous laboratory, bench and pilot scale tests conducted on fine grained oil sands tailings streams, the under-flow solids content varies considerably and is influenced by the factors described previously (such as PSD and feed C_w). However, the testing does show there is a potential to produce a paste like underflow with solids contents from 40 to >60 % (Lord and Liu 1998; Yuan and Lahaie 2009; Shaw et al. 2010; Longo et al. 2011).

2.3.3 In-line Thickening

An interesting development in the oil sands industry is the in-line thickening tailings (ILTT) concept. Fine tailings may be dewatered through a combination of chemical addition and strategic deposition into constructed containment cells. A large scale pilot and commercial demonstrations of the ILTT process are currently underway at Syncrude (Jeeravipoolarn, 2010; Wilson et al. 2011; Fair and Beier 2012), Suncor (Wells and Riley 2007; and Wells et al. 2011), and Shell (Kolstad et al. 2012). The process negates the need for constructed thickeners as described in the previous section because the deposition cell essentially becomes the "thickener vessel". Polymer solutions are injected directly into the transfer pipeline ("in-line") containing the fine tailings and mixing occurs en route to the deposition point. Substantial instantaneous dewatering occurs upon deposition (as surface runoff) of the 'in-line" amended tailings slurry as a result of aggregation of fine particles into flocs. This aggregation process enhances the saturated hydraulic conductivity and water release properties of the tailings. Subsequently, additional dewatering is achieved via a combination of settlement, seepage and environmental dewatering (desiccation, freeze/thaw). The process described above is illustrated in Figure 2.6 and Figure 2.7. The ILTT process aims to manipulate the fine tailings floc structure to maximize water release and to balance the rheology (yield strength and viscosity) to allow the tailings to flow upon deposition and stack subsequent layers.

The dewatering rate and rheology modifications achieved during the ILTT process depend on the feed tailings material properties (mineralogy, clay content, PSD, and pore fluid chemistry), polymer type and dosage, injection type, and applied shear during mixing, transport and deposition (Wells et al. 2011). At Suncor, three samples of fine tailings taken from two different ponds, at different depths, with similar water chemistries reported a similar optimum floc dosage of approximately 1700 g per dry tonne of clay (Wells et al. 2011). Their work suggests the floc dosage is a function of clay content and not solids content or slurry density. The impact of polymer injection, mixing and shear conditions is depicted on Figure 2.8. There are four stages evident during the ILTT shearing processes (mixing, transport, and deposition) (Wells et al. 2011).

1. Floc assemblage: the polymer is dispersing within the slurry. There is a rapid increase in yield strength as the polymer interacts with solid mineral phase. The dewatering rate is low.
2. Floc re-arrangement: floc formation is at a maximum and is balanced by floc breakdown. A gel state forms where the maximum yield stress plateaus with continued shear. Dewatering rate is increasing as the flocs assemble into a network.
3. Floc breakdown and dewatering: the yield strength decreases with continued shear because the flocs and network are breaking down. Water is released during breakage of the flocs (a maximum rate).
4. Over shear zone: Further shear leads to complete breakdown of flocs, loss of yield strength and materials reverts to it's original state.

As can be seen in Figure 2.8, insufficient shear will lead to a high "strength material" with low dewatering potential. This material will hold onto water and may not flow sufficiently from the deposition point. Too much shear breaks apart the flocs and leads to a mixture with dewatering properties similar to the feed tailings. The optimal shearing conditions occurs in stage three. Here the yield stress is sufficient to allow the material to flow and still permit subsequent stacking of layers while the dewatering rate is at a maximum. Using an optimized ILTT process, Wells et al. (2011) report that 25 percent of the feed slurry water can be released within 24 hours. The remaining water is only removed through downward seepage, drainage along the cracks, evaporation, and freeze-thaw. At 20 cm thick lifts, these deposits would dewater to 80% solids by mass within 6 days, allowing for subsequent layers to be placed. Operational experience at Shell suggest thin lifts of ILTT (<30 cm) may require at least 20 days to meet strength targets (Kolstad et al. 2012). Wilson et al. (2011) report at least 21 days of evaporation and drainage were required to dry an ILTT layer at Syncrude. Given the interdependence of several variables that affect the performance of the ILTT process, the oil sands industry is still working towards developing a consistent and readily predictable ILTT process.

2.3.4 Centrifuge

The previous thickening processes relied on gravitational force to effect the separation of water from the solid phase. There are instances where greater rates of separation are required or separation is impractical to achieve due to the size and residence time needed for the settlement vessels using gravitational forces alone. Centrifugal force may be utilized in these situations because it can induce particle accelerations that are several thousand times that of gravity. Centrifugal forces can be generated by two different methods (Richardson et al. 2002):

1. Introduce a fluid tangentially into a cylindrical or conical vessel, such as a hydrocyclone. The heavier particles collect near the outer walls of the vessel and lighter particles and fluid accumulate near the central axis.
2. In a centrifuge device, the fluid is introduced into a rotating vessel and is rapidly accelerated. Frictional drag causes the fluid to rotate with the vessel. Heavier particles accumulate at the walls of the vessel and lighter fluid and particles are forced to the central axis. With time, a zone of particle free water develops at the central axis (centrate) (Figure 2.9). As particles move towards the outer wall, two distinct zones develop. A settling zone where particles are not in contact with each other and a compression zone where particle to particle contact occur (cake).

There are two main applications of centrifuges for the purpose of tailings dewatering, decanting or filtering. A summary of various types of centrifuges is included in Figure 2.10. Decanting centrifuges can be used to separate particles from a slurry due to a difference in size or density because centrifugal forces produce higher rates of sedimentation than gravitational forces. Centrifugal forces can also be used to replace the applied pressure difference during the filtration of a slurry. In filtering centrifuges, the solids are retained on the filter medium as a cake while the fluid passes through the cake and filter (Norton and Wilkie 2004). Both centrifuge processes can operate on batch or continuous mode. The application of the various centrifuge types depends on the characteristics of the slurry and operational constraints such as the required throughput and dewatering rate (Norton and Wilkie 2004). The specific gravity of the solids significantly impacts the performance of centrifuges. If the solids and liquids have similar specific gravities, separation will be difficult. For slurries with a wide range of PSD, the recovery of solids is dominated by particles that are less than 100 micron. Slurries with PSD greater than 100 micron are best suited for filtering centrifuge applications. Filtering centrifuges can produce the "driest" solids, however, they typically loose up to five percent of solids through the filter. Where centrate clarity is of concern, decanting centrifuges can provide the best clarity.

Decanter centrifuges are under investigation for their applicability to oil sand fine tailings management at the Syncrude mine (Ahmed et al. 2009; Fair 2012). Fine tailings streams with solid contents of 30 % are treated with polymers to capture the fine clay particles and then fed to decanter centrifuges (Mundy and Madsen, 2009). The process can dewater the fine tailings to 50 - 55 % solids content and produce a clear centrate with less than 0.5 % of solids. The resulting cake is then deposited into containment cells (either in pit or above ground) for further dewatering due to consolidation, evaporation, and freeze-thaw.

A filtering centrifuge was also tested by Nik et al. (2008) to dewater MFT through bench and meso scale experiments. The dewatered MFT was intended to be mixed with sand to create a higher solids content NST. MFT at 36% solids was mixed with various concentrations of gypsum and fed into a batch style filtering centrifuge with a filter pore size of 0.8 microns. Nik et al. (2008) produced centrifuge cake at solids contents ranging from 43 % to 65 %. They also found that untreated MFT had a tendency to segregate during centrifuging.

2.3.5 Filtration

Solids may also be separated from a suspension with the use of a porous medium in the process of filtration. The porous medium traps the solid particles and allows the liquid or "filtrate" to pass through with the assistance of pressure or vacuum. This form of filtration is often referred to as "dead-end" filtration because the slurry feed and driving force (pressure or vacuum) are perpendicular to the filter medium. Typically, the pore size of the filter medium is larger than the solids, therefore the initial layers of solids that are trapped in or on the filter do the bulk of the filtration. The retained solids are referred to as "cake". A schematic of a typical "dead-end" filter is depicted as Figure 2.11. As the cake builds up in thickness above the filter medium, the resistance to flow progressively increases and the filtrate rate decreases. Once the desired cake moisture content is reached, the cake is removed from the filter medium. (Richardson et al. 2002; Jewell and Fourier 2006).

The filter medium is essentially a support layer for the cake. Various types of filter medium are used including polymer based membranes and woven fabrics, woven steel fabrics, and perforated or porous materials such as sintered metal or ceramic media (Sparks 2011). A filter medium must be mechanically strong such that it can withstand the applied pressure or vacuum and cake removal process. It must also be resistant to the chemicals in the suspension and offer little resistance to flow of the filtrate (Richardson et al. 2002).

The properties of the feed slurry such as solids content, PSD, particle shape, and mineralogy will influence the characteristics of the filter cake, which ultimately impact the filtration process. The solids may form filter cakes that are compressible or incompressible depending on the material properties outlined above. Compressible cakes are influenced by changes in pressure or filtrate rate. Increasing the pressure or filtrate rate may compact a compressible cake and lead to a high resistance to flow. Incompressible filter cakes are not sensitive to pressure or filtrate rate changes. Regardless of the filtration pressure, the filtration rate will be at a maximum during the initial stages of filtration as the filter cake develops. However, high initial rates may lead to clogging of the filter medium, thus producing a high resistance to flow.

The development of large capacity pressure and vacuum filter technologies contributed to the adoption of filtration in mining operations. There are a number of different variations and styles of filtration equipment, but they generally fall into one of five classifications: drum (vacuum), disc (vacuum), belt (vacuum), plate and frame (pressure), and belt presses (pressure) (Jewell and Fourier 2006). Filters can generally produce cake with lower moisture contents than underflow from thickeners (Figure 2.3). Depending on the feed slurry properties and chosen filtration equipment, the filter cake may be sufficiently dry that conveyors or trucks are required for transport to the deposition area. The cake may then be spread and optionally compacted to form an unsaturated, dense tailings stack ("dry-stack"). (Davies and Rice 2001).

Although it has yet to be implemented commercially, filtration is still deemed a viable technology option for oil sand tailings. In the recent Tailings Technology Roadmap and Action Plan study, it was identified as a "highlighted technology" among the hundreds of technology options for managing oil sand tailings (Sobkowicz 2012). Filtration offers a potential technology to fulfill regulatory and public pressure to reduce the footprint of tailings impoundments, increase water recycle, and achieve a dry tailings deposit. Bench scale experiments were conducted by Xu et al. (2008) to investigate the feasibility of producing dry stack tailings by filtering flocculated whole oil sand tailings (coarse and fine fractions from the extraction plant). Their study investigated the impact of filtration pressure, feed slurry C_w and fines content (less than 44 micron), and flocculant dose. For tailings from an average grade ore, with optimal flocculation, the resistance to filtration was sufficiently low that filtration was possible (Xu et al. 2008). However, the calculated throughput rate that could be achieved in a commercial application (5 tonne/(m^2 hr)) would require a filtration surface of approximately 1500 m^2.

Due to the build up of cake on the filter medium in dead end filtration, the filtration rate reduces with time and slows the dewatering process. Therefore, conventional filtration methods are often difficult and expensive to operate due to cake build up and maintenance of the equipment. Crossflow filtration (CFF) as a dewatering method can offer improvements over dead end filtration. Crossflow filtration is a pressure driven filtration process that can be used for dewatering slurries of fine particles. It is typically used with microporous membranes in the size range of 0.02 to 20 μm (Ripperger and Altmann 2002). In CFF, the slurry would flow parallel to the filter material. A filter cake will develop on the pipe surface. However, due to shear of the flowing slurry, the build up of cake will reach an equilibrium thus maintaining a relatively constant filtration rate (Richardson et al. 2002) (Figure 2.12). Richardson et al. (2002) report in many cases it may not be possible to reach a steady rate of filtration due to cake formation dynamics. Depending on the degree of scour and erosion, layers deposited during CFF can themselves become dynamically formed membranes.

Variations of CFF have been attempted previously to dewater mine waste slurries. Farrnand and Sawatzky (1988) used a laboratory scale crossflow microfiltration apparatus to dewater a sample of de-oiled oil sands tailings sludge from 6 % to 37.5 % C_w. The process suffered from low production rates and high costs for the proposed set-up. Crossflow filtration was also used to dewater gold slimes using a braided steel hose (Yan et al. 2003). A bench scale experiment using a 100 m long braided steel hose as the filter medium dewatered gold slime tailings from 44% to 53% C_w. Their process was trialed in the field and was only able to achieve a 1.2% increase in solids content over 24 m. Beier and Sego (2008) and Zhang et al. (2009) investigated the potential of CFF to dewater oil sand whole tailings. Based on the experimental results, approximately 450 m of 50 mm diameter porous pipe would be required to dewater a total tailings stream at a feed flow rate of 2.26 L/s from 55 % to approximately 70 % C_w.

2.4 TAILINGS DEPOSITION

Central to the tailings management plan is the deposition technique. Several options for deposition are available including sub-aerial (above fluid surface), subaqueous (below fluid surface), thickened/paste discharge, dry stacking, and co-disposal. The behaviour of the tailings upon deposition is influenced significantly by the chosen depositional method. Conventional tailings disposal practices utilize hydraulic deposition techniques along or near the external impoundment dyke crest. Tailings that have been thickened significantly prior to deposition are typically discharged from a central location or from several risers leading to the formation of a conical shape (Jewell and Fourie 2006; Simms et al 2011; and Theriault et al 2003). If the tailings are dewatered to a higher degree, dry staking using conveyors or trucks are utilized to transport and deposit the tailings. The most applicable method of deposition that will fulfill the objective of the tailings management plan depends on many factors including (Vick 1990; Dixon-Hardy and Engels 2007):

- tailings properties and engineering behaviour;
- proposed impoundment type (i.e. surface storage, in-pit, underground backfill);
- tailings treatment (i.e. thickening)
- practical use of tailings (i.e. construction material);
- reclamation requirements;
- water requirements (need for recycle); and
- climate.

2.4.1 Sub-aerial Deposition

Discharging of tailings slurries in thin layers on a beach or dyke surface above the water level is referred to as sub-aerial deposition or beach above water (BAW) (Qiu and Sego 1998 and 2006). This depositional method is typically used in conventional surface storage impoundments. Tailings are deposited above the decant level from several spigots or single discharge points onto the beach (Figure

2.13). Solid particles settle out of the slurry as the tailings flows down the beach and looses energy. Following deposition, the density of the settled particles can increase due to drainage and evaporation (depending on climate). Released water is collected in the low area (or pond) and is either recycled, transferred to alternative storage facilities, left to evaporate in place, or treated and released. Since mine operations continually produce tailings, a typical impoundment may be divided into several deposition areas to allow continuous operation and promote drainage and evaporation. Depending on the material properties of the tailings slurry, tailings deposited sub-aerially may be used as construction material for impounding dykes. Segregation may occur during sub-aerial deposition resulting in coarser (or denser) particles concentrating near the deposition point or beach zone and finer particles flowing towards or into the pond (Figure 2.13).

2.4.2 Sub Aqueous

Tailings with the potential to oxidize and produce AMD (acid mine drainage) may require subaqueous (below water) disposal to limit the available oxygen in contact with the sulphidic tails (MEND 2001). Depositing tailings below the water surface may create steep deposit slopes (as compared with sub-aerial deposition) which may result in slumping and differential settlement of the tailings deposit. In turn, slumping and excessive settlement may lead to damage of impoundment and liner. To ensure and even distribution of tailings, deposition generally occurs from a moveable floating barge with regular surveys conducted to establish the tailings surface profiles (Engels 2006).

2.4.3 Thickened and Paste Tailings

If tailings are dewatered (thickened) sufficiently, they may form a more homogeneous, non-segregating deposit upon deposition. For reference, a few case studies of thickened tailings disposal operations are presented by Barbour et al (1993), Jewell and Fourie (2006) and Theriault et al (2003). Such thickened tailings are deposited layer upon layer forming a “stack” or self-supporting pile. Thickened tailings can generally achieve greater slopes, thus reducing the required footprint of the impoundment. Deposition generally occurs from topographical high points or from constructed towers within the impoundments (Jewel and Fourie 2006; Engels 2007). Water released after deposition and surface runoff can be collected at the perimeter of piles or toe of the slope. There is a high degree of flexibility when designing the geometry of the thickened tailings deposit. Impoundments may have one central discharge location resulting in a single large conical pile or several risers/discharge points can be used creating a series of smaller piles (Figure 2.14; Jewell and Fourie 2006).

Paste tailings are dewatered to the point (Figure 2.3) where little to no bleed water occurs during deposition (Bussiere 2007). Due to the high viscosity, positive displacement pumps are required to transport tailings paste to the deposition point, limiting the economical distance they can be transported (Engels 2007). Similar to thickened tailings, paste will form a typically concave, conical pile upon deposition (Fitton 2007).

2.4.4 Dry Stack

When tailings are dewatered to moisture contents lower than paste tailings they are no longer transportable by pipeline and must be transported by conveyor or truck. Upon deposition the material may require spreading and compaction for stability and form a "dry stack". Often, no retention structure is required for dry stacking, however, some infrastructure may be needed for surface runoff management. Dry stacking is often used where water conservation is critical, in high seismic areas, or where high recovery of dissolved process chemicals is required. Compared with the other depositional and storage techniques, dry stacking offers a smaller footprint and may be easier to reclaim. However, they have a greater potential for fugitive dust (Davies and Rice 2001).

2.4.5 Co-Disposal

Co-disposal is the simultaneous (co-mixing) or alternating (layering co-disposal and waste rock inclusion) deposition of fine grained tailings and coarse waste into a single storage area (Bussiere 2007). Co-mixing refers to the mixing/blending of the two waste streams prior to deposition. Layering co-disposal is achieved when alternating layers of coarse and fine grained waste are deposited in the same storage facility. Waste rock may also be placed independently into a storage facility (the waste rock may form internal dykes) to provide stability and drainage layers (waste rock inclusion). By mixing the two waste streams, the properties of the resulting deposit can be optimized such that they are an improvement over the properties of each stream separately (Bussiere 2007). Additionally, where AMD is a concern, the addition of fine grained waste to coarse waste rock can reduce the air and water permeability and increase the water retention properties (Wilson et al. 2002; Bussiere 2007). This can reduce the oxygen supply to only diffusion, thus decreasing the AMD potential. One of the challenges with co-disposal is controlling the blending and deposition strategy to ensure a consistent mixture is created with predictable behavior.

The coarse waste stream does not need to be waste rock. In the oil sands, fine grained tailings streams are co-mixed with sand to produce a non-segregating mixture (CT or NST). The fines are trapped within the void space of the coarse sand grains and the resulting deposit allows earlier reclamation (Matthews et al. 2002; Chu et al. 2008). Mixing of MFT with dry, clay-shale overburden has also been investigated (Lord et al. 1991 and 1993; Lord and Issac 1989). The clay shale exhibits a wide range of water contents between the plastic and liquid limits, therefore it can absorb water and still remain plastic (Lord et al. 1991). By mixing the MFT with the clay shale, water can be absorbed into the clay shale, resulting in a soft clay deposit, an improvement over the low solids content slurry of MFT.

2.4.6 Segregation of Tailings Slurry

When tailings are hydraulically deposited there may be a tendency for the slurry to segregate, forming a concentration gradient along the deposit (Kupper 1991; Mihiretu et al. 2008; Blight 2010). The degree of segregation depends on particle gradation and type, solids content, rheology of the tailings fluid, and flow conditions (Kupper, 1991; Fitton, 2007). Segregation causes hydraulic sorting of the solid particles, with coarse/heavier particles deposited near the deposition point and finer particles further along the flow path. Deposits from segregating slurries tend to be relatively dense with pronounced grain arrangements (Kupper 1991). Sorting is more pronounced for higher flow rates, lower slurry solids contents and small flow velocities on the beach (Kupper, 1991). Slurries with high solids contents and that contain very fine particles tend to behave as non-segregating materials. Non-segregating slurries create massive deposits with solids uniformly distributed throughout and which lack stratigraphic features.

Under constant flow conditions, the slurry properties of particular tailings stream depend on the relative amounts of solids and water (solids content) (Kupper 1991). A further distinction between the coarse fraction and fine fraction must be made since the fine fraction can impact the rheology of the slurry. Azam and Scott (2005) recognized this fact and presented a diagram to understand the slurry behavior based on the proportions of sand, fines, and water (Figure 2.15). Several boundaries can be identified on the diagram such as the sedimentation/consolidation boundary, segregation boundary, limit of pumping and liquid/solid boundary. These boundaries are unique for a particular tailings and were developed for oil sand tailings. As evident in the diagram, tailings can be rendered non-segregating by increasing the solids content or by adding chemical reagents which alters the rheology and thus shifts the segregation boundary.

Blight et al. (1985) proposed the following empirical relationship to predict the mean particle size (D_{50H}) due to hydraulic sorting at any point (H) along a tailings beach of length X:

[2.1] $$\frac{D_{50H}}{D_{50}} = e^{-B_o H/X}$$

Where D_{50} is the mean particle size of the feed tailings and B_o is a parameter based on the characteristic of the tailings. As evident from the equation, the parameter B_o must be determined from prior beaching and is only applicable to a particular tailings slurry.

In an effort to evaluate tailings deposition behavior and hydraulic sorting, Sisson et al. (2012) used a combination of rheological characterization, shear cell testing, pilot flume testing, and analytical modeling to develop a segregation model. The model was applied to several discharge conditions and disposal area configurations. They found high quality tailings slurries (higher solids contents, lower fines contents) and low energy depositional methods tended to decrease the amount of segregation and increase the fines retained within the deposit. The modeling procedure in its current form is specific to the tailings tested and requires significant tailings characterization to implement.

2.4.7 Beach Slope Prediction

The behavior of a slurry upon deposition (segregating vs. non segregating) has a significant effect on the geometry of the deposit. Segregating slurries tend to form shallow slopes with beaches that are concave. Whereas non-segregating slurries tend to form steeper slopes and exhibit convex profiles. This behavior change is represented on **Error! Reference source not found.** where Fitton (2007) interpreted Robinsky's (1978) data depicting the slope characteristic of segregating vs. non segregating tailings deposits. For a particular tailings, the transition from segregating to non-segregating behavior depends on the factors detailed above. Understanding and predicting the beach profile/slope is important because it allows the planner to predict the location of the pond, determine the available tailings storage capacity and available freeboard (Blight et al., 1985).

To estimate the slope (i, %) of sub-aerially deposited tailings beaches (including segregating slurries), Kupper (1991) proposed the following empirical relationship:

[2.2] $$i = 5\left(\frac{A_p(g*(G-1)*D_{50})^{0.5}C_w}{Q}\right)^{0.5}$$

Where D_{50} is the mean particle size, Q is the total flow rate at the discharge point, A_p is the area of the discharge pipe, g is the acceleration of gravity, G is the ratio of the specific weight of the grains to the specific weight of the water. Kupper

(1991) states this equation is valid for beaches up to a few hundred metres long and works well in practice for segregating slurries where discharge parameters don't vary significantly.

Blight et al. (1985) recognized that the profiles of a series of beaches from segregating hydraulic fills can be represented by a single master profile (Figure 2.17; Equation 2.3):

$$[2.3] \qquad \frac{h_{aw}}{Y} = \left(1 - \frac{H}{X}\right)^{n_o}$$

Where h_{aw} is the elevation above the pond level, Y is the discharge elevation above the pond level, H is the distance along the beach and X is length of the beach from the deposition point to the pool edge, and n_o is a dimensionless constant unique to a particular tailings. The exponent n_o was shown to be influenced by solids content, the grain size distribution, and specific gravity of the solids. Blight et al. (1985) attributed the concavity of the profile to sorting along the beach, with flatter slopes as the material becomes finer further from the deposition point. However, Fitton (2007) argues the concavity may also arise from variability in the tailings feed. With all else equal, changes in solids content, or flow characteristics will invariable impact the deposit slope of a tailings slurry. Fitton (2007) illustrates this point by numerically simulating the deposition of multiple layers of tailings at different solids contents or discharge parameters. He demonstrated that tailings, which would form shallow slopes, run out along the beach and tailings forming steep slopes, deposit near the deposition point.

For non-segregating tailings deposits there are several empirical and semi empirical and theoretical approaches developed to estimate the overall slope of a non-segregating tailings deposit (Simms et al. 2011). Based on a large database of field tailings deposits, large flume tests, and natural analogies (deltas), Fitton (2007) proposed the following empirical equation to estimate the slope of a thickened tailings deposit:

[2.4] $$i = \frac{26.6 * {C_w}^2}{\sqrt{Q}}$$

Where Q is the flow rate from a spigot (L/s). This method offers a potential to provide a "preliminary" estimate slope based on solids content and flow rate. With site specific data, the formulation could be calibrated improving the predictive capability.

According to Simms et al. (2011) the following three methods are promising and have shown partial success in predicting or modeling field data. They each involve several in-depth steps and calculations; therefore they will not be described in detail. The reader is referred to Simms et al. (2011) for further information on each method.

Fitton (2007) also developed a semi empirical procedure that integrates non-newtonian open channel flow and sediment transport theories. The maximum beach slope is assumed to occur when an equilibrium is reached between erosion and sedimentation in channelized tailings flows. The method requires calculating or determining experimentally the minimum transport velocity for the tailings and the rheology of the tailings slurry.

An alternative method developed by McPhail (2008) assumes the slope profile is based on the dissipation of energy as the tailings move downslope in a channel. The energy in this method is referred to as stream power. To utilize this method, one must first determine the initial stream power of the beach OR the height of the deposit OR length of the deposit and also the initial slope (Simms et al. 2011). Application of this method also requires small scale field trials to determine fitting parameters.

A final method presented by Simms et al. (2011) is based on lubrication theory. The procedure allows the Navier-Stokes equation to be reduced to relatively simple equations for equilibrium profiles of non-newtonian fluids. The method assumes that the layer thickness to overall length is small and the ratio of inertial forces to gravitational and viscous forces are also small. The theory is valid for non-newtonian fluids that spread under their own weight. This method works well to represent tailings deposits in flumes, however, if channelized flow develops the lubrication theory fails and is unable to predict the slope.

As with segregating slurries, there are also differing opinions on the cause of the typical concave profile exhibited by non-segregating (i.e. thickened) tailings stacks (Simms et al. 2011). Fitton (2007) attributes this concave shape to the variability in tailings feed properties and discharge conditions. Other methods directly simulate a concave profile and believe the concavity occurs even with constant tailings properties (Simms et al. 2011).

2.5 NATURAL DEWATERING POST DEPOSITION

Once deposited, tailings undergo dewatering through particulate settling, sedimentation and consolidation. These settling processes have been described by Imai (1981) with three stages as shown in Figure 2.18. During the first stage no settling takes place and the suspended particles flocculate. The newly formed flocs gradually settle in the second stage and start to form a layer of sediment. This sediment undergoes consolidation (reduction in water content). The interface between the settling zone and the lower sediment is the birth place of new sediment. With time, the settling zone becomes thinner and eventually vanishes. The last stage occurs once all of the flocs have transformed into sediment and then undergo self-weight consolidation. Under the right circumstances, environmental processes such as evaporation, freeze-thaw, and evapotranspiration may also enhance dewatering of fine tailings. The following section will outline these processes related to dewatering tailings post deposition.

2.5.1 Sedimentation

Particles dispersed in a slurry tend to settle under gravity. When the solids content of the slurry is very low, and the solids are non-surface active, the particles can settle freely and do not interact or hinder each other. This condition is referred to as particulate settling or clarification and can be predicted by a using the formula presented by Stokes (Jeeravipoolvarn 2010).

[2.5] $$V_s = \frac{2}{9}\frac{r^2 g(\rho_p - \rho_w)}{\mu}$$

Where V_s is particle settling velocity, r is a Stokes particle radius, g is gravitational acceleration, ρ_p is particle density, ρ_w is fluid density and μ is fluid viscosity.

If the settling particles are surface active or clay sized, the simple Stokesian model is not sufficient to predict the settlement. The downward motion of particles of this type is referred to as "hindered sedimentation". Kynch (1952) developed a theory for sedimentation which assumes the speed of the falling particles (V_s) is a function of the suspension concentration (c_1). He only considered the continuity of the solid phase and also ignored effective stress in the sediment layer. The governing equation for the Kynch theory is as follows (Kynch, 1952):

[2.6] $$\frac{\partial c_1}{\partial t} + \frac{d}{dc_1}[c_1 V_s(c_1)]\frac{\partial c_1}{\partial x_t} = 0$$

where $V_s(c_1)$ is the velocity flux of particles, t is the time and x_1 is the elevation above a datum. The formulation can be solved by method of characteristics for hyperbolic partial differential equations.

2.5.2 Consolidation

The process where soils decrease in volume due to an applied load is called consolidation. The applied load results in the generation of excess pore water pressure. Over time, this excess pressure dissipates, resulting in a volume change or settlement. During consolidation, effective stresses control the deformation of the soil. The fundamental relationships governing the response of a soil to loads, based on the stresses and displacements, were first developed by Terzahgi in 1923 (Schiffman et al. 1969). The theory is based on fluid flow, a continuity equation and the principle of effective stress. The theory also assumes the soil is completely water saturated, strains and stress increments are small, the soil is homogeneous and its properties do not vary with stress and strain. The one-dimensional governing equation for consolidation is mathematically defined as:

[2.7] $$C_v \frac{\partial^2 u}{\partial z^2} = \frac{\partial u}{\partial t}$$

and

[2.8] $$C_v = \frac{k}{m_v \gamma_w}$$

Where C_v is the coefficient of consolidation, u is the excess pore pressure, t is time, and z is the depth, k is the saturated hydraulic conductivity, γ_w is the unit weight of water, and m_v is the coefficient of volume compressibility. Terzaghi's one dimensional theory is widely used in geotechnical engineering for settlement calculations. However, for soft materials such as fine tailings, the theory is not sufficient for consolidation predictions and tends to over predict the settlement times and predicts slower dissipation of excess pore pressure (Bromwell 1984). The assumption of linear material properties and small strains do not apply to soft materials like tailings. Tailings can have highly non-linear compressibility and hydraulic conductivity relationships and will undergo significant settlements.

These shortcomings lead to the development of finite strain consolidation theory which has the ability to handle large strains and non-linearity. Gibson et al. (1967) developed a one-dimensional non-linear finite strain theory for soft fine grained soils and slurries. Their theory also considers the influence of self-weight on the consolidating layer, unlike the conventional theory. In terms of void ratio, the governing equation presented by Gibson et al. (1967) is:

[2.9] $$\pm\left(\frac{\rho_s}{\rho_w} - 1\right)\frac{d}{de}\left[\frac{k(e)}{1+e}\right]\frac{\partial e}{\partial z} + \frac{\partial}{\partial z}\left[\frac{k(e)}{\rho_w(1+e)}\frac{d\sigma\prime}{de}\frac{\partial e}{\partial z}\right] + \frac{\partial e}{\partial t} = 0$$

Where ρ_s is solids density, ρ_w is fluid density, e is void ratio, k is saturated hydraulic conductivity, σ' is effective stress, t is time, and z is a material coordinate.

The theory can also be cast in terms of excess pore pressure (Somogyi 1980) or porosity (Lee 1979). The formulations are fundamentally identical to Gibson et al. (1967) but require different initial and boundary conditions to solve.

The normal coordinate system employed in geotechnical engineering is the Eulerian system. It follows that an element is fixed in space and the medium moves within that element. This results in both solids and liquids changing within the element. When large strains occur, the thickness of soil layer (i.e. top boundary) also changes. The moving boundary is therefore time dependent, and inconvenient to model. Utilizing a Lagrangian coordinate system can overcome this shortfall. In a Langrangian system, an element of mass always encloses the same material, thus the vertical coordinate is fixed. Only fluid changes within the element. Recast in Langrangian coordinates, the Gibson et al. (1967) is as follows (Shiffman et al. 1988):

[2.10] $$\frac{\partial}{\partial a}\left[\frac{k(e)}{\gamma_w}\frac{1+e_0}{(1+e)}\frac{\partial u}{\partial a}\right] = \frac{a_v(e)}{1+e_0}\left[\frac{\partial u}{\partial t} - \frac{\partial \sigma}{\partial t}\right]$$

Where e_0 is the initial void ratio, and a is the material coordinate.

To model the finite strain consolidation equations, two key relationships are required: void ratio-effective stress and hydraulic conductivity-void ratio (Jeeravipoolvarn 2010). The behavior of these relationships depends on the properties of the tailings solids and the chemistry of the pore fluid. Typical curves for various tailings slurries are presented in Figure 2.19 (void ratio-effective stress) and Figure 2.20 (saturated hydraulic conductivity-void ratio).

Continuous functions of each relationship are required in order to solve the governing equations. Jeeravipoolvarn (2010) states the functions are determined from experimental data and mathematical formulas are often used to represent the compressibility and saturated hydraulic conductivity. The most common is a power law formulation where A, B, C and D are experimentally derived parameters:

[2.11] $e = A\sigma'^B$

[2.12] $k = Ce^D$

Other equation are available, however for detailed consolidation modeling, the constitutive relationships should be determined from direct experimental measurements. They should also accurately represent the range of void ratios that are expected in the field.

2.5.2.1 Consolidation Models

Most models for predicting settlement and consolidation of tailings are based on Gibson et al. (1967) method. They employ a finite difference scheme to numerically solve the governing equations. Both explicit and implicit schemes have been utilized. Jeeravipoolvarn (2010) provides a review of these numerical approaches and the reader is referred to his thesis for more detail. He also states that both finite difference approaches (explicit and implicit) generate nearly identical predictions provided proper spatial and temporal variations are selected.

Pollock (1988) developed a commercially available consolidation program based on the Gibson et al (1967) method, FSConsol. The model was found to successfully predict interface settlement of oil sands CT deposits (Pollock 2004). However, the model was not able to predict the behavior of MFT likely due to creep and required model and/or parameter adjustments (Pollock 2004). Therefore, the FSConsol program can be reliably capture the behavior of CT and other oil sand thickened tailings deposits (Jeeravipoolvarn 2010).

2.5.2.2 Sedimentation – Consolidation Modeling

The interface between sedimentation and consolidation is not well understood and the void ratio at which effective stresses originate has been shown to be dependent on the initial void ratio (Imai 1981). Schiffman et al (1988) also recognized that in industrial operations such as tailings disposal, sedimentation and consolidation occur simultaneously. Been (1980) and later Pane and Schiffman (1985) considered the only difference between Kynch's theory and Gibson's theory was effective stress and the coordinate systems. Pane and Schiffman (1985) presented a formulation to link the sedimentation and consolidation processes with the effective stress equation and an interaction coefficient, β :

$$[2.13] \qquad \sigma = \beta(e)\sigma' + u_w$$

Where β(e) is the interaction coefficient that is a function of void ratio and u_w is the pore pressure. The theory proposes that above a void ratio threshold (β(e) = 0), the mixture behaves as suspension, effective stress is gone and the finite strain equations reduce to Kynch's theory.

A one dimensional numerical model of coupled sedimentation/consolidation was proposed by Masala (1998). He derived the model from Kynch's theory and finite strain consolidation. The model utilizes saturated hydraulic conductivity as a hydrodynamic interaction for the solid fluid interactions in the suspension. Masala (1998) used the model to predict experimental results provided by Toorman (1999) and the sedimentation of oil sand fines tailings. Reasonable agreement was found between the model results and the experimental data set.

Jeeravipoolvarn (2010) also developed a finite strain sedimentation/consolidation model. He used the finite strain theory of Gisbon et al. (1967) and Pane and Schiffman's (1985) interaction coefficient as the basis for the numerical model. The model provided satisfactory predictions for oil sands in line thickened tailings and experimental data from the literature.

2.5.2.3 Thixotropy

Thixotropy is defined by Mitchell and Soga (2005) "as an isothermal reversible time-dependent process occurring under conditions of constant composition and volume whereby material stiffens while at rest". The material is also softened or liquefied during remolding. With time, the build-up of bond strength between the particles may prevent a soil from compressing or releasing water (Mitchell 1960). Suthaker and Scott (1997) showed that oil sand fine tailings exhibit a high thixotropic strength gain when compared to typical clays. It is this thixotropy that is thought be the cause of the apparent over-consolidation of oil sand fine tailings (Jeeravipoolvarn 2010). Compressibility curves that are based on conventional large-strain consolidation tests may not capture this time-dependent effect and therefore may overestimate the consolidation.

2.5.2.4 Creep

Another time dependent phenomenon that can impact the compressibility of tailings is creep. Creep is the time dependent reduction of soil volume under constant vertical effective stress (Jeeravipolvarn 2010). Jeeravipoolvarn et al (2009) and Kabwe et al (2013) presented evidence that oil sand fine tailings compress without a significant increase in effective stress. Conventional consolidation theory may not be suitable to represent this time-dependent process. A Soft-Soil-Creep model developed Leoni et al (2008) was applied by Vermeer et al (2015) to model the complex consolidation and creep phenomena in oil sands MFT. Beuth et al (2014) further extended the Soft-Soil-Creep model in an effort to assess the feasibility sand capping ultra-soft mature fine tailings.

2.5.3 Desiccation

With proper management, optimal sub-aerial deposition or aggressive drainage techniques within existing deposits have been shown to over-consolidate various fine tailings leading to a potentially reclaimable deposit. Since these disposal options rely on evaporation/desiccation it is important to understand the factors that influence these processes. Desiccation of tailings depends upon material properties, as well as the top and bottom boundary conditions (Newson and Fahey 2003; Simms and Grabinsky 2004). These various factors are illustrated in Figure 2.21.

As evident in Figure 2.21, at the upper boundary condition, evaporation controls the rate of water transfer from the tailings surface to the atmosphere. The evaporation rate at the surface depends on the available energy (radiation), the distribution of energy within the system (albedo effects), and the local meteorological conditions which impact the ability of the air to transfer water vapour away from the surface (Newson and Fahey 2003; Qiu and Sego 2006). Material properties such as particle size distribution, saturated/unsaturated hydraulic conductivity and soil-water characteristic curve (SWCC) influence the availability and flow of water to the evaporation surface. In fine tailings where bitumen content is relatively high, the bitumen may form a surface coating that thickens as the tailings dewater. This layer may impede moisture transfer from the tailings to the atmosphere essentially reducing or halting evaporation. The presence of salts within the fine tailings leading to salt crusting may also lead to a reduction in the evaporation rate (Newson and Fahey 2003; Simms et al 2007). This may be due to an increase in the tailings surface albedo, resistance to moisture transfer at the surface and decrease in the saturation vapour pressure. A detailed description on the impacts of salts on the evaporation rate can be found in Newson and Fahey (2003).

Conditions below the evaporating layer may also impact the amount of desiccation occurring. Suction from a dry underlying layer may contribute to desiccation of a fresh layer of fine tailings. In deep deposits of fine tailings, underlying saturated tailings may provide moisture to recharge the desiccating surface. These interlayer interactions will depend upon the moisture conditions and hydraulic conductivity of the underlying layer (Qiu 2000).

Evaporation from saturated tailings occurs in three stages (Figure 2.22) (Newson and Fahey 2003). In stage 1, evaporation from the saturated tailings is governed by the climatic and environmental conditions (Wilson et al 1994). As the tailings layer dries, its moisture content will reduce from well above the liquid limit, w_L, at point A down to the plastic limit, w_P. The rate of evaporation in stage 1 is near the potential evaporation rate, Ep.

Common methods for estimating Ep include using the pan evaporation rate or calculating Ep from detailed weather data using one of several available models (Blight 2010; Qiu 2000; Simms and Grabinsky 2004; Wilson et al 1994). In Fort McMurray, Alberta, the average annual precipitation (rain and snow) from 1971-2000 was 455 mm (EC 2009). Of which 334 mm were measured between April 1 and September 30 (days with average daily temperature above 0^{o}C). Yearly evaporative conditions from 1972- 1994 were 572 mm (Abraham 1999) leading to a rainfall deficit of 238 mm. Assuming all precipitation was drained away from the surface of a tailings deposit, the potential evaporation may be 3.13 mm/day. Qiu (2000) calcuated Ep for Fort McMurray, Alberta to be 5-8 mm/day.

Stage 2 begins when tailings can no longer satisfy the evaporation demand, and they begin to desaturate. Moisture content continues to drop below the w_P approaching the shrinkage limit, w_S. In stage 2, a rapid decrease in the evaporation rate occurs and is dependent on both the climatic conditions and soil properties (permeability). Once the moisture content drops to residual (below the w_S), evaporation and further volume change has reached stage 3 and is effectively zero (Wilson et al 1994).

Models are available for estimating the degree of drying that may occur in fine tailings. However, they require material properties such as SWCC, saturated and unsaturated conductivity and the maximum or potential evaporation rate (Qiu 2000; Simms and Grabinsky 2004). Given the particle size, it is very difficult to obtain a SWCC for MFT.

To achieve maximum evaporation, excess surface water from precipitation or consolidation must be adequately drained from surface of drying tailings. If adequate drainage is not provided, desiccation of the tailings will not occur until the evaporation rate exceeds the rate of water release from consolidation and precipitation.

Other factors such as climatic conditions may also impact the effectiveness of the desiccation process. In field trials of atmospheric drying of oil sands fine tails, Burns et al. (1993), found wind and rain lead to shifting islands of dry, crusted tailings surrounded by ponded water. Fresh fine tailings would then well up through the surface crustal cracks preventing the formation of a uniform stable crust.

2.5.4 Freeze –Thaw

Freeze-thaw is an effective natural way for dewatering fine tailings such as MFT. Freeze-thaw alters the macro and micro-fabric which enhances water release. This in turn allows the tailings to consolidate rapidly under self-weight conditions and then under the effective stress induced during freezing. Dewatering and structural enhancement are attributed to moisture migration from both "open" and "closed" system freezing that occurs within the fine tailings deposit during winter months (Stahl and Sego 1995). Open system freezing is characterized by freezing of in situ moisture and the migration of moisture to the freezing front within the frozen fine tailings (Konrad and Morgenstern 1980; Stahl and Sego 1995). As the frost front advances through the tailings, the in situ pore fluid freezes. Additionally, under the action of temperature induced suction gradients, water migrates to the freezing front, from the underlying unfrozen tailings. The migrating water leads to the formation of an ice lens near the freezing front while dewatering (consolidating) the unfrozen tailings. Further advancement of the front is stalled due to the release of latent heat from water moving to the freezing front that freezes. Eventually, the migration of water is retarded and the freezing front advances until thermal equilibrium conditions are again reestablished and new ice lenses begin forming.

In closed system freezing, moisture migration occurs only at the micro-scale due to redistribution of in situ moisture within the already frozen zone. A matrix of reticulate pore ice develops within the frozen tailings, surrounding the mineral particles and their unfrozen adsorbed water films. The adsorbed water from the partially frozen structure is drawn to the ice lens matrix by temperature induced, internal suction gradients. This localized moisture migration results in the reduction of absorbed water and consolidation of the mineral peds. The fine tailings are transformed into a three-dimensional system of ice lenses and over consolidated soil "ped" structures. Upon thaw, a segregated profile develops consisting of melt water from the ice matrix and over consolidated soil peds which settle as water is released to the surface (Nixon and Morgenstern 1973; Stahl and Sego, 1995). The resulting volume change is referred to as thaw strain and represents the change in volume due to freeze-thaw under negligible change in effective stress (Dawson et al. 1999).

Volume change is most pronounced when tailings are frozen in thin layers under a closed freezing system. Sego (1992), Johnson et al (1993), Proskin (1998), and Dawson et al (1999) investigated the dewatering potential of closed system freezing of on MFT. Various samples of oil sand fine tailings and MFT were subjected to 1 to 3 cycles of freeze thaw. The resulting thaw strain increased the solids content of the fine tailings from ~30% up to 54% after several cycles of freeze thaw. The studies demonstrated that freeze-thaw becomes less effective in dewatering as the initial frozen solids content increases. Dawson et al. (1999) also found the lower the freezing rate, the higher the thawed settled solids contents. Therefore, layer thickness, temperature gradient and boundary temperature also affect freeze-thaw dewatering.

The freeze-thaw process also enhances the saturated hydraulic conductivity and compressibility of fine tailings. Upon thaw, remnant ice fissures provide channels for fluid flow and account for an increased hydraulic conductivity at low stresses (less than 100 kPa) and void ratios above 1 (Proskin 1998; Dawson et al. 1999). In MFT, the micro-fabric also changes after freeze thaw from an edge to face flocculated, disaggregated card house fabric to a compact, aggregated fabric. The latter micro-fabric retains less water which accounts for an increase in solids content (Proskin 1998). These changes allow for dewatering rates that are several orders of magnitude greater compared to unfrozen fine tailings at the same initial void ratio. Dawson et al. (1999) explain the difference by comparing the coefficient of consolidation (C_v) of unfrozen and frozen-thawed fine tailings undergoing volume change (Figure 2.23). The C_v can be calculated from the compressibility (A and B) and hydraulic conductivity (C and D) power law constants in the following equation (Dawson et al. 1999):

[2.14] $$C_v = \frac{Ce^D(1+e_0)}{-\gamma_w AB\left({}^{e}/_{A}\right)^{\frac{B-1}{A}}}$$

MFT at 30 % solids content has an in situ void ratio of 5.8 (point A on Figure 2.23). During freezing, the C_v increases significantly to point B. Upon thaw, the void ratio decreases from 5.8 to 2.3 (30 to 52% solids content) rapidly as the dense soil peds settle and released water is drained from the surface. When sufficient downward drainage exists, significant self-weight consolidation occurs, further decreasing the void ratio to 1.7 (point B to C). Surcharge loading the following winter (overlying frozen tailings layer) will further dewater the tailings to void ratios approaching the never frozen line, assuming pore pressure can be dissipated. At void ratios of approximately 1 and effective stress of 100 kPa, the frozen-thawed fine tailings behave like never frozen tailings.

2.5.5 Plant Dewatering

Plants have been utilized by civil engineers to enhance slopes, prevent erosion and landscape reclamation for many years (Wu et al. 2010). Plants can also enhance the desiccation of fine tailings deposits through a process known as evapotranspiration. When rooted in tailings, plants can transfer wate through their roots from the deposit and transpire it through their leaves, thus dewatering the tailings in addition to evaporation. The root systems that develop in the surface (up to 2 m depth) can also add fiber reinforcement which may contribute to increasing bearing capacity at the surface (Silva 1999; Wu et al. 2010). A schematic of plant dewatering and structural enhancement is presented as Figure 2.24 (Silva 1999). The rate at which plants can uptake and transpire water may be limited by the soil properties (permeability and SWCC), the plant properties (i.e. leaf area index) and the atmospheric conditions. During the early growth stage, evapotranspiration is dominated by the actual soil evaporation rate. With little vegetative cover, evaporation from a wet soil surface is dependent on the energy available as described previously (Stage 1 drying). As the surface soil dries, the evapotranspiration rate becomes a function of the hydraulic properties of the surface soil (Figure 2.25: drainage to the surface and interlay interaction). With time, as the plant canopy increases, evapotranspiration is dominated by transpiration from the plants. The transpiration rate depends on leaf area and the ability of the roots to uptake water from the soil deposit (Wu et al. 2010).

Early attempts to utilize plants for dewatering were carried out in the Netherlands where high water content oceanic sediments were deposited into polders and seeded with plants (Volker 1982). The resulting evapotranspiration accelerated the drying and reclamation process. Evapotranspiration has also been used to dewater wastewater sludge in Austria and southern Germany (Neurohr 1983) and reclamation of tailings disposal areas (Leroy 1972). A fully vegetated tailings deposit in northeastern Canada had the potential to transpire 4.5 to 9.0 mm/day/acre with proper fertilizer application and management of the deposit (Leroy 1972). Dewatering of fine grained tailings and CT tailings have also been investigated by Johnson et al. (1993), Stahl (1996), Silva (1999), and Wu et al. (2010). In one growing season, Johnson et al. (1993) dewatered oil sand tailings from 50% solids to 80 % and increased the undrained shear strength to 120 kPa. Stahl (1996) studied changes in the surficial stability of fine coal tailings deposit when the surface was subjected to evaporation, evapotranspiration and root fibre reinforcement. In some cases, the bearing capacity of rooted tailings was up to 60% greater than un-rooted tailings. In green house experiments, Silva (1999) evaluated the response and dewatering potential of several plant species in oil sands CT deposits. In one growing season, the solids content in the surface (upper 50 cm) of the CT deposit increased from 65 % up to 95 %. Wu et al. (2010) utilized similar experiments to investigate the impact of native plants species and seeding techniques on dewatering of CT deposits. Again, the solids content increased from 65 % up to an average of 90.5 %.

2.6 TAILINGS CONTAINMENT

Mining and extraction processes that produce a slurry tailings stream typically utilize a constructed storage impoundment/facility. The purpose of the impoundment is to contain the fine grained tailings slurry and to clarify and store water for extraction process (Vick 1990). Tailings impoundment configurations share many common design features, however, individually they must balance cost, stability (short and long term) and environmental performance realizing actual designs are highly site-specific. US EPA (1994) states that the "Design depends on the quantity and the individual characteristics of the tailings produced by the mining and milling operation, as well as the climatic, topographic, geologic, hydrogeologic and geotechnical characteristics of the disposal site, and on regulatory requirements related to dam safety and to environmental performance." Costs are often related to amount of fill material used to construct the impoundment. Major cost saving can be realized by maximizing the use of the topography and local materials including utilizing the tailings as a construction material.

There are four main configurations of tailings impoundments: valley fill, ring-dyke, in-pit, and specially dug pit (Vick 1990). Valley fill impoundments make use of natural depressions to provide containment for the tailings and minimize the amount of construction material required. Often, a single embankment is constructed connecting two valley walls to provide containment (Figure 2.25a). Where there is a lack of relief, (slopes less than 10% grade) side hill impoundments may be used utilizing a three-sided embankment constructed against a hillside (Figure 2.25b). Designs for valley fill impoundments must include sufficient storage for run-off and drainage to the valley catchment area. In regions where there is a lack of topographic depressions, a ring-dyke configuration may be utilized (Figure 2.25c). Embankments are required on all sides of the impoundment to provide the necessary containment for the tailings and process water. Due to the increased length of the embankments, more materials are needed for ring-dykes than the valley fill impoundments. Ring-dyke impoundments are often lower in height and are not located in a natural catchment area, thus they may be simpler to design (US EPA 1994). Another common storage configuration exploits available space in the mine pit for storage. For in-pit impoundments, tailings are deposited into previously mined out pits such that containment is provided by the pit walls. If mining progresses laterally, embankments may be constructed to separate deposition and storage of tailings within the pit from active mining operations.

Embankments are normally constructed with waste rock, overburden, natural material from borrow pits and/or coarse fraction of the tailings placed mechanically or hydraulically. An appropriate level of compaction of construction material is required as well as sufficient drainage measures to ensure stability and prevent uncontrolled leakage from the impoundment. Starter dykes are normally constructed with waste or borrow material. Construction may then shift to hydraulically placed tailings to raise the embankment in a timely manner to meet storage requirements (Vick 1990; Blight 2010).

There are three construction methods to raise an embankment: downstream, upstream, and centerline (Vick 1990; Blight 2010). A comparison of the benefits of each method is included in Table 2.1. In the downstream method, the centerline of the embankment moves progressively downstream as construction advances (Figure 2.26a). A downstream embankment offers the greatest stability, however requires significant construction material and a large footprint. The embankment in the upstream method steps over previously beached tailings in the upstream direction (Figure 2.26b). It is crucial that the tailings beach forms a competent foundation to ensure stability of the structure. This method offers the least amount of construction material, however is susceptible to liquefaction. The dyke crest in a centre-line embankment does not move laterally as construction progresses (Figure 2.26c). Tailings are beached off the crest of the embankment in the upstream direction, while placement and compaction of fill continues in the downstream direction. This method of construction capitalizes on the advantages of the other methods while mitigating their drawbacks.

At mining operations where centrifuges, ultra-high density thickeners, or filters (Figure 2.3) are utilized, the resulting high density tailings paste or cake may be self-supporting or stackable thus may not require full containment. The tailings may be deposited from one or more discharge locations within or along a perimeter embankment forming a sloped pile. These tailings deposits naturally shed precipitation due to the inherent slope formed during deposition. A perimeter embankment will likely be required to confine the tailings stack and manage runoff. Embankments for paste or cake deposits can be significantly smaller than conventional slurry embankments (Figure 2.27). However, the resulting tailings deposit must be geotechnically stable and not susceptible to liquefaction. A challenge as discussed previously with these types of containment facilities is predicting the design slope of the tailings deposit. Small changes in the final profile of the tailings deposit can have significant implications on the foot print of the facility or the discharge structure. For example, at a given discharge height, flatter slopes can result in larger footprints whereas steeper slopes will result in the tailings deposit overwhelming the discharge structure (Jewell and Fourie 2006).

2.7 CONCLUSIONS

Management of tailings and waste rock are an integral part of all mining operations. Several aspects of waste management for tailings were discussed including mechanical/chemical dewatering, transport to and construction of geotechnically sound impoundments, depositional behavior, and post depositional dewatering. The physical and chemical material properties of the tailings particles and slurry such as particle size, mineralogy, specific gravity, slurry density, pore fluid chemistry, and chemical additives, were shown to have an impact on each of the above processes.

The selection of a particular tailings treatment and depositional method (Figure 2.1 and Figure 2.3) will depend upon the objectives of the operator/regulator or applicable regulations for the mine site. Typically, operators look for efficient and cost effective systems that provide sufficient protection of the environment and satisfy regulations. The TMS must also be dynamic to cope with a tailings facility whose geometry and operational considerations change over the life of the mine (decades). Facilities should be constructed and operated in an orderly fashion to ensure the main objectives of the TMS are achieved.

2.8 TABLES

Table 2.1 Comparison of tailings embankment construction methods (modified from US EPA 1994).

	Embankment Type		
Embankment Attributes	Downstream	Upstream	Centre-line
Mill tailings requirements	Suitable for any type of tailings	At least 60% sand in whole tailings. Low pulp density for grain size segregation	Sands or low-plasticity slimes
Discharge requirements	Varies according to design detail	Peripheral discharge, well controlled beach	Peripheral discharge of at least nominal beach necessary
Water-storage suitability	Good	Not suitable for significant water storage	Not recommended for permanent storage. Temporary flood storage can be designed.
Seismic resistance	Good	Poor in high seismic areas	Acceptable
Raising rate restrictions	None	Less than 4.5 to 9 m/yr most desirable. Over 15 m can be hazardous.	Height restrictions for individual raises may apply
Fill requirements	Sand tailings, waste rock, native soils	Native soil, sand tailings, waste rock	Sand tailings, waste rock, native soil
Relative cost	High	Low	Moderate
Use of low permeability cores	Possible (inclined cone)	Not possible	Possible (Central cone)

2.9 FIGURES

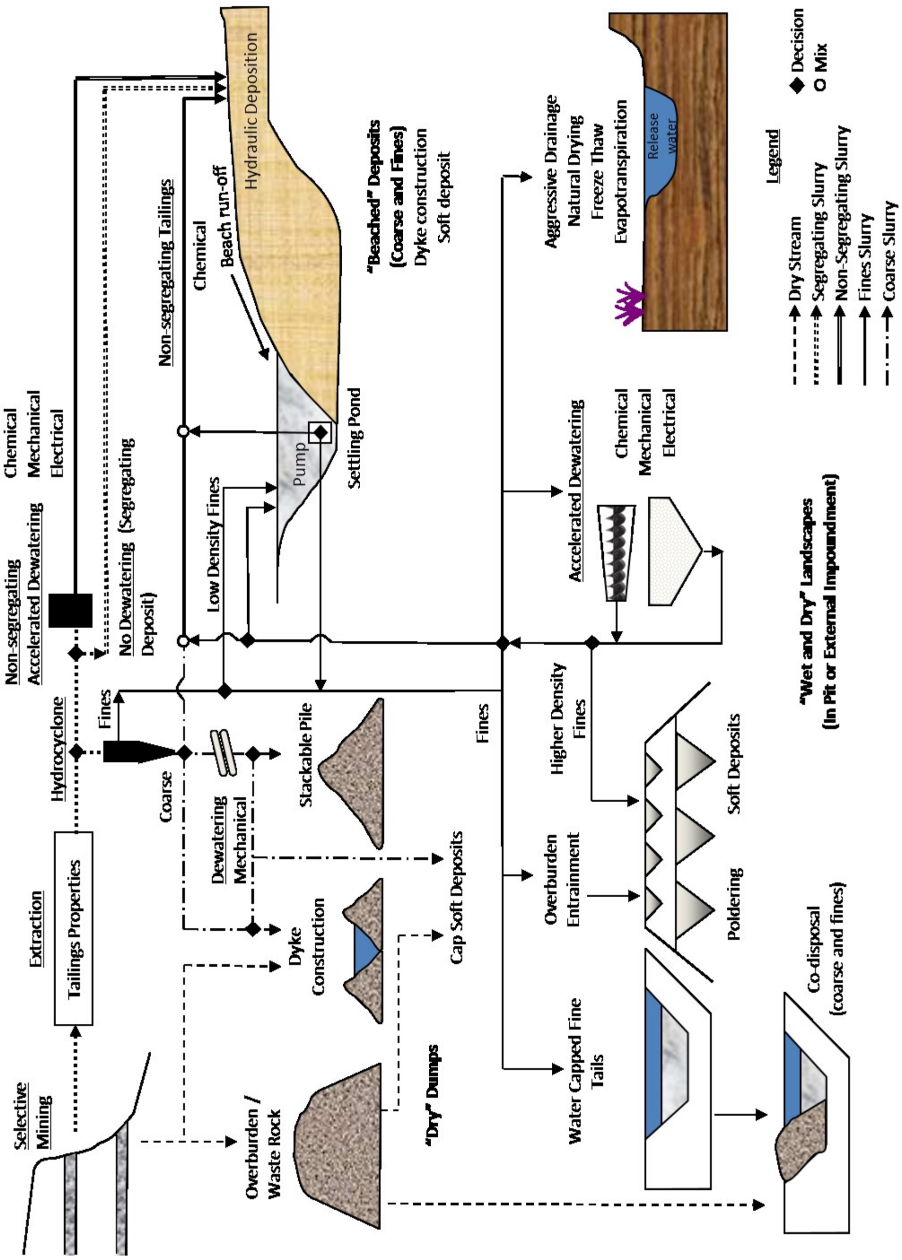


Figure 2.1. Mine Waste Management Strategies (Modified from Sheeran 1993).

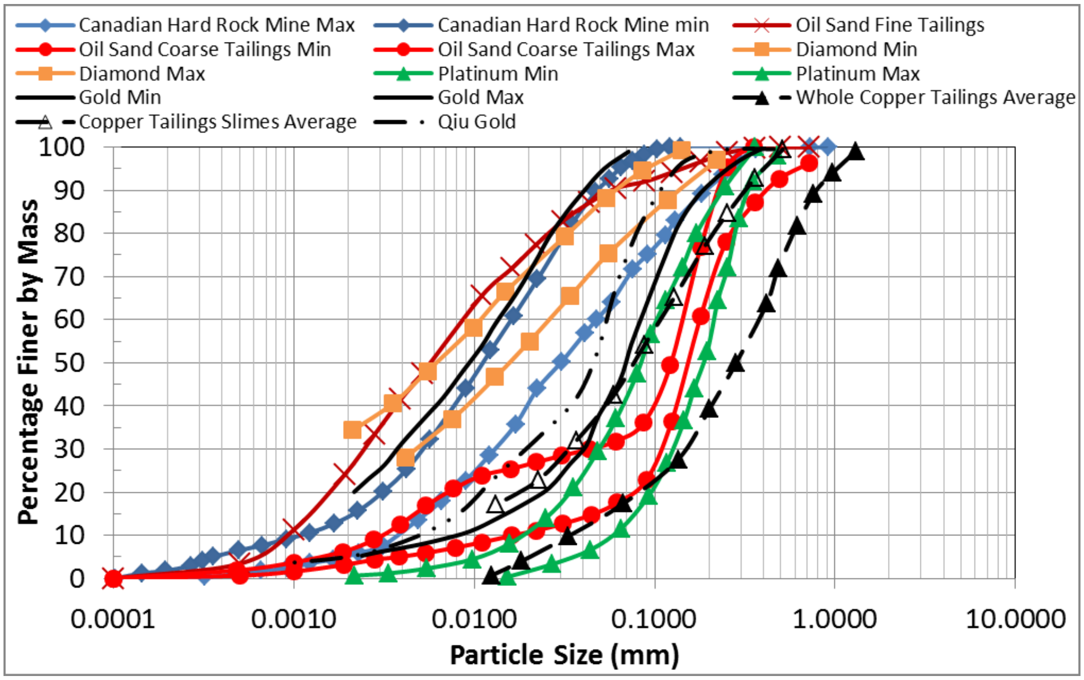


Figure 2.2 Typical grain size distributions of various mine tailings (Bussiere 2007: Canadian Hard Rock Mines; Fair 2008: Oil Sands Fine and Coarse Tailings; Blight 2010: Diamond, Gold and Platinum; Vick 1990: Copper Tailings; and Qiu 2000: Gold Tailings).

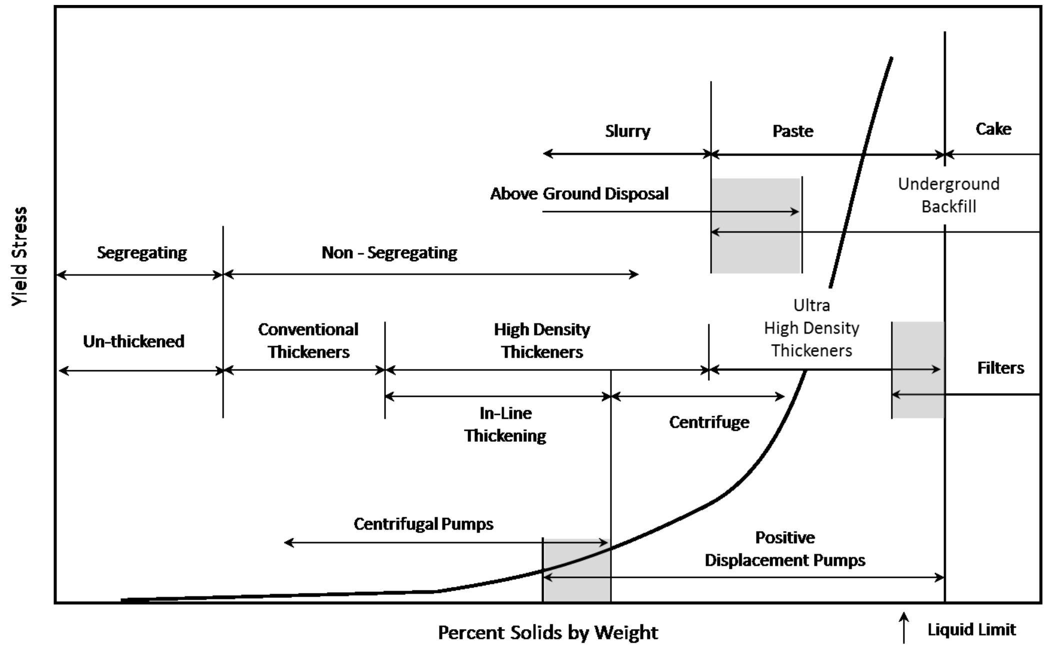


Figure 2.3 Continuum of dewatering methods and tailing behaviour. (modified from Jewell and Fourie 2006)

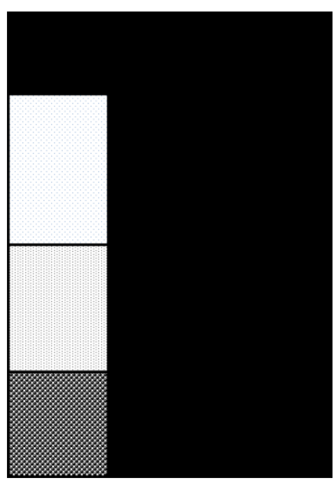


Figure 2.4 Schematic of the sedimentation process in thickeners.

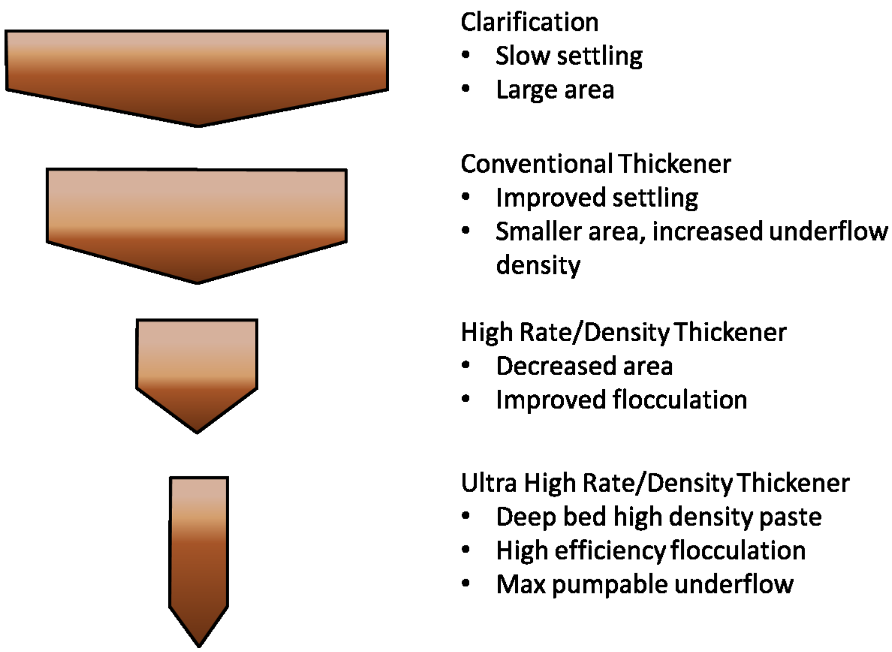


Figure 2.5 Continuum of thickener designs (modified from Jewell and Fourie 2006).

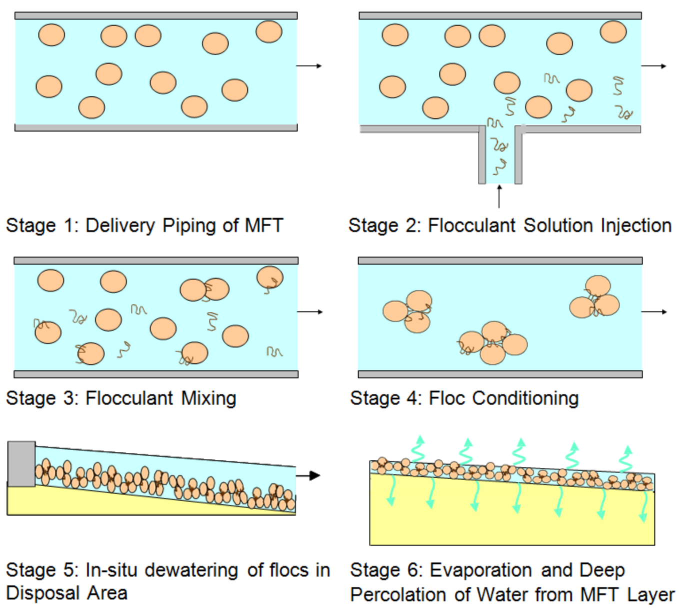


Figure 2.6 In-line thickened tailings process illustration (modified from Kolstad et al. 2012).

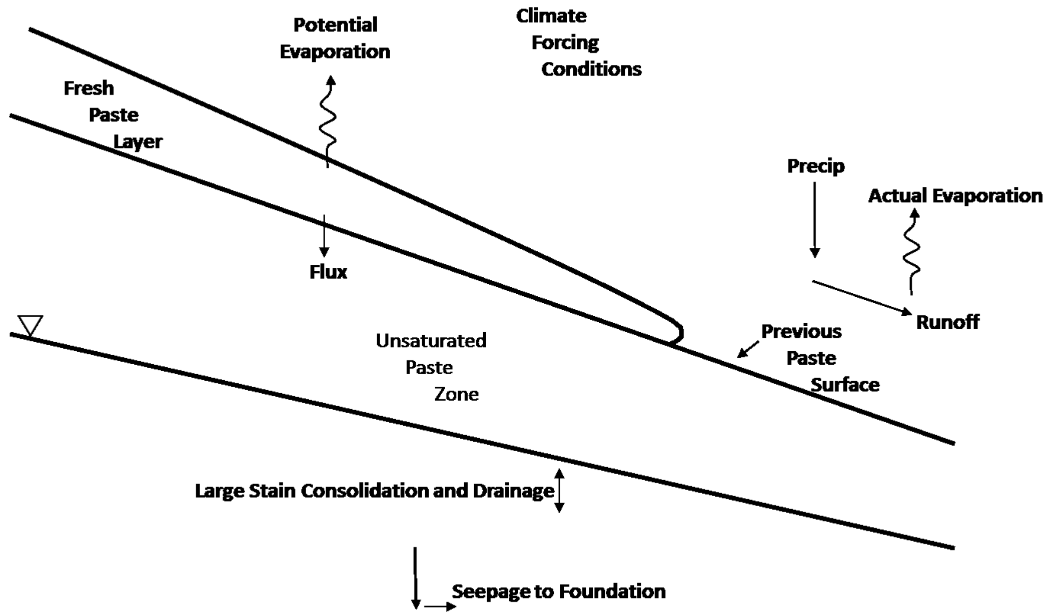


Figure 2.7 Schematic of the ILTT process (modified from Wilson 2010).

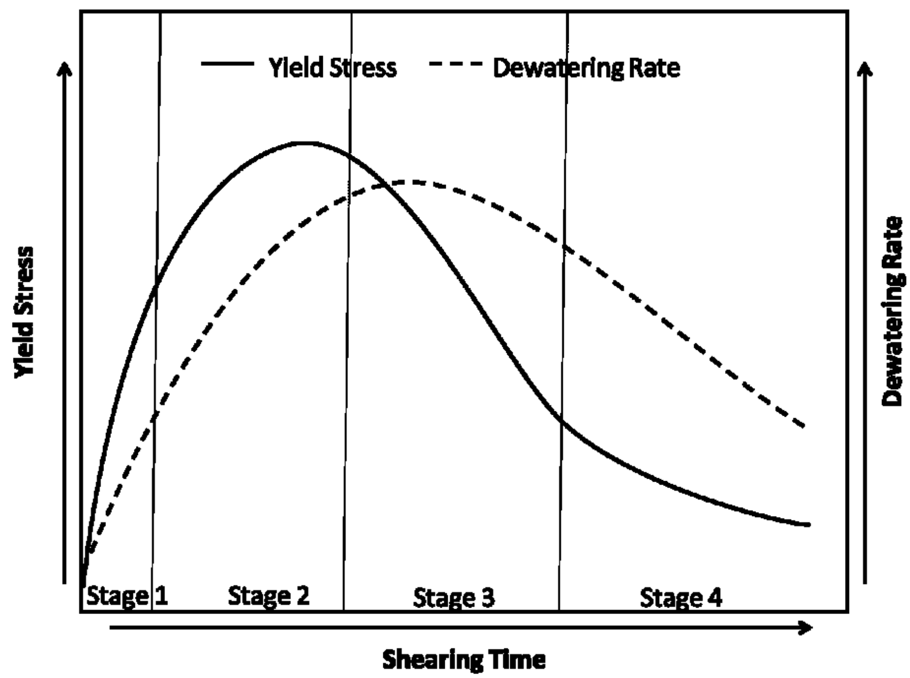


Figure 2.8 Stages of dewatering and strength gain for the ILTT process. (modified after Wells et al. 2011). Stage 1 - Floc assemblage: the polymer is dispersing within the slurry; Stage 2 - Floc re-arrangement: floc formation is at a maximum and is balanced by floc breakdown; Stage 3 - Floc breakdown and dewatering; : continued shear breaks down flocs and network; Stage 4 - Over shear zone: Further shear leads to complete breakdown of flocs.

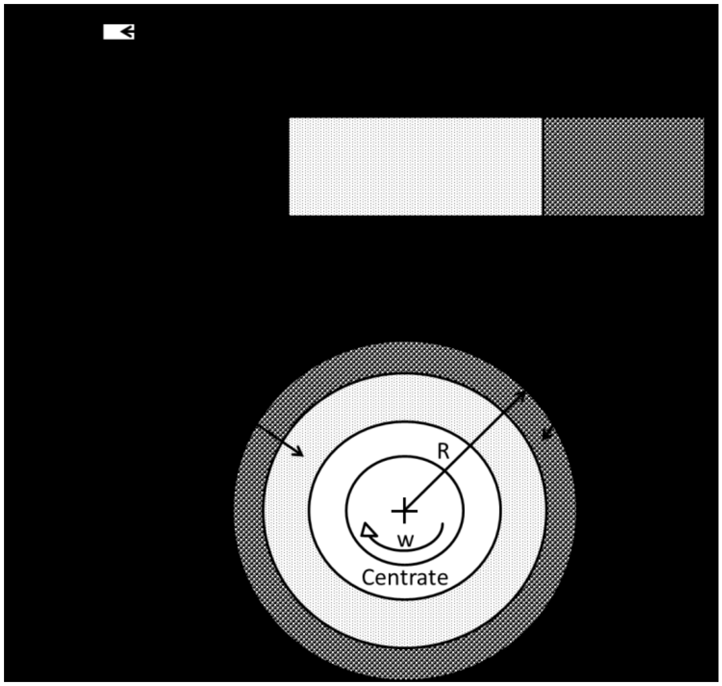


Figure 2.9 Solids concentration zones within a centrifuge (Modified after Burger and Concha 2001). a) Rotating tube of slurry with constant cross section; b) rotating cylinder of slurry.

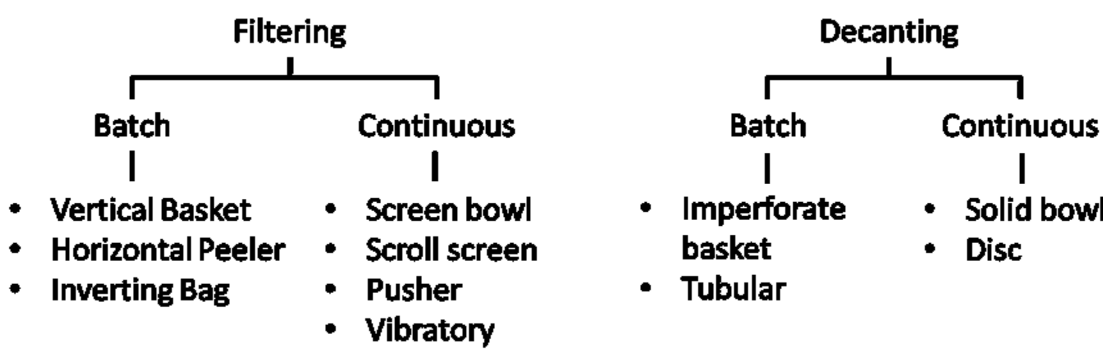


Figure 2.10 Types of clarifying centrifuges (modified after Norton and Wilkie 2004).

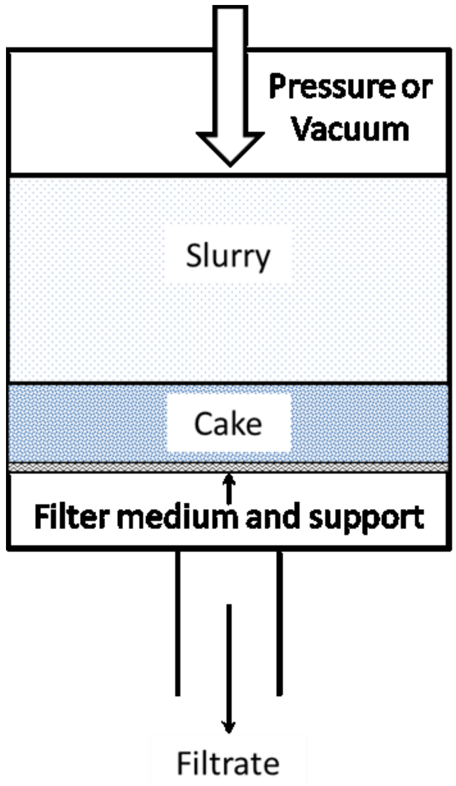


Figure 2.11 Schematic of a dead end filter (modified after Richardson et al. 2002).

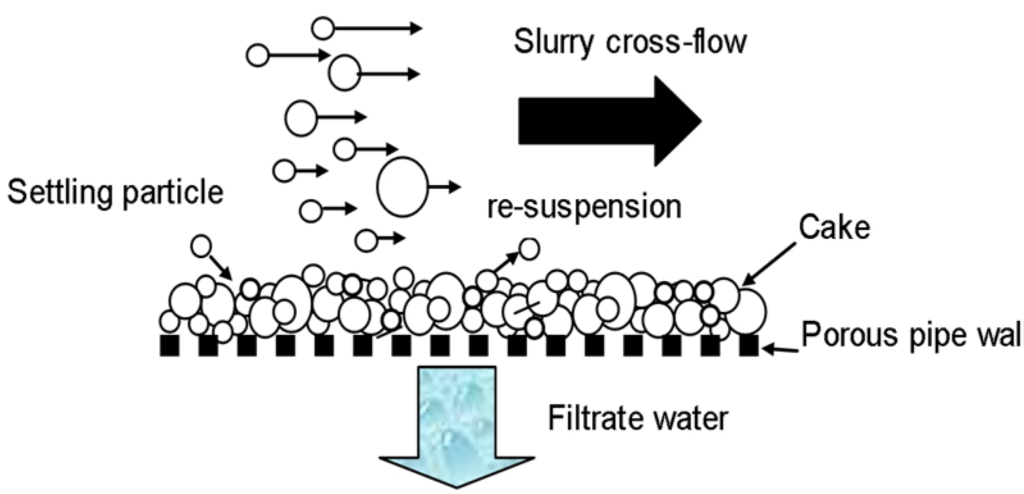


Figure 2.12 Schematic of the cross flow filtration process (modified after Beier and Sego 2008).

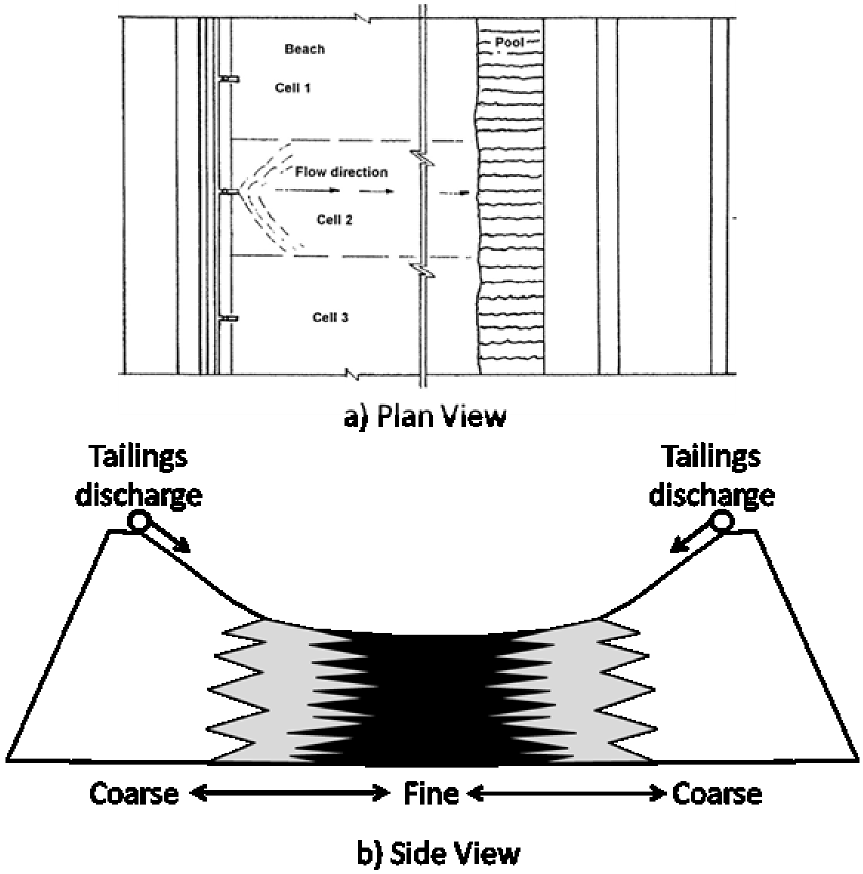


Figure 2.13 Sub aerial deposition a) plan view of multiple spigots and deposition cells (modified from Qiu and Sego 1998), b) Cross section of an impoundment depicting the potential segregation from sub-aerial deposition (modified from Al and Blowes 1999).

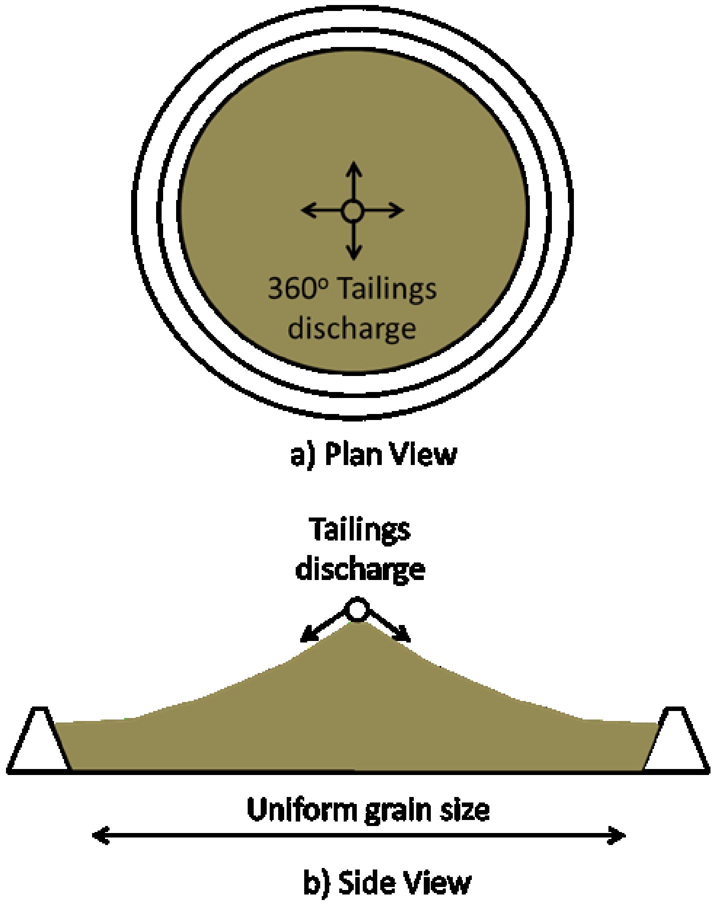


Figure 2.14 Thickened or Paste tailings deposition a) plan view of a central discharge, b) Cross section of a central discharge thickened tailings stack (modified from Al and Blowes 1999).

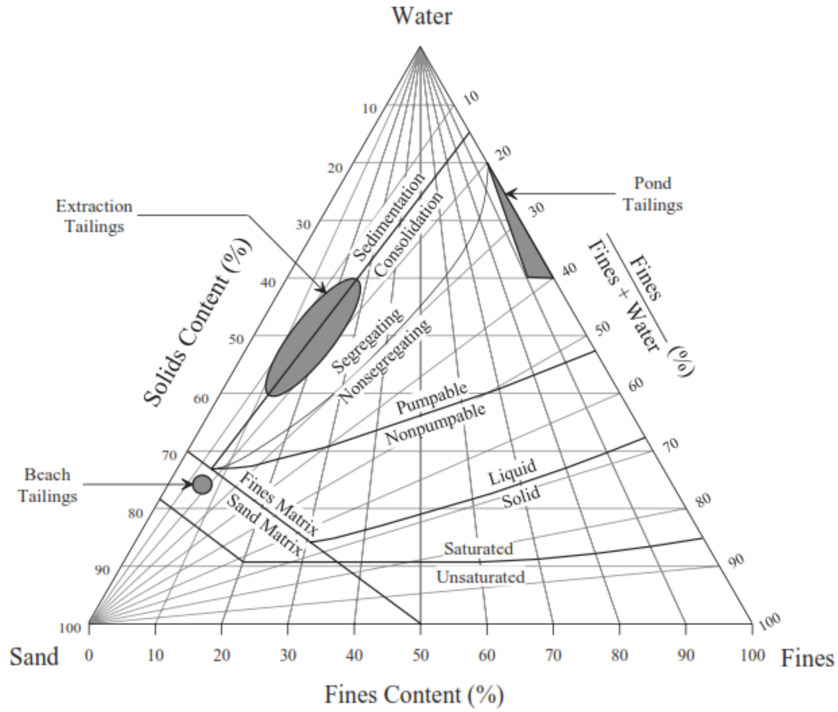


Figure 2.15 Ternary diagram for oil sand tailings (modified from Azam and Scott 2005). Fines refers to material passing 45 µm. Sand refers to material greater than 45 µm.

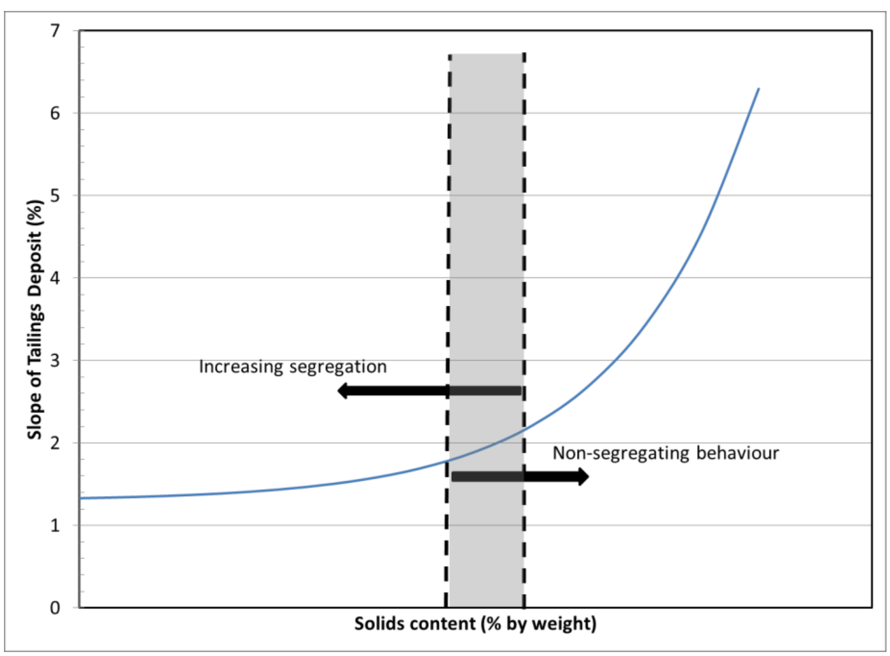


Figure 2.16. Robinsky's (1978) plot of solids content versus deposit slope (modified from Fitton 2007).

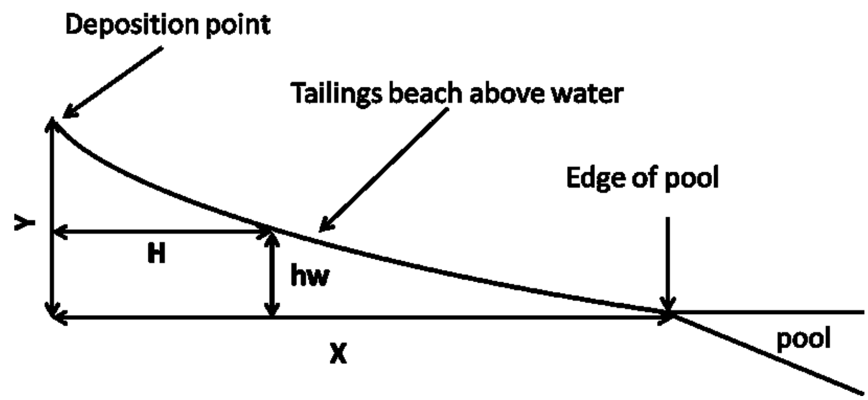


Figure 2.17 Master profile of tailings beaches (modified from Blight et al. 1985).

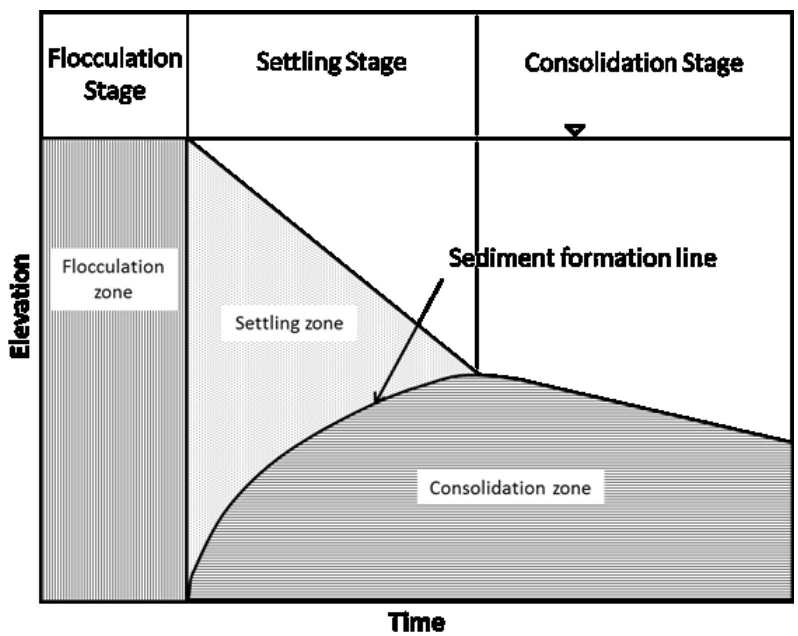


Figure 2.18 Sedimentation and Consolidation Schematic (modified from Imai 1981).

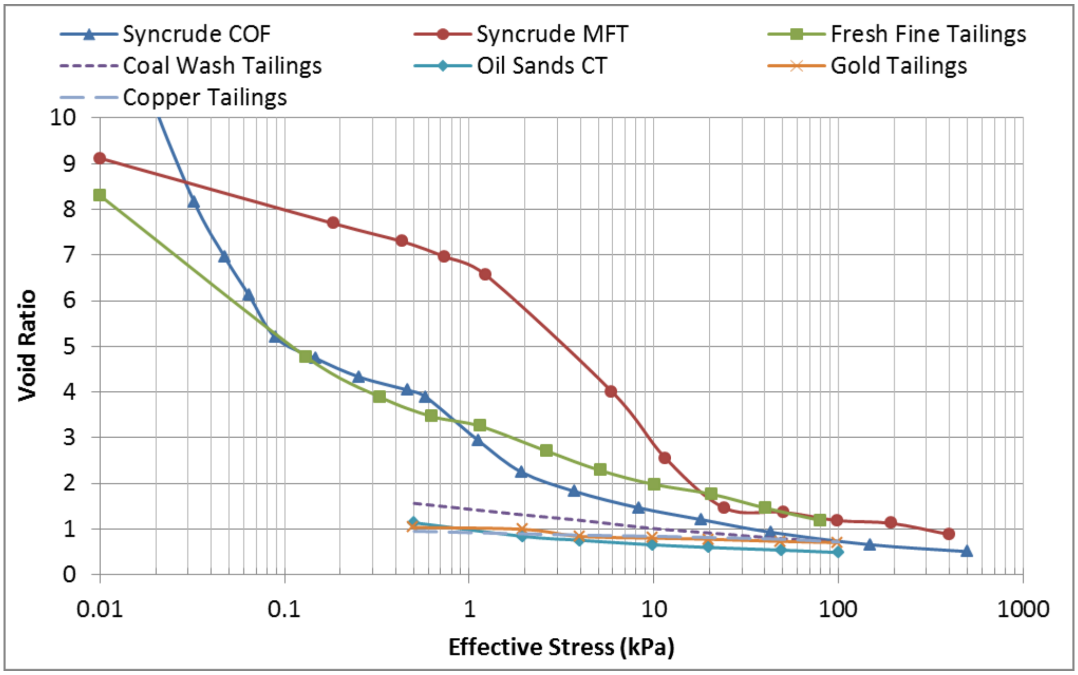


Figure 2.19 Compressibility of various mine tailings slurries (Jeeravipoolvarn 2010: Syncrude COF; Miller et al. 2011: Syncrude MFT and Fresh Fine Tailings; Qiu 2000: Coal wash, Oil Sands CT, Gold and Copper tailings).

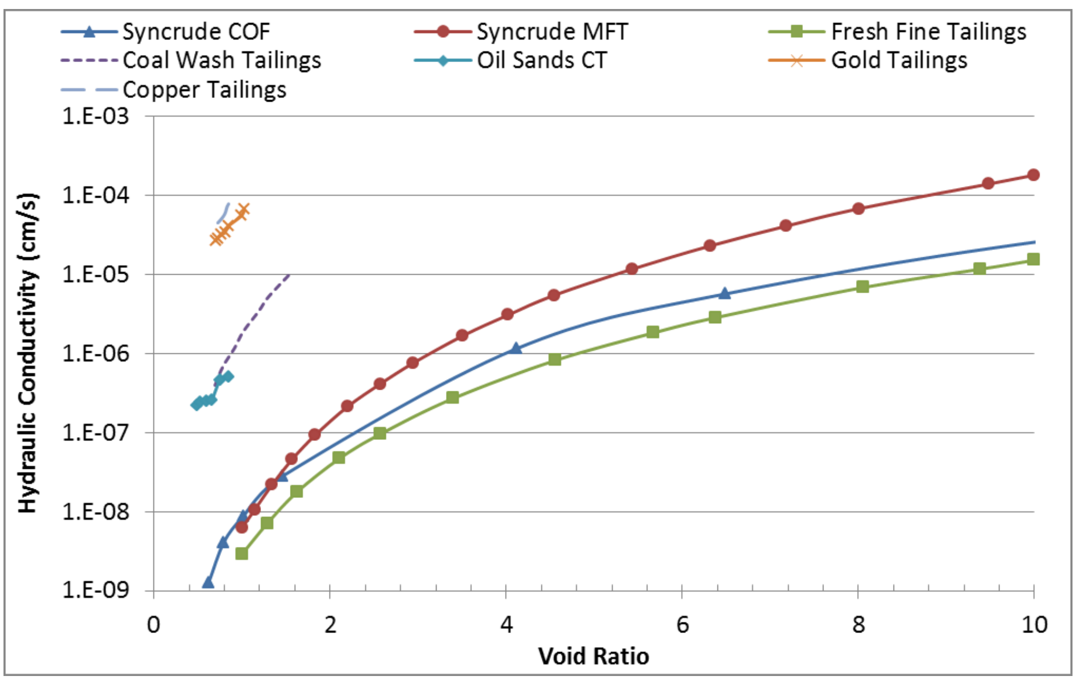


Figure 2.20 Saturatede hydraulic conductivity of various mine tailings slurries (Jeeravipoolvarn 2010: Syncrude COF; Miller et al. 2011: Syncrude MFT and

Fresh Fine Tailings; Qiu 2000: Coal wash, Oil Sands CT, Gold and Copper tailings).

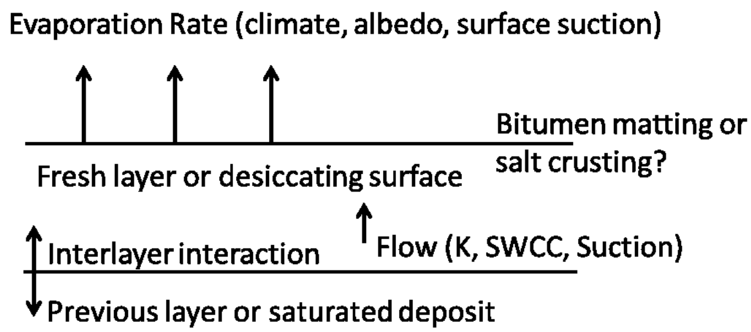


Figure 2.21 Factors influencing desiccation of tailings (modified from Simms and Grabinsky 2004).

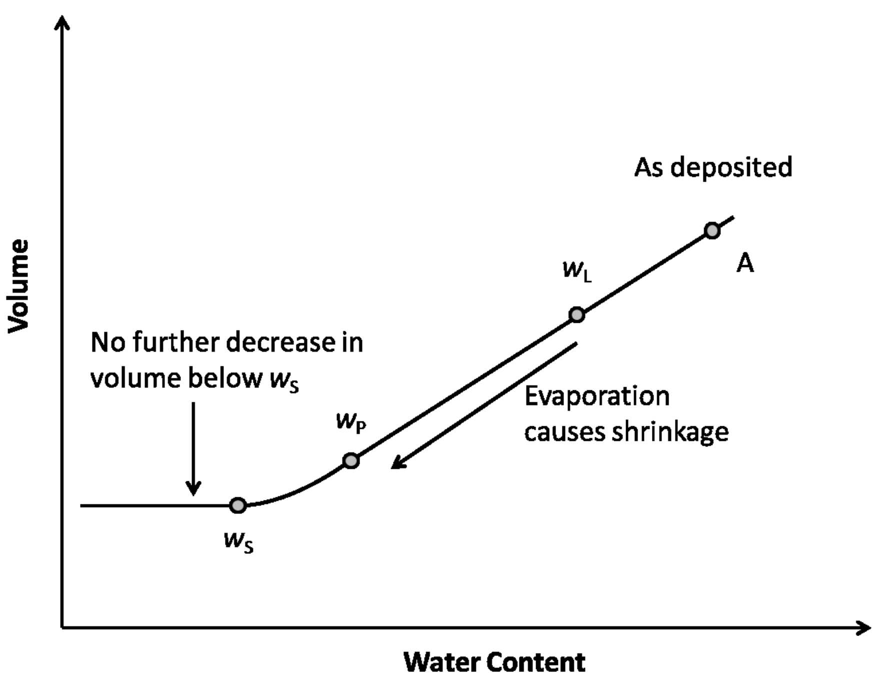


Figure 2.22 Volume change during desiccation of tailings (modified from Newson and Fahey 2003).

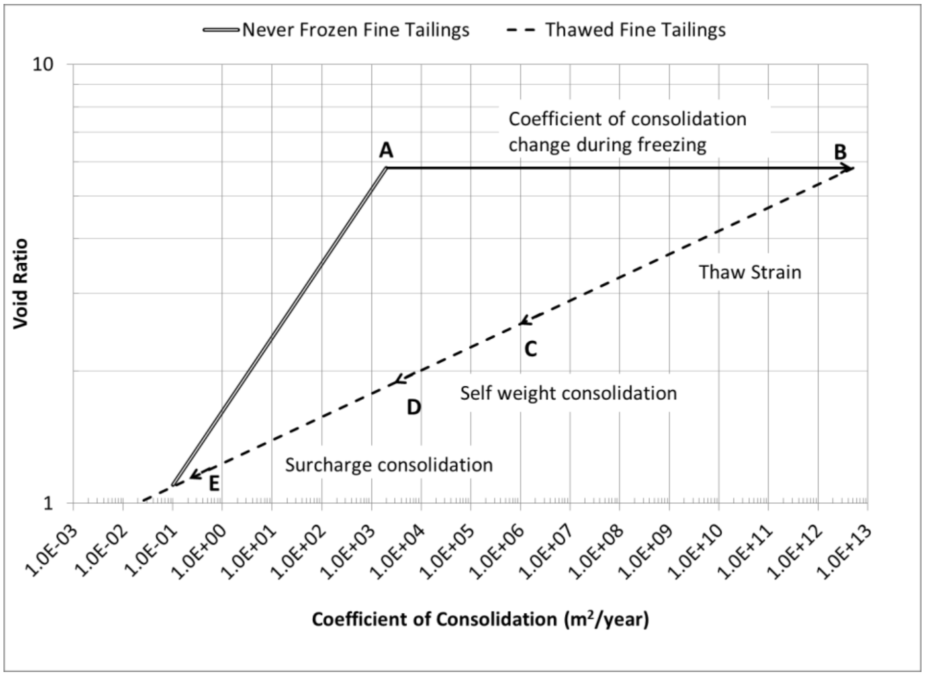


Figure 2.23 Freeze thaw dewatering consolidation behavior of fine tailings (modified from Dawson et al. 1999).

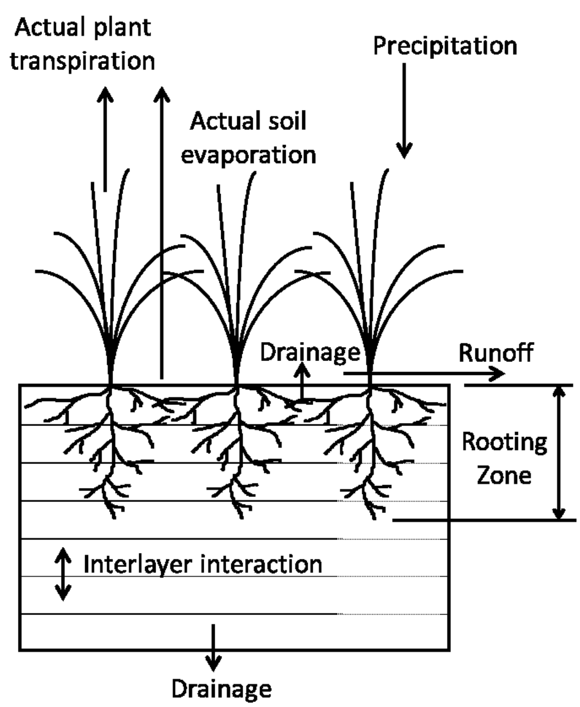


Figure 2.24 Schematic of plant dewatering (modified from Silva 1999).

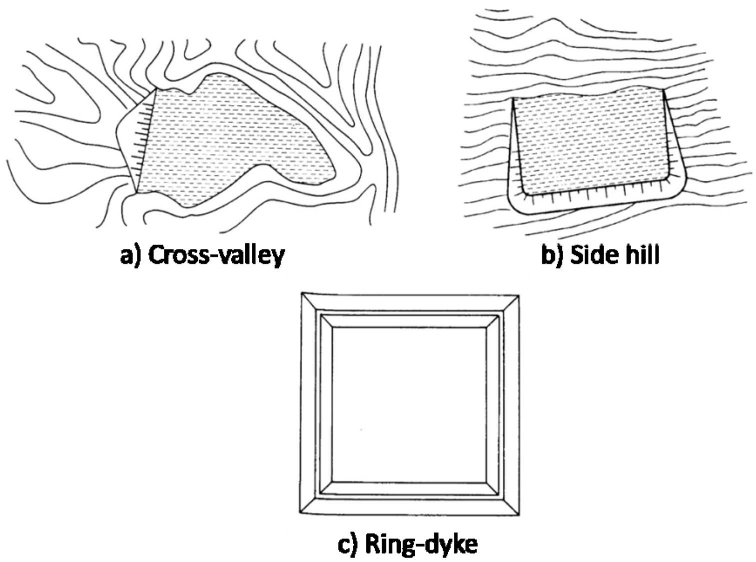


Figure 2.25 Common tailings impoundment configurations (modified from Vick 1990).

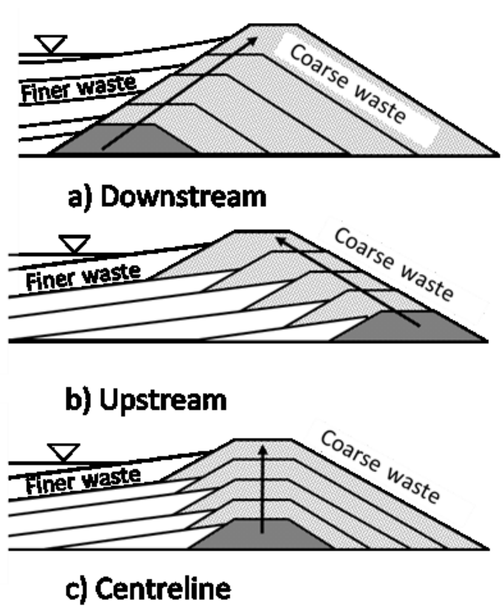


Figure 2.26 Tailings embankment types (modified from Blight 2010).

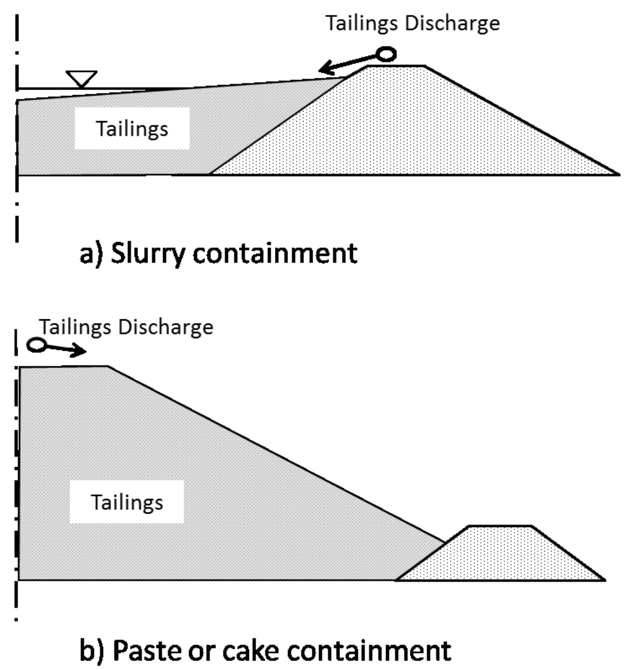


Figure 2.27 Slurry tailings versus paste or cake containment.

2.10 REFERENCES

Abraham, C. 1999. Evaporation and Evapotranspiration in Alberta. Report 1912-1985, Data 1912-1996. Water Sciences Branch, Water Management Division, Alberta Environmental Protection. January 1999.

Ahmed, I., Labelle, M., Brown, R. and Lahie, R. 2009. Paste pumping and deposition field trials and concepts on Syncrude's dewatered mft (centrifuge cake). Tailings and Mine Waste 2009, Banff, Alberta, 1-4 November 2009. pp. 417-427.

Al, T.A. and Blowes, D.W. 1999. The hydrogeology of a tailings impoundment formed by central discharge of thickened tailings: implications for tailings management. Journal of Contaminant Hydrology, 38(1999): 489-505.

Allen, E. 2008. Process water treatment in Canada's oil sands industry: I. Target pollutants and treatment objectives. Journal of Environmental Science, 7: 123-138.

Australian Government. 2007. Tailings Management. Leading practice sustainable development program for the mining industry. Commonwealth of Australia. www.ret.gov.au/sdmining , accessed Aug 8, 2007.

Azam, S. and Scott, J.D. 2005. Revisiting the ternary diagram for tailings characterization and management. Geotechnical News, December 2005, 43-46.

Barbour, S.L., Wilson, G.W., and Arnaud, L.C. 1993. Evaluation of the saturated-unsaturated groundwater conditions of a thickened tailings deposit. Canadian Geotechnical Journal **30**: 935-946.

Been, K. 1980. Stress strain behaviour of cohesive soil deposited under water. Ph.D. thesis, Oxford University, UK.

Beier, N. and Sego, D. 2008. Dewatering of Oil Sands Tailings Using Cross Flow Filtration. 61st Canadian Geotechnical Conference and 9th joint IAH-CNC

Groundwater Specialty Conference, Edmonton, Alberta, 22-24 September 2008. 8 pp

Beier, N., Alostaz, M. and Sego, D. 2009. Natural Dewatering Strategies for Oil Sands Fine Tailings. Tailings and Mine Waste 2009 Conference, Banff, Alberta, 1-4 November 2009. Pp. 845-858.

Beuth, L:, Dykstra, C., and Jacobs, W. 2014. Modelling of hydraulic sand capping on creeping ultra-soft tailings. 4th International Oil Sands Tailings Conference, Edmonton, Ab. 7-10 December 2014, pp. 10.

Blight, G. 2010. Geotechnical Engineering for Mine Waste Storage Facilities. CRC Press, Taylor and Francis Group. Pp 634.

Blight, G., Thomson, R.R., and Vorster, K. 1985. Profiles of hydraulic-fill tailings beaches and seepage through hydraulically sorted tailings. Journal of South African Institute of Mining and Metallurgy. 85(5): 157-161.

Boxhill, L. and Hooshiar, A. 2012. A review of design and operations considerations associated with thickening technologies and their application to oil sand tailings. 3rd International Oil Sands Tailings Conference, Edmonton, Alberta, 3-5 December 2012, pp. 237-247.

Bromwell, L.G. 1984. Consolidation of mining wastes. Sedimentation consolidation models: predictions and validation: In proceedings of a symposium, R.N. Yong and F.C. Townsend Eds, ASCE, Newyork: 275-295.

Burger, R. and Concha, F. 2001. Settling velocities of particulate systems: 12. Batch centrifugation of flocculated suspensions. International Journal of Mineral Processing 63 (2001): 115-145.

Burns, R., Cuddy, G., and Lahaie, R. 1993. Dewatering of fine tailings by natural evaporation. Fine Tailings Symposium, Environment Canada, Edmonton, Alberta, 4-7 April 1993, 36 pp.

Bussiere, B. 2007. Colloquium 2004: Hydrogeotechnical properties of hard rock tailings from metal mines and emerging geoenvironmental disposal approaches, Canadian Geotechnical Journal, 44(9): 1019-1052.

Carrier, W.D. III, Scott, J.D., Shaw, W.H., and Dusseault, M.B. 1987. Reclamation of Athabasca oil sand sludge. Proceedings of Geotechnical Practice for Waste Disposal, Ann Arbor, MI., 15-17 June 1987 pp. 377-391.

Chu, A., Paradis, T., Wallwork, V., and Hurdal, J. 2008. Non segregating tailings at the Horizon oil sands project. First International Oil Sands Tailings Conference, Edmonton, Alberta, 7-10 December 2008. 10 pp

Davies, M.P., and Rice, S. 2001. An alternative to conventional tailings management - "dry stack" filtered tailings. Proceedings of the International Conference on Tailings and Mine Waste ‘01, Vail, Colorado, pp 411-419

Dawson R.F., and Sego D.C. 1993. Design concepts for thin layered freeze-thaw dewatering systems. 46th Canadian Geotechnical Conference, Saskatoon, 1993. pp. 283-288.

Dawson, R.F., Sego, D.C., Pollock, G.W., 1999. Freeze–thaw dewatering of oil sands fine tails. Canadian Geotechnical Journal 36(4): 587–598.

Dixon-Hardy, D.W. and Engels, J.W. 2007. Guidelines and recommendations for the safe operation of tailings management facilities. Environmental Engineering Science, 24(5): 625-637.

Environment Canada (EC). 2009. National Climate Data and Information Archive. Accessed May 11, 2009, www.climate.weatheroffice.ec.gc.ca

Engels, J. 2006. An expert management system for surface tailings storage. PhD Thesis, Mining, Quarry and Mineral Engineering, University of Leeds.

Fair, A. 2008. Past, Present, Future – Industry Perspective. First International Oil Sands Tailings Conference, Edmonton, AB, Canada, 7-10 December 2008. Presentation Only.

Fair, A.E. 2012. Canada's Oil Sands: A collaborative approach to addressing oil sand tailings. World Heavy Oil Congress, Aberdeen, Scotland, 10-13 September 2012, 10 pp.

Fair A.E. and Beier, N.A. 2012. Keynote Paper: Collaboration in Canada's Oil Sands: Fluid Fine Tailings Management. In Proceedings of the 3rd International Oil Sands Tailings Conference, Edmonton, Alberta, 3-5 December 2012, pp. 3-12.

Farnand, B. and Sawatzky, H. 1988. Microfiltration dewatering of deoiled tailings pond sludge. Eastern Oil Shale Symposium, Lexington, Kentucky. 30 November to 2 December 1988 Institute for Mining and Minerals Research, University of Kentucky, Lexington, Kentucky.

Fitton, T. 2007. Tailings beach slope prediction. PhD Thesis, School of Civil, Environmental and Chemical Engineering, RMIT University, Melbourne, Australia.

Flintoff, B.C., Plitt, L.R., and Turak, A.A. 1987. Cyclone modeling: A review of present technology, CIM Bulletin, 80: 39-50.

FTFC (Fine Tailings Fundamentals Consortium), 1995. Advances in oil sands tailings research, Alberta Department of Energy, Oil Sands and Research Division, Edmonton, Alberta, Canada.

Gibson, R.E., England, G.L. and Hussey M.J.L. 1967. The Theory of One-dimensional Consolidation of Saturated Clays. I. Finite Non-linear Consolidation of Thin Homogeneous Layers. Géotechnique, 17(3): 261–273.

Imai, G. 1981. Experimental studies on sedimentation mechanism and sediment formation of clay materials. Soils and Foundations, 21(1): 7-20.

Jeeravipoolvarn, S. 2010. Geotechnical behavior of in-line thickened oil sands tailings. PhD Thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, Alberta. pp. 431.

Jeeravipoolvarn, S., Scott, J.D., and Chalaturnyk, R.J. 2009. 10 m standpipe tests on oil sands tailings: long term experimental results and prediction. Canadian Geotechnical Journal, 46:875-888.

Jewell, R.J. and Fourie, A.B. 2006. Paste and thickened tailings – A guide. 2nd Edition. Australian Centre for Geomechanics, Nedlands, Western Australia, Australia. Pp. 257.

Johnson, R.L., Bork, P., Allen, E.A.D., James, W.H., and Koverny, L.1993. Oil sands sludge dewatering by freeze-thaw and evapotranspiration. Alberta Conservation and Reclamation Council Report No. RRTAC 93-8, 247 p.

Kabwe, L.K., Scott, J.D, Sorta, A.R, Wilson, W., Ulrich, A., Jeeravipoolvarn, S., and Kone, M. 2013. Geotechnical characteristics of the mature fine tailings in a 30-year 10-m standpipe experiment. GeoMontreal2013, Montreal, Quebec, 29 September to 3 October 3 2013, pp. 7.

Kolstad, D., Dunmola, A., Dhadli, N., O'Kane, M., Song, J. and Masala, S. 2012. Towards the improvement in geotechnical performance of atmospheric fines drying (AFD) deposits at Shell's Muskeg River Mine. 3rd International Oil Sands Tailings Conference, Edmonton, Alberta, 3-5 December 2012, pp. 411-420.

Konrad, J.M., Morgernstern, N.R., 1980. A mechanistic theory of ice lens formation in fine-grained soils. Canadian Geotechnical Journal 17(4): pp 473–486.

Kupper, A. 1991. Design of Hydraulic Fill. PhD thesis. University of Alberta. Edmonton, Alberta, Canada.

Kynch, G.J. 1952. A theory of Sedimentation. Transactions of Faraday Society, 48: 166-176.

Lahaie, R. 2008. Syncrude Canada Ltd. - New Tailings Concepts. First International Oil Sands Tailings Conference, Edmonton, AB, Canada, 7-10 December 2008 pp. 17. Presentation Only.

Lee, K. 1979. An analytical and experimental study of large strain soil consolidation. DPhil, University of Oxford.

Leoni, M., Karstunen, M., Vermeer, P.A. 2008. Anisotropic creep model for soft soils. Géotechnique, **58**(3): 215-226.

Leroy, J.C.. 1972. How to establish and maintain growth in Canada-cold winters and short growing seasons. In Tailings disposal today. Edited by Aplin, C.L. and Argalls, G.O. Miller Freeman, San Francisco. pp. 411-449.

Longo, S., Francoeur, R., Labelle, M., and Wislesky, I. 2011. Oil sands tailings dewatering – can it be done? Paste 2011, Fourteenth International Seminar on Paste and Thickened Tailings, Perth, Australia, 5-7 April 2011. Pp. 213-224.

Lord, E.R.F. and B.A. Isaac, 1989. Geotechnical investigations of dredged overburden at Syncrude oil sand mine in northern Alberta, Canada. Canadian Geotechnical Journal 26: 132-153.

Lord, E.R.F., D.W. Mimura and J.D. Scott, 1991. Disposal of oil sand fine tails (sludge) in overburden waste dumps. IN: Proceedings of the First Canadian Conference on Environmental Geotechnics, Montreal, Quebec. 129-138

Lord, E.R., Maciejewski, W. Cymerman, G., and Lahaie, R. 1993. Co-disposal of fine tails and overburden utilizing pipeline techniques. Oil Sands – Our Petroleum Future Conference, Edmonton, Alberta, 4-7 April 1993. Pp 20.

Lord, E.R. and Liu, Y. 1998. Depositional and geotechnical characteristics of paste produced from oil sands tailings. Tailings and Mine Waste 1998. Balkema, Rotterdam. pp. 147-157.

Masala, S. 1998. Numerical Solution of sedimentation and consolidation of fine tailings, M.S.c. thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, AB.

Matthews, J.G., Shaw, W.H., MacKinnon, M.D., and Cuddy, R.G. 2002. Development of composite tailings technology at Syncrude. International Journal of Surface Mining, Reclamation, and Environment, 16(1): 24-39.

McPhail, G. 2008. Prediction of the beach profile of high density thickened tailings from rheological and small scale trial deposition data. Proceedings of the 11th International Seminar on Paste and Thickened Tailings Kasane, Botswana, 5-9 May 2008. Pp. 179-188.

MEND. 2001. Mend Manual, Report 5.4.2d, Vol. 4. Prevention and Control. CANMET, Ottawa, Ontario.

Mihiretu, Y., Chalaturnyk, R.J. and Scott J.D. 2008. Tailings segregation fundamentals from flow behaviour perspective. In First international Oil Sands Tailings Conference. Edmonton, AB, 7-10 December 2008, pp. 112-120.

Mikula, R.J., Munoz, V.A., and Omotoso, O. 2008. Centrifuge options for production of "dry stackable tailings" in surface mined oil sands tailings management. In Canadian International Petroleum Conference / Petroleum Society's 59th Annual Technical Meeting, Calgary, AB, Canada, pp. 7.

Miller, W.G., Scott, J.D. and Sego, D.C., 2011. Effect of extraction water chemistry on the consolidation of oil sands fine tailings. CIM Journal **1**(2): 113-129.

Mitchell, J.K. 1960. Fundamental aspects of thixotropy in soils. Journal of soil Mechanics Foundation Divisions American Society of Civil Engineers, 83(3): 19-52.

Mitchell, J.K. and Soga, K. 2005. Fundamentals of Soil Behaviour, 3rd Edition, John Wiley and Sons, Hoboken, New Jersey, USA.

Mittal, H.K. and Morgenstern, N.R. 1975. Parameters for the design of tailings dams. Canadian Geotechnical Journal, 12(2): pp. 235-261.

Mohamedelhassan, E. 2008. Electrokinetic sedimentation and dewatering of clay slurries. First International Oil Sands Tailings Conference, December 7-10, 2008, Edmonton, Alberta, Canada. pp. 153-160.

Morgenstern, N.R. and Scott, J.D. 1995. Geotechnics of fine tailings management; geoenvironment 2000; characterization, containment, remediation, and performance in environmental geotechnics, Geotechnical Special Publication, 46: 1663-1683.

Mundy, P. and Madsen, B. 2009. The development of centrifugal separation technology for tailings treatment. Tailings and Mine Waste 2009, Banff, Alberta, 1-4 November 2009, pp. 397-403.

Neurohr, G. A. 1983. Use of macrophytes for sludge treatment. In Proceedings of the 6th Symposium on wastewater treatment, Montreal, Quebec, 16-17 November 1983. pp. 413-430.

Newson, T.A. and Fahey, M. 2003. Measurement of evaporation from saline tailings storages. Engineering Geology, 70(2003): 217-233. doi:10.1016/S0013-7952(03)00091-7

Nik, R. M., Sego, D.C. and Morgenstern, 2008. Possibility of Using Centrifugal Filtration for Production of Non-Segregating Tailings. First International Oil Sands Tailings Conference, Edmonton, AB, Canada, 7 – 10 December 2008 pp. 200-208.

Nixon, J.F., Morgernstern, N.R., 1973. The residual stress in thawing soils. Canadian Geotechnical Journal 10 (4), 571–580.

Norton, V. and Wilkie, W. 2004. Clarifying centrifuge operation and selection. Chemical Engineering Progress, 100:8. Pp 34- 39.

Pane, V. and Schiffman, R.L. 1985. A note on sedimentation and consolidation. Gėotechnique, 35(1):69-72.

Pollock, G. 1988. Large Strain Consolidation of Oil Sand Tailings Sludge, M.S.c. thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, AB.

Pollock, G. 2004. Consolidation Modeling of Oilsand Tailings. CONRAD Tailings Seminar. Alberta Research Council, November 8, Edmonton, AB

Proskin, S. 1998. A Geotechnical Investigation of Freeze-Thaw Dewatering of Oil Sands Fine Tailings. PhD thesis, University of Alberta.

Qiu, Y., 2000. Optimum deposition for sub-aerial tailings disposal. PhD thesis, University of Alberta.

Qiu, Y. and Sego, D.C., 1998. Design of sub-aerial tailings deposition in arid regions, in Proceedings of the 51st Canadian Geotechnical Conference, pp. 615 – 622.

Qiu, Y. and Sego, D. C. 2006. Optimum deposition for sub-aerial tailings disposal: concepts and theories. International Journal of Mining, Reclamation and Environment, 20:4, 272 — 285

Richardson, J.F., Harker, J.H., and Backhurst, J.R. 2002. Coulson and Richardson's Chemical Engineering Handbook, Volume 2, Particle Technology and Separation Processes. Butterworth Heinemann, 5th Edition.

Ripperger, S. and Altamnn, J. 2002 Crossflow microfiltration – State of the art. Separation and Purification Technology, 26 (2002): 19-31.

Ritcey, G.M. 1989. Tailings Management: Problems and Solutions in the Mining Industry. Elsevier Science Publishers, Amsterdam. pp 970.

Robinksy, E.I. 1978. Tailings disposal by the thickened discharge method for improved economy and environmental control. Tailings Disposal Today. 2nd International Tailings Symposium, Denver Colorado, USA. May 1978. Pp. 75-92.

Sego, D. 1992. Influence of pore fluid chemistry on freeze-thaw behaviour of Suncor Oil Sand Fine Tails (Phase I). Submitted to Reclamation Research Technical Advisory Committee, Alberta Environment, 35p.

Shaw, B., Hyndman, A., and Sobkowicz, J. 2010. Consolidation projections for thickened tailings. In Proceedings of the 2nd International Oil Sands Tailings Conference, Edmonton, Alberta, 5-8 December 2010, pp 253-267.

Sheeran, D. 1993. An improved understanding of fine tailings structure and behavior. Fine Tailings Symposium, Environment Canada, Edmonton, Alberta, 4-7 April 1993, 11 pp.

Schiffman, R.L., Chen, A.T-F, and Jordan, J.C. 1969. An analysis of consolidation theories. Journal of Soil Mechanics and Foundation division, Proceedings of the American Society of Civil Engineers, 95(SMI): pp. 285-312.

Schiffman, R.L., Vick, S.G. and Gibson, R.E. 1988. Beahviour and properties of hydraulic fills. Hydraulic Fill Structures, Eds: DJ Van Zyl and SG Vick. American Society of Civil Engineers, Colorado State University, pp. 166-202.

Silva, M. J.. 1999. Plant dewatering and strengthening of mine waste tailings. PhD thesis. University of Alberta. Department of Civil and Environmental Engineering. 271 p.

Simms. P. and Grabinsky, M. 2004. A simple method for estimating rates of drying and desaturation of paste tailings during surface deposition. Tailings and Mine Waste 2004, 11-13 October 2004, pp 287-292.

Simms. P., Grabinsky, M., and Zhan, G. 2007. Modelling evaporation of paste tailings from the Bulyanhulu mine. Canadian Geotechnical Journal **44**: 1417-1432.

Simms, P., Williams, M.P.A., Fitton, T.G., and McPhail, G. 2011. Beaching angles and evolution of stack geometry for thickened tailings – a review. 14th International Seminar on Paste and Thickened Tailings, Perth Australia, 5-7 April 2011. Pp. 323-338.

Sisson, R., Lacoste-Bouchet, P., Vera, M., Costello, M., Hedblom, E., Sheets, B., Nesler, D., Solseng, P., Fandrey, A. Van kesteren, W., Tamon, A., and Sittoni, L. 2012. An analytical model for tailings deposition developed from pilot scale testing. 3rd International Oil Sands Tailings Conference, Edmonton, AB, 2-5 December 2012, pp 53-63.

Sobkowicz, J.C. and Morgenstern, N.R. 2009. A geotechnical perspective on oil sands tailings. 13th International Conference on Tailings and Mine Waste, Banff, Alberta, 1-4 November 2009. University of Alberta, pp. xvii-xli.

Sobkowicz, J. 2012. The oil sands tailings technology roadmap and action plan: Introduction and key recommendations. 3rd International Oil Sands Tailings Conference, Edmonton, AB, 2-5 December 2012, pp. 13-22.

Somogyi, F. 1980. Large Strain Consolidation of Fine Grained Slurries. Presented at the Canadian Society for Civil Engineering, Winnipeg, Manitoba, May 29-30.

Sparks, T. 2011. Solid-Liquid Filtration. Elsevier, pp 208. http://dx.doi.org.login.ezproxy.library.ualberta.ca/10.1016/B978-0-08-097114-8.01001-7.

Stahl, R.P. 1996. Characterization and natural processes enhancing dry landscape reclamation of fine processed mine wastes. Ph.D. thesis, University of Alberta, Edmonton, Alberta.

Stahl, R.P., Sego, D.C., 1995. Freeze–thaw dewatering and structural enhancement of fine coal tails. American Society of Civil Engineering Journal of Cold Regions Engineering 9 (3), 135–151.

Suthaker, N.N. and Scott, J.D. 1997. Thixotropic strength measurement of oil sand fine tailings. Canadian Geotechnical Journal, 34: 974-984.

Theriault, J.A. Frostiak, J., and Welch, D. 2003. Surface disposal of paste tailings at the Bulyanhulu gold mine, Tanzania. Sudbury 2003, Mining and the Environment III. Sudbury, Ontario, 25-28 May 2003, pp. 1-8.

Toorman, E.A. 1999. Sedimentation and self-weight consolidation: constitutive equations and numerical modeling. Géotechnique, 49(6):709–726.

U.S. Environmental Protection Agency (EPA). 1994. Technical Report. Design and Evaluation of Tailings Dams. Unpublished report # EPA530-R-94-038, US EPA Office of Solid Waste, Washington DC. August 1994. Last accessed May 13, 2013 at http://www.epa.gov/osw/nonhaz/industrial/special/mining/techdocs/tailings.pdf

Usher, S.P. and Scales, P.J. 2005. Steady state thickener modeling from the compressive yield stress and hindered settling function. Chemical Engineering Journal, 111(2005): 253-261.

Vermeer, P.A., Dykstra, C., Jacobs, W., Beuth, L. 2015. Creep-Consolidation Analyses of Alberta Mature Fine Tailings. Prepared for publication.

Vick, S.G. 1990. Planning Design and Analysis of Tailings Dams. BiTech Publishers Ltd. Pp. 369.

Volker, A. 1982. Polders: An ancient approach to land reclamation. Nature and Resources **18**: 2-13.

Wells, P.S. and Riley, D.A. 2007. MFT drying – case study in the use of rheology modification and dewatering of fine tailings through thin lift deposition in the oil sands of Alberta. Paste 2007, Tenth International Seminar on Paste and Thickened Tailings, Perth, Australia, 13-15 March 2007. Pp. 271-284.

Wells, P.S., Revington, A., Omotoso, Ol, 2011. Mature fine tailings drying – technology update. Paste 2011, R.J. Jewel and A.B. Fourie (eds), Australia Centre for Geomechanics, Perth, ISBN 978-0-9806154-3-2.

Wills, B.A. 2006. Wills' mineral processing technology. Elsevier, Great Britain.

Wilson, G.W., Fredlund, D.G, and Barbour, S.L. 1994. Coupled soil-atmosphere modelling for soil evaporation. Canadian Geotechnical Journal **31**: 151-161.

Wilson, G.W., Wickland, B., and Fines, P. 2002. Concepts for co-mixing waste rock and tailings. Symposium on Mining and the Environment: Challenges and Prospects. Rouyn-Noranda, Quebec. 3-5 November 2002.

Wilson, G.W. 2010. Keynote Address: Environmental Dewatering of Fluid Fine Tailings. Second International Oil Sands Tailings Conference, Edmonton, AB, December 5-8, 2010. Presentation only.

Wilson, G.W., Kabwe, L.K., Donahue, R., and Lahaie, R. 2011. Field performance of in-line flocculated fluid fine tailings using thin lift deposition. Mine Closure 2011. Lake Louise, AB, 18-21 September 2011. pp. 473-481.

Wu, S., Sego, D., and Naeth, M. 2010. Evapotranspiration dewatering effect on CT deposits by grasses. Proceedings of the 63rd Canadian Geotechnical Conference, Calgary, AB. 12-16 September 2010. 8 pp.

Xu, Y., Dabros, T., and Kan, J. 2008. Filterability of oil sands tailings, Process Safety and Environmental Protection, 86(4): 268-276.

Yan, D., Parker, T., and Ryan, S. 2003. Dewatering of fine slurries by the Kalgoorlie Filter Pipe. Minerals Engineering, 16: (2003), 283-289.

Yuan, S. and Lahaie, R. 2009. Thickened tailings (Paste) technology and it's applicability in oil sands tailings management. Tailings and Mine Waste 2009, Banff, Alberta, 1-4 November 2009, pp. 829-844.

Zhang, C., Alostaz, M., Beier, N., and Sego, D. 2009. Cross Flow Filtration of Oil Sand Tailings. Tailings and Mine Waste 2009 Conference, Banff, Alberta, 1-4 November 2009. Pp. 799-812.

3 TAILINGS MANAGEMENT SIMULATION MODEL DEVELOPMENT

3.1 INTRODUCTION

Mining and mineral processing ultimately lead to the production of waste by-products including waste rock and a finer grained slurry called "tailings". Management of the tailings and waste rock currently results in environmental challenges and financial burdens for operators. Environmental pressure has increased as of late to reduce the storage of fluid tailings ultimately minimizing risk from releases and leading to the timely reclamation of these deposits.

In a technical publication, the Australian Government, (2007) defined tailings management as "managing tailings over their life cycle, including their production, transport, placement, storage, and the closure and rehabilitation of the tailings storage facility." The selection of a particular tailings treatment train and management method will depend upon the objectives of the operator/regulator or applicable regulations for the mine site. Typically, operators look for efficient and cost effective systems that provide sufficient protection of the environment and satisfy regulations. The TMS must also be dynamic to cope with a tailings facility whose geometry and operational considerations change over the life of the mine (decades). Facilities should be constructed and operated in an orderly fashion to ensure the main objectives of the TMS are achieved and are socially responsible. These objectives may include the following (McKenna 2008; Scott and Lo 1992; Vick 1990):

- Energy efficiency: processes and transportation (pumping of tailings, water and sand slurry);
- Cost effectiveness: minimize disruption to mine operations, ensure adequate storage is available for tailings and overburden materials, minimize disposal active area, reduce/eliminate long term containment of fine tailings; process flexibility and robustness;
- Safety and integrity of impoundment: maximize integrity and stability of the tailings impoundment to ensure no release of tailings;

- Environmental protection: minimize seepage, total disturbed area and detrimental effects to local species and habitats; and
- Reclamation: tailings surface suitable to sustain reclamation activities; create a trafficable landscape at the earliest opportunity to facilitate progressive reclamation

In Alberta's Oil Sands industry, current tailings management practices resulting in continual accumulation of fine tailings has prompted the ERCB to regulate fluid fine tailings through performance criteria. 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. To meet the 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.

3.1.1 Tailings Management Evaluation

Evaluating all the options available to develop a sound management system or understanding the implications of modifications (new technology) or upsets to an existing TMS can be complex, time consuming and expensive. A sound, well thought out TMS will help fulfill the mining industry's commitments to achieve sustainability and to apply the best available technologies to minimize the risk of failures. Each of the underlying processes (discussed in Chapter 2) are typically modeled separately using complex, analytical tools. There are few available models or approaches that look at the tailings management system as a whole system over the life of the mine.

The Alberta Energy Research Institute (AERI) published a study in 2010 on oil sands tailings technologies and practices referred to as the "AERI model" (Devenny 2010). The model was developed to be a screening tool to evaluate alternative technologies. The focus of the model was to determine the fresh water make-up needs and time to reclamation. The study included background information on mineable oil sand projects including tailings and reclamation components. A database of relevant mining and tailings planning parameters was compiled from public sources and review of current operating mines. A "base case" mine and tailings plan was developed as part of the screening study and it represents the early Syncrude Mildred Lake operations. The model was based on a material balance for full mine life cycle. Evaluation among technologies also included a high level economic assessment. Five technologies were assessed including a base case of conventional tailings management creating fine tailings, a paste thickener, CT, and two fine tailings centrifuge scenarios. Only one technology could be simulated at a time (no concurrent technologies). The tailings forecast model was deterministic and based on static ore body components and extraction model. The model assumed final volumes of tailings and did not assess operational issues such as stage requirements and process water management.

Kalantari (2011) developed a simulation model linking long term mine plans with a CT production planning model. The model was initially developed as a deterministic simulation to establish a mining schedule that takes into account the final quantity of tailings produced (CT) and required storage impoundments. It was then extended to a stochastic framework to capture and quantify uncertainties with the CT production process and their influence on the mine plan. The CT production model was based on Suncor's process (Kalantari 2011). Several limiting assumptions were implemented including fixing the MFT and CT solids contents (C_w, by mass), and the material composition of the cyclone underflow. Stochastic simulations were based on probabilistic distributions for ore rejects (total and sand content), target sand to fines ratio and on-spec CT. Each of these parameters influenced the final mass of CT produced.

Ben-Awuah and Askari-Nasab (2011 and 2012) and Ben-Awuah et al (2012) developed a theoretical optimization model to determine the net present value (NPV) of a mine plan by integrating waste material allocation and dyke construction requirements. Dyke construction requirements were based on a specified design configuration and anticipated final tailings volume. The final storage volume of tailings was calculated using Devenny's (2010) "base case" oil sand ore body. It was assumed that coarse tailings were available for dyke construction or stored as waste when not needed. No tailings management technology was specified for the model (i.e. CT, thickened tailings, end pit lakes, etc.). Additionally, process water storage requirements were not explicitly calculated or included.

A similar model was developed by Badiozamani and Askari-Nasab (2012a and b) that maximized the NPV of a mine plan by integrating the estimated reclamation materials and associated handling costs. The mine plan was constrained by the required tailings capacity and reclamation material needs (i.e coarse tailings and overburden for capping). Tailings volumes were based on Suncor's model for cyclone underflow (coarse tailings) and overflow (fine tailings).

The above models assumed final storage volume requirements and did not assess required operational stage curves/storage requirements. Substitution of alternative tailings technologies or utilization of multiple technologies concurrently was not possible either. These models were focused solely on optimizing the mine plan rather than assessing the merits of particular tailings management technology.

In 2012, a consortium of tailings consultants (CTMC 2012) prepared the Oil Sands Tailings Technology Development Roadmap (Tailings Roadmap) for the Alberta Innovates- Energy and Environment Solutions (AI-EES) (Sobkowicz 2012). The project was conducted to assist regulators and the oil sands industry in creating and implementing technology solutions to meet regulatory goals. Over 550 potential technologies were identified as having potential application in the oil sands industry. Of these technologies, only 101 unique types and variations were further evaluated. A set of tailings reclamation objectives were outlined and the 101 remaining technologies were then assessed against the objectives. The assessment and screening process was based on the quantitative collective judgment from a group of 8 to 10 relevant professionals. The group ranked each technology for its ability to meet the specified reclamation objective. The outcome of the project and assessment process was a suite of technologies that could improve the tailings management practices. A total of nine technology roadmaps were developed based on the suite of technologies. The project recommendations indicate the technology suites should be assessed for individual operators to ensure site specific needs are included. They also identified the need for continual updating and assessment of technologies as they develop. Additionally, the process could be extended to include assessment of tools and chemical amendments on the tailings management process.

3.1.2 Objective

Existing tailings and mine planning models do not take into account the dynamic nature of the tailings management process and rely solely on final tailings volumes or static tailings forecast models. The tailings roadmap project has developed a foundation for assessing tailings technologies but the process was based on qualitative assessments. There currently is no single simulation/assessment model available to assist the planner or regulator in evaluating the management options quickly and efficiently. The objective of this research program is to develop a simulation model (tool) that will guide the tailings planner/operator/regulator through the process of tailings management to attain a practical, economical, and environmentally sound solution. The model will allow the tailings planner to simulate the tailings system over time, demonstrate various outcomes by alternating management practices, and conduct sensitivity analyses. Essentially, the simulation model will be a what-if tool to experiment with various operating strategies or design alternatives to support technology assessment, scenario-analysis, fore-sighting and mine planning (Scaffo-Migliaro 2007; Halog and Chan 2008). Sensitivity/uncertainty analyses could also be used to strategically guide further research and resource expenditure.

This research aims to assist in the assessment of tailings management options through the development of a tailings simulation model. The model will simulate the tailings system behaviour and complex relationships from production to the onset of reclamation. The previous chapters of this thesis described the various tailings management options available to the planner. Also discussed were the various dewatering processes the tailings slurry may undergo. The following chapter will detail the components and approach taken to develop the simulation model.

3.2 TAILINGS MANAGEMENT SYSTEM MODEL

3.2.1 Tailings Management

There are several options available to the mine operator for management of their dry waste and tailings streams as illustrated in Figure 3.1 and described in Chapter 2. As the tailings progress within the tailings management system from formation in extraction through dewatering phases and ultimately to deposition, the material composition may change as well as their physical and mechanical properties (i.e strength, saturated hydraulic conductivity, compressibility, etc.). In the oil sands industry, a ternary plot of sand (>45 μm), fines (<45 μm) and water content can be utilized to characterize and explain tailings behavior as the tailings evolve with time in a tailings management system. This plot is illustrated in Figure 3.2 and explained below. The sand content (S) represents the mass ratio of sand to the total mass of the tailings (sand, fines, water and bitumen). Similarly, fines content (F) is the mass ratio of fines to the total mass. Finally, the water content (w, represented by the horizontal lines) is mass of water over total mass and can be converted to solids content by 1-w (Sobkowicz and Morgenstern 2009). The ternary plot also identifies the limits of slurry pumping (pumpable boundary = thick solid line) and the unsaturated zone (dashed line).

The development of strength is of interest for reclamation and regulatory compliance. Typical fine tailings (i.e. mature fine tailings [MFT]) has an undrained shear strength (S_u) on the order of Pascals (Pa) and increases exponentially as the fines dewater from a liquid state to a solid (Beier et al. 2013). Lines of S_u for MFT (w_L of 45%) are shown on Figure 3.2 (Sobkowicz and Morgenstern 2009). MFT also exhibits a highly thixotropic strength gain as compared with typical clays. Essentially, the material stiffens with time under no change in composition, effective stress or volume and therefore can retard consolidation (Suthaker and Scott 1997).

The rate at which water can be removed from the tailings is represented by the saturated hydraulic conductivity (k). Coarse tailings streams such as whole tailings (WT) have a k of approximately 10^{-3} cm/s (Sobkowicz and Morgenstern 2009). As evident in Figure 3.2, k decreases by four to five orders of magnitude as the F increases and *w* decreases. Similarly, the compressibility of tailings decreases by several orders of magnitude as the fines void ratio decreases (not shown on Figure 3.2).

The starting point on the ternary diagram (Figure 3.2) is the WT zone, a product of the extraction process. In order to meet reclamation and regulatory objectives (Targets for end products, Figure 3.2) suitable for reclamation, tailings must attain solid contents of 75-80 % by mass (Figure 3.2). The WT must endure changes in S_u, k, and compressibility of several orders of magnitude to achieve these solids contents. At this point, the tailings will have developed sufficient long term stiffness and strength (50 to 100 kPa) to support reclamation activities.

Tailings will typically undergo at least three stages of dewatering before they meet their end reclamation targets (Boswell and Sobkowicz 2010). The first stage involves classification of the tailings stream. Here mechanical classification such as a hydrocyclone may be used to separate the WT into a fine grained cyclone overflow (COF) and coarser cyclone underflow (CUF). A thickener may be used to further dewater the COF stream (TT). The WT stream may also be sub-aerially discharged allowing natural segregation to occur. In this case, a coarse beach deposit (BT) and a thin fine tailings stream (TFT) are formed. With time the TFT may settle, and in the case of oil sands, will form MFT. As can be seen from Figure 3.2 and Figure 3.3, significant dewatering is still required to meet the reclamation targets.

The second stage of dewatering includes the various mechanical, chemical and electrical methods described in Chapter 2 and as depicted in Figure 3.2. The feed stock for Stage 2 dewatering may include WT that have not been classified, classified products from Stage 1 (CUF and TT) and re-handled tailings that have previously been deposited (i.e. MFT). The dewatering efficiencies of Stage 2 technologies will inevitably vary, however a couple of trends are evident (Boswell and Sobkowicz 2010). The technologies will dewater the tailings streams to near, but still wet of their liquid limit and the S_u will increase to a few hundred Pa. Some of the potential products of Stage 2 dewatering are depicted on Figure 3.4 (terminus of arrows). According to Boswell and Sobkowicz (2010), tailings with higher liquid limits will generally plot higher on the ternary diagram.

The final stage of dewatering, Stage 3, includes the time dependent and environmental dewatering processes following deposition. This includes sedimentation/consolidation processes, freeze/thaw dewatering, desiccation and evapotranspiration as described in Chapter 2. For coarse grained deposits such as consolidated tailings (CT), Stage 3 dewatering may include self weight consolidation and subsequent loading. For fine grained deposits, depending on the deposition mode (thick versus thin layer), Stage 3 dewatering will include self weight consolidation (thick layer) with subsequent loading and environmental dewatering (thin layer). Following Stage 3 dewatering, the tailings products are near the target end points (Figure 3.2). With time, these processes may ultimately allow the S_u to reach hundreds of kPa. Boswell and Sobkowicz (2010) and Hyndman and Sobkowicz (2010) suggest these strengths are needed to ensure the once fluid products are self-supporting and pose minimal risk of liquefaction.

A tailings management system must also include the construction and operation of the tailings storage facilities (i.e impoundments). This includes the deposition and storage of tailings and storage of process water. The required capacity of the impoundment is then a function of the tailings dewatering processes (described above), the interaction with the environment (i.e. seepage, precipitation, evaporation), and process water demands from the extraction process. This capacity must be balanced with timely construction to ensure appropriate capacity is available including necessary freeboard. Impoundments may be constructed from the tailings themselves or with other mine waste (Vick 1990).

3.2.2 Systems Model

Tailings management systems are typically multifaceted, interrelated and complex. Additionally, mining operations are constantly evolving with time due to inherent changes within ore bodies and subsequently extraction processes as well as economic conditions. Dynamic system analysis and modeling or dynamic simulation modeling (DSM) is therefore appropriate to capture and represent the complex, evolving system through time.

Models of real systems are not intended to capture all the intricacies of the system. Rather they are a simplification of the system within a specified boundary. Using a systems model approach, the complex flow of material portrayed in Figure 3.1 can be simplified into the following systems diagram (Figure 3.6). The TMS systems model called TMSim is composed of a suite of sub-models representing individual components such as the mining and extraction phase, the three tailings dewatering stages, the impoundment and the environment. Critical processes (such as consolidation, evaporation, or dewatering) within each component will dictate mass transfer between components. The systems model will then track the stocks (accumulation/depletion) and flows of mass (solids [mineral including both fine (F) and coarse (C)], water (W), and chemicals) throughout the TMS depicted in Figure 3.6. The following section will outline the each of the sub-models used in the TMS model.

3.3 SIMLUATION MODEL COMPONENTS

3.3.1 Performance Measures/Objectives

The Tailings Roadmap study developed a flexible evaluation framework for assessing the potential of tailings technologies (CTMC 2012). The framework consisted of a series of objectives and sub-objectives related to the mining and reclamation life cycle. Their objectives were based on consideration of the applicable regulations (i.e Energy Resources Conservation Board Directive 074) and aligned with the objectives of stakeholders such as the mining companies, regulators and non-government organizations and are summarized below:

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

These objectives and their applicable sub-objectives were then ranked by their applicability to key indicators used in the evaluation process. The following indicators in Table 3.1, ranked medium to high in relation to the objectives and were given a higher importance weighting during the evaluation process. Based on the objectives provided above and the indicators in Table 3.1, performance measures for the current tailings simulation model have been defined to assess management strategies and new technologies (Table 3.2). The performance measures chosen represent parameters that can be directly quantified for example:

- What is the required impoundment storage volume (for both solids and water)?
- How much material is required to construct impoundments?
- What is the available storage volume?
- How much material is available for construction of impoundments?

- What is the available recycle water volume and quality?
- What is the seepage rate to the environment and its quality?
- Time frame to produce a stable tailings deposit?
- Sensitivity/flexibility of disposal option?

Some of the objectives and key indicators (design and construction complexity, corporate reputation, operating, energy, and closure costs) will require evaluation of the above performance measures in addition to professional judgment and detailed design work and therefore were not incorporated into the model.

The intent of the TMS 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 performance measure and therefore overall success of the tailings management system and would be the target of further research, or more detailed design.

3.3.2 Extraction

The mining and extraction sub-model will define the quantity of concentrate extracted, material rejected at the concentration plant and tailings and waste (overburden and rejects) produced at each time step (Figure 3.7). Inputs to this sub-system are derived from a mine plan and include the ore feed rate (Q_{ore_feed}, tonne/yr), grade (%), mineral content (for oil sands this includes both fines, F_{ore}, % and coarse, S_{ore}, %), moisture content (W_{ore}, %) of the ore, the reject rate (Q_{reject}, tonne/yr) defined as a fraction of the ore feed, and the overburden to ore ratio. Each of these inputs can be a function of time. The amount of concentrate extracted from the ore ($Q_{concentrate}$, tonne/yr) is based on the extraction efficiency (E, %), ore grade and Q_{ore_feed} (Equation 3.1). It can be a user defined function (UDF), a static value or a stochastic function.

[3.1] $$Q_{concentrate} = E\% * grade * Q_{ore\_feed}$$

For oil sands, E% can be based on the Energy and Resource Conservation Board ID 2001-7 (Devenney 2010; Sycrude 2010) and specified as a function of ore grade:

[3.2] $$E\% = 54.1*\text{grade} - 2.5*(\text{grade})^2 - 202.7$$

where grade is the ore grade in percent % valid for ore grades above 7%.

The extraction sub-system is a water-based process; therefore the process water demand ($Q_{process\_water}$, m^3/yr) must also be calculated. The user can define the demand as a function of the ore quality (i.e grade %), the ore feed rate (Q_{ore_feed}), or as a function with time (i.e. monthly rates).

The rate of tailings production (Q_{tails}) defined as a dry tonnes of solids/year is the sum of the mass components into and out of the extraction sub-system (Equation 3.3):

[3.3] $$Q_{tails} = Q_{ore\_feed}(1 - W_{ore}) - Q_{concentrate} - Q_{reject}(1 - W_{reject})$$

where Q_{reject} is the reject rate (tonne/yr), W_{reject} is the moisture content of the reject stream. For oil sands, the fines content (< 45 μm) of the tailings must also be determined (Equation 3.4).

[3.4] $$F_{tails} = \frac{Q_{ore\_feed}*F_{ore} - Q_{concentrate}*F_{concentrate} - Q_{reject}*F_{reject} + Q_{residual\_conc.}}{Q_{tails}}$$

Where $Q_{residual\_conc.}$ (tonne/yr) is the mass of concentrate (i.e. bitumen) that reports to the tailings stream and F_x refers the fines content of stream 'x' (ore, reject, etc.). The volume of water reporting to the tailings stream (Q_{tail_water}, m^3/yr) must also be calculated to determine the C_w or void ratio of the extraction tailings (Equation 3.5).

[3.5] $$Q_{tail\_water} = Q_{process\_water} + Q_{ore\_feed} * \frac{W_{ore}}{S_{g_{water}}} - Q_{concentrate} * \frac{W_{concentrate}}{S_{g_{water}}} - Q_{reject} * \frac{W_{reject}}{S_{g_{water}}} - Q_{mill\_losses}$$

Where $W_{concentrate}$ and W_{reject} are the moisture content of concentrate and reject streams and Q_{mill_losses} (m^3/yr) is the water lost in the mill due to spillage, evaporation, etc.

The user can also define the overburden removal rate ($Q_{overburden}$) as a static or stochastic function, a function of time or a function of the Q_{ore_feed}. If the overburden is to be used for construction purposes (containment dykes), the user can also define the fraction of overburden (OB_{split}) that is deemed suitable for use as construction material (Q_{OB_const}). The remainder of the overburden would be stockpiled on site as waste dump (Q_{OB_waste}).

3.3.3 Stage 1 Classification

Where Stage 1 classification is required, the sub-model will define the relative quantities in each classified stream at each time step based on the users input. For hydrocyclone separation (Figure 3.8) the user must specify functions directly to quantify the classification of the feed tailings stream (Q_{tails}) into a hydrocyclone overflow stream (Q_{COF}, tonne/yr) and an underflow (Q_{CUF}, tonne/yr). For example, using data from the Syncrude Aurora mine (Syncrude 2012), empirical relationships were developed (Appendix 1) to determine the amount of solids reporting to the underflow (Q_{CUF}) including the amount of fines (F_{CUF}) as a function of the F_{ore}. From these values, the amount of cyclone overflow (Q_{COF}) and fines (F_{COF}) is simply the remainder from the feed tailings stream.

An alternative classification method is to allow the material to naturally segregate upon deposition into a coarse grained beach and a dilute slurry containing fine grained particles (Figure 3.9). To determine the solids going to the beach (Q_{beach}, tonne/yr) (Equation 3.6), the user must first specify the fines capture in the beach (FC%). This value can be static, stochastic, or a function of the tailings being deposited (i.e. F_{tails}).

[3.6] $$Q_{beach} = Q_{tails} * F_{tails} * FC\% + Q_{tails} * S_{tails}$$

Where S_{tails} is the coarse fraction of the tailings stream. The solids in the runoff is then simply calculated as:

[3.7] $$Q_{runoff} = Q_{tails} * F_{tails} * (1 - FC\%)$$

The target dry density of the beach (ρ_{dbeach}, tonne/m^3) is then used to calculate the volume of beach. This value must be specified by the user and can be static, stochastic, or a function of the tailings being deposited. Using the appropriate specific gravity and standard mass-volume relationships, the density of the cyclone over flow/underflow or the beach and the runoff can be determined.

3.3.4 Stage 2 Dewatering.

Most tailings management plans will include some form of Stage 2 dewatering involving mechanical, chemical or electrical processes (Figure 3.1). Tailings from extraction (Q_{tails}), Stage 1 dewatering (Q_{COF} and Q_{CUF}), or re-processed tailings (Q_{dredge}, tonne/yr) requiring Stage 2 dewatering will undergo one of or a combination of the following processes prior to depostion:

- Mixing with chemical additives ($Q_{chemical}$, tonne/yr) such as flocculants to enhance further Stage 2 or Stage 3 dewatering (i.e. Shell's atmospheric fines drying process);
- Blending of multiple tailings streams to enhance the depositional behavior and Stage 3 dewatering; and/or
- Undergo physical (centrifuge or filtration) or electrical dewatering prior to deposition and further Stage 3 dewatering.

By-products of the Stage 2 dewatering may include a dewatered tailings stream (Q_{treat_tails}, tonne/yr), liberated water (Q_{treat_water}, tonne/yr), and potentially recovered concentrate ($Q_{treat\_concentrate}$, tonne/yr). A schematic of the Stage 2 dewatering process is included as Figure 3.10.

3.3.4.1 Stage 2 Dewatering Sub-Model

One of the goals of the TMSim model is to evaluate dewatering technologies and their influence on the performance measures. In the Stage 2 dewatering sub-model, the user must specify relationships between the feed properties (for example: F_{tails}, clay content, ρ_{dtails}, pore fluid chemistry, Q_{tails}, etc.) and the final product quantity (Q_{treat_tail} and Q_{treat_water}) and quality (F_{treat_tail}, ρ_{dtreat_tail}, FC%, Stage 3 dewatering behavior, etc.). Depending on the maturity of the technology to be evaluated, the level of detail can vary significantly. Mature technologies may have detailed physics based mathematical models that can be incorporated directly into the TMSim code as a UDF or linked via a spreadsheet or black box software. Less mature technologies are likely based on a combination of physics based and empirical based models and will be coded directly into TMSim as a UDF. These models may include static parameters, stochastic variables, and mathematical functions. The Stage 2 sub-model has been developed to allow for implementation of models directly into TMSim, via spreadsheet models, and external, black box software. The user must specify which option is appropriate for the current simulation and update the selected UDFs as required prior to initiation.

For oil sand tailings, the chemical additive rate ($Q_{chemical}$) is related to the feed tailings loading (i.e. Q_{tails}) and the clay content (<2 µm, C_{tails}) of the tailings stream (Equation 3.8). For extraction tailings, defining a relationship between the fines content of the ore and clay content is possible (Masliyah et al. 2011). However, tailings that have been processed (i.e. Stage 1 dewatering or dredged from an impoundment) these relationships are not valid. In these cases, the clay content can be represented by static or stochastic values as a UDF based on experimental data or professional judgment.

[3.8] $$Q_{chemical} = (Q_{tails} * C_{tails}) * X_{chem}$$

Where X_{chem} is the tonne of chemical required per tonne of clay and C_{tails} is calculated as:

[3.9] $$C_{tails} = f(F_{tails})\ or\ UDF$$

If only the fines content is required, the C_{tails} is equal to the F_{tails}.

Where blending is required as part of the Stage 2 dewatering, like the CT process as Syncrude Aurora Mine (Syncrude 2010), the user must define the mixing ratios and final product targets. For example, a sand to fines mass based ratio, SFR, can be used to calculate the mixing ratio between two streams of tailings and the final product dry density, ρ_{dtreat_tails}, is calculated. This process is included in the TMSim Stage 2 sub-model. Tailings reporting from the Stage 1 not used for cell construction (i.e. cyclone underflow, Q_{CUF}) are mixed with a fine tailings stream (i.e. MFT) dredged from an existing impoundment (Q_{dredge}). The Q_{dredge} required can be calculated knowing Q_{CUF}, the target SFR and the fines content of the two streams (F_{CUF} and F_{dredge}):

[3.10] $$F_{dredge} * Q_{dredge} = \left(\frac{1}{SFR}\right) * (1 - F_{CUF}) * Q_{CUF} - F_{CUF} * Q_{CUF}$$

This equation ignores any sand in the dredged tailings. Sand in the dredged tailings will increase the final SFR, improving the final product. The actual SFR achieved is therefore calculated as:

[3.11] $$SFR = \frac{(1-F_{CUF})*Q_{CUF}+(1-F_{dredge})*Q_{dredge}}{F_{CUF}*Q_{FUC}+F_{dredge}*Q_{dredge}}$$

Using the appropriate specific gravity and standard mass-volume relationships, Q_{treat_tails} and ρ_{treat_tails} can then be determined.

A dewatering sub-model is also included in the TMSim using published data on filtration of oil sands tailings (Xu et al. 2008 and Wang et al. 2010). Alternative dewatering models may be substituted into TMSim with conditional UDFs as required. The current dewatering model is based on the theoretical filtration model proposed by Coulson et al. (1991).

[3.12] $$\frac{t}{V} = \frac{\mu f*SR_F*Cv}{2P_fA_f{}^2}V + \frac{ufL_m}{P_fA_f}$$

Where t is the filtration time (seconds), V is the volume (m^3) of filtrate, μ_f is the viscosity of the filtrate (Pa s), A_f is the filter surface area (m^2), P_f is the filtrate pressure (Pa), SR_F is the specific resistance to filtration (m/kg), C_v is the solids concentration in the suspension (kg/m^3), and L_m is the resistance of the filter medium (i.e. filter paper, m_{-1}). The user must specify the operational constraints of the filtration process (i.e. A_f , P_f and R) and the filtration properties of the tailings (SR_F). The SR_F is a function of F_{tails} and X_{chem} (Equation 3.13) and can be determined experimentally or estimated based on professional judgment.

$$[3.13] \qquad SR_F = f(F_{tails}, X_{chem})$$

Therefore, knowing the F_{tails} and the filtration equipment design, the filtration rate can be calculated from equation 3.12. Using the feed tailings rate (Q_{tails}) and the calculated filtration rate, the volume of filtrate (Q_{treat_water}) can be determined and the properties of the filter cake (Q_{treat_tails} and ρ_{dtreat_tails}).

3.3.4.2 Stage 2 Re-handling or Dredging Sub-Model

Tailings management plans may include re-handling of deposited tailings, therefore a dredging sub-model is required. Due to the tailings ponds size and variability of deposits within, it is very difficult to determine the properties of the dredged tailings *a priori*. Material properties such fines content (F_{dredge}), clay content (C_{dredge}), solids content ($C_{w\ dredge}$) will inherently vary with time as the dredging operation proceeds. Therefore, the user can specify these values as static, stochastic or functions of time based on the level of detail available or required by the simulation. The values can be based on operational data when available or expected ranges based on professional judgment. The dredging rate (Q_{dredge}) is either based on a calculated demand from the Stage 2 or 3 dewatering sub-models (i.e. CT) or specified by the user based on required treatment rate.

3.3.5 Stage 3 Dewatering

The Stage 3 dewatering sub model includes the post deposition, time dependent and environmental dewatering processes mentioned above in section 2 and discussed in Chapter 2. For coarse grained tailings slurries such as CT, Stage 3 dewatering may include rapid sedimentation followed by self weight consolidation and subsequent loading (i.e. capping for reclamation). For fine grained deposits, Stage 3 dewatering can include sedimentation and self weight consolidation which may be concurrent with environmental dewatering such as freeze-thaw or desiccation. The following sections outline the sub models for sedimentation, consolidation, and environmental dewatering.

3.3.5.1 Sedimentation

The first process in the deposit dewatering sub model is sedimentation. Tailings from extraction (Q_{tail}), Stage 1 classification (Q_{COF} and Q_{run_off}), and/or Stage 2 dewatering (Q_{treat_tail}) may undergo dewatering due to sedimentation and discharge water to the water cap (Q_{sed_water}, m^3/yr) and a dewatering tailings stream to the consolidation process (Q_{sed_tail}, m^3/yr). A schematic of the process is presented in Figure 3.11. The first step in this sub-model is to determine if the tailings will actually undergo sedimentation upon deposition into the impoundment. The user must first define the sedimentation characteristics of the various tailings streams (including sedimentation rate and void ratio [e_m] where grain to grain contact is achieved signaling the start of consolidation). If the void ratio of the tailings prior to deposition (e_0) is less than the consolidation void ratio (e_m), no sedimentation will occur. If the sedimentation rate is quite rapid, the deposited tailings may reach the e_m before the end of the global TMSim model time step. In this scenario, the sedimentation sub model will assume an instantaneous transition from the feed tailings density (e_0) to the e_m and calculate the volume of released water accordingly (Q_{sed_water}). For slower sedimentation rates, the sedimentation process must be suitably modelled as follows.

The sedimentation sub model incorporates Masala’s (1998) approach to solve Kynch’s (1952) theory (Figure 3.12). Given the current trend in the mining industry to move away from management plans that create large volumes of fluid tailings (void ratios above e_m therefore undergo sedimentation), towards adopting technologies that create higher density tailings deposits (below e_m), the sedimentation model will be simplified to focus resources to the consolidation/deposition modelling components. The simplification will negate the ability to predict the concentration profile within the suspension layer zone (thickness of h_f, Figure 3.12). However, it will provide a reasonable estimate of the water/suspension interface height (Figure 3.12, Z_f) and the suspension/sediment interface height (H_{sed_tail}). This trade-off is acceptable since the concentration profile in the suspension zone is not a required performance measure for the tailings management simulation.

Equation 3.14 and Figure 3.12 represent Masala's (1998) proposed sedimentation model.

[3.14] $$d\rho_f = \frac{dH_{sed\_tail}(\rho_f - \rho_{sed}) + dZ_f(\rho_w - \rho_f)}{h_f}$$

Where $d\rho_f$ is the change in suspension density (ρ_f) with time, dH_{sed_tail} is the change in sediment height (H_{sed_tail}) with time, ρ_{sed} is the sediment density at e_m, dZ_f is the change in suspension/water interface with time (change in height [H_{water}], and h_f is the suspension thickness. Masala (1998) proposed a simplification can be made by assuming the time rate of change of suspension density ($d\rho_f$) is zero (Equation 3.15)

[3.15] $$dH_{sed\_tail} = \frac{-dZ_f(\rho_w - \rho_f)}{(\rho_f - \rho_{sed})}$$

The model then simplifies to Kynch's theory and provides a computing advantage due to it's simplicity and independence of material models. Therefore, no special requirements for dH_{sed_tail} or dZ_f are required. Simple laboratory settling tests can be used to determine dZ_f and then calculate dH_{sed_tail}. Oliveira-Filho and Van Zyl (2006) also presented this simplification as an acceptable method for evaluating tailings dewatering for engineering design and planning purposes. The above equation 3.15 can also be expressed in terms of void ratio and volume per unit area (equation 3.16) which is more convenient for model calculations (Oliveira-Filho and Van Zyl 2006).

[5.16] $$dH_{sed\_tail} = dZ_f * \left(\frac{1+e_m}{e_0 - e_m}\right)$$

To determine the required sedimentation model output the user must first specify a representative function for dZ_f (m^3/m^2) and convert to Q_{sed_water} in terms of

m^3/yr. Sedimentation columns tests can be used to determine the change in height (dZ_f) or void ratio (e_0 to e_m) with time for the tailings slurry. The void ratio data can then be normalized by a unit mass of solids by equation 3.17.

$$[3.17] \qquad Vol_{slurry} = \left(\frac{1}{S_g}\right) * (1+e)$$

Where Vol_{slurry} is the volume of tailings slurry per unit mass of tailings solids at the current void ratio, e, with a specific gravity of S_g. From the normalized data set the user can now extract a function of $Vol_{slurry}(t)$ [m^3 slurry/m^3 solids/yr] of the form:

$$[3.18] \qquad Vol_{slurry}(t) = A_1 \ln m + A_2$$

Where A_1 and A_2 are fitting parameters and *m* is the time since deposition in years. Then, Q_{sed_water} (m^3/yr) can be determined by subtracting $Vol_{slurry}(t)$ from the initial volume of tailings (Vol_{slurry}) at time zero (void ratio e_0) and multiplying by the current mass of tailings deposited (i.e. Q_{tail}) (Equation 3.19).

$$[3.19] \qquad Q_{sed\_water} = \left[\left(\frac{1}{S_g}\right) * (1+e_0) - A_1 \ln m + A_2\right] * Q_{tail}$$

Substituting Q_{sed_water} for dZ_f in equation 3.16, the user can now calculate Q_{sed_tail}.

To this point, the sedimentation model equations represent batch settling or quiescent conditions (no active filling). However, since tailings will be continuously added to the impoundment, the model must account for release water and sedimented tailings from previously deposited layers (i.e. previous time steps). Thus, the sedimentation sub model will treat the total amount of tailings added in a given time step as one layer. Then, the total amount of $Q_{sed\_water,\, t}$ or $Q_{sed\_tail,\, t}$ will simply be the sum from each layer, n (Equations 3.20 and 3.21):

$$[3.20] \quad Q_{sed\_water,t} = \sum_{n=1}^{t} \left[\left(\frac{1}{S_g}\right) * \left(1+e_{0,n}\right) - A_{1,n} \ln m_n + A_{2,n}\right] * Q_{tail,n}$$

$$[3.21] \quad Q_{sed\_tail,t} = \sum_{n=1}^{t} Q_{sed\_water,n} * \left(\frac{1+e_m}{e_{0,n}-e_m}\right)$$

Where n is the layer number, t is the current time step and m is the time since deposition of layer n. The model will also track the initial void ratio ($e_{0,n}$), sedimentation fitting parameters ($A_{1,n}$ and $A_{2,n}$), time since deposition (m_n) and the mass of tailings deposited ($Q_{tail,n}$) for each layer.

3.3.5.2 Consolidation

Following sedimentation, tailings (Q_{sed_tail}), at a void ratio of e_m, may undergo the process of self weight consolidation and subsequent volume change (Q_{consol_tail}). During consolidation, water may be released from the tailings deposit to the water cap ($Q_{consol\_release}$) and/or lost to the base of the tailings deposit as seepage ($Q_{basal\_seepage}$), depending on the drainage conditions. A simple schematic of the process is presented in Figure 3.13. To simulate the consolidation process, an appropriate consolidation theory is required, such as those discussed in Chapter 2. The mining industry is looking to technologies and management plans to create high density tailings deposits rather than generate large volumes of low density tailings. Therefore, a finite strain theory such as Gibson et al. (1967) would be suitable for modeling higher density tailings deposits. This negates the requirement to model coupled sedimentation-consolidation processes. Additionally, a one dimensional model was chosen to simplify the modeling process. Given the degree of detail needed for full three dimensional modeling and the high level nature of the TMSim model, choosing a one-dimensional model is acceptable. The TMSim model will utilize a "black box" concept to simulate the consolidation process, where the black box will be a 3rd party, commercially available software called FSConosl utilizing Gibson et al. (1967) finite strain theory. The 3rd party software will be controlled by and dynamically linked to the TMSim model. The consolidation software will require loading rates, material properties and boundary conditions for the impoundment and will output the change in volume of the deposit (Q_{consol_tail}), $Q_{consol\_release}$, and the void ratio profile of the tailings deposit. The linkage between the TMSim model and the consolidation software is represented in Figure 3.14.

Prior to starting the TMSim simulation, the user must define the material properties (specific gravity) and constitutive relationships (void ratio-effective stress [σ'] and saturated hydraulic conductivity [k] -void ratio) for each of the potential tailings products that may be generated. The most common form of these relationships is a power law formulation (Equation 3.22 and 3.23)

[3.22] $$e = A\sigma'^{B}$$

[3.23] $$k = Ce^{D}$$

where A, B, C and D are fitting parameters defined by the user. Depending on the information available to the user, these parameters may be determined from direct experimental measurements, selected from the literature, or based on professional judgment. Regardless of their source, the final relationships must accurately represent the range of void ratios that are expected.

Since the consolidation sub model is based on one-dimensional analysis, for each impoundment and deposition scenario, the user must specify an appropriate observation point. The observation point should be chosen such that it is representative of the impoundment. For complex deposition scenarios, multiple observation points may be required for each impoundment. Additionally, at each deposition the user must specify the expected boundary conditions such as the incorporation of underdrains and intermediation drainage layers, surcharge loads, or presence of a water cap.

As tailings are deposited into the impoundment at a rate of (Q_{tail}), at a void ratio of e_0, the mass loading rate (q_{tail}; kg/day/m^2) is calculated by the deposition sub model (Section 3.6.2) for the given observation point. The deposition sub model will also determine if a water cap is present and its thickness (H_{water}). Using the tailings material properties, boundary conditions and deposition loading rate, the consolidation software will then return the updated height of the tailings deposit and amount of water released ($Q_{consol\_release}$). This data is then used by the deposition sub-model to calculate the tailings deposit volume at the end of consolidation for the given time step (Section 3.3.6.2).

3.3.5.3 Environmental dewatering

Untreated tailings from extraction (Q_{tail}) or Stage 2 dewatered tailings (Q_{treat_tail}) may also undergo enhanced dewatering upon deposition through environmental processes. These processes include freeze-thaw (Dawson, et al 1999), evaporation (Qiu and Sego 2006; Simms and Grabinsky 2004), and evapotranspiration (Silva 1999). During environmental dewatering, water may be liberated (Q_{env_water}) from the deposited tailings as an evaporative loss (during evaporation and evapotranspiration), as runoff to the water cap (during freeze-thaw), and as seepage to underdrains/interdrains depending on drainage conditions. Figure 3.15 depicts the environmental dewatering process. To simulate these processes, an appropriate theory is again required, such as those previously discussed in Chapter 2.

Modeling of environmental processes inherently requires detailed site information including available energy (radiation), the distribution of energy within the system (albedo effects), and the local meteorological conditions which may impact the evaporation or heat transfer at the surface of the deposit (Newson and Fahey 2003 Simms and Grabinsky 2004). This information may not be readily available or difficult to obtain, therefore the user must determine if the effort required to obtain this information is justified based on the requirements of the simulations. Depending on the level of detail required and site information available, the Stage 3 environmental dewatering sub model can incorporate black box software, spreadsheet models (i.e. DOSTAR; Qiu and Sego 2006) or be directly coded into TMSim. The TMSim currently utilizes "black box" consolidation software (FSConsol) to aid in detailed freeze-thaw dewatering calculations.

The freeze-thaw environmental dewatering model implemented in TMSim is based on the method outlined by Dawson, et al (1999). The freeze thaw model requires tailings properties such as expected thaw strain (ε_{th}) expressed as a fraction, and thawed material constitutive coefficients (A, B, C, and D) for compressibility (Equation 3.22) and saturated hydraulic conductivity (Equation 3.23). Depending on the information available to the user, these parameters may be determined from direct experimental measurements, selected from the literature (Appendix 1), or based on professional judgment.

The user must first decide if the dewatering process is freeze controlled (thaw more than can be frozen) or thaw controlled (freeze more than can be thawed) based on the site climatic data. Then the max depth of tailings that can be frozen/thawed in one year based on expected climatic conditions must be specified *a priori*. This determines the maximum yearly amount to be treated by the freeze-thaw process. For example, at the oil sands mine sites, the operational maximum thickness to be processed by freeze thaw is thaw controlled. Tailings at initial void ratio of e_o, are then deposited during winter months until the maximum volume/depth (Vol_{frozen}/d_{frozen}) is reached. During the spring and summer months, the depth of thaw (d_{thaw}) can be determined from the methods by Dawson, et al (1999) and Martel (1988), outlined in Appendix 1. During the thawing period, water released ($Q_{env\_release}$) and the thawed deposit void ratio (e_{th}) can be calculated from ε_{th} and e_o (Equations 3.24 and 3.25).

[3.24] $$e_{th} = e_o * (1 - \varepsilon_{th})$$

[3.25] $$Q_{env\_release} = \left(\frac{d_{thaw}}{d_{frozen}}\right) * Vol_{frozen} * \varepsilon_{th}$$

Since the thawing process is not coupled with settlement and consolidation, the consolidation calculations only initiate after the entire depth is thawed. According to Proskin (1999), this procedure is a conservative approach and will underestimate the total settlement since thaw is actually not instant and settling occurs as the frozen material melts. Upon thaw, the TMSim will utilize the 3rd party software FSConsol to calculate the settlement and water released due to consolidation from the thawed material properties including specific gravity, e_{th}, and the constitutive equation coefficients A, B, C, and D.

The user must also specify the boundary conditions for the freeze-thaw impoundment. For example, if underdrains and intermediate sand drainage layers are placed between frozen layers. Figure 3.16 depicts these potential boundary conditions. After the first layer of tailings are frozen and thawed, subsequent layers of tailings placed during winter months, will act as surcharge on underlying deposit (Figure 3.16). Upon thaw, the frozen surcharge layer is then removed and modeled as a new consolidating layer.

Using the appropriate site environmental conditions, tailings material properties, boundary conditions and tailings deposition rate (Q_{tail} or Q_{treat_tail}), the environmental dewatering sub-model will return the updated volume of the tailings deposit (Q_{env_tail}) and amount of water released ($Q_{env\_release}$) for the given time step. For environmental dewatering to be effective, little to no water should be maintained on the tailings deposit, unless temporarily used to melt frozen tailings layers (Dawson et al 1999).

3.3.5.4 Strength of the Tailings Deposit

The development of shear strength in a tailings deposit is important to understand in order to evaluate stability of the impoundment or facilitate reclamation activities. Additionally, understanding the shear strength development may be a regulatory requirement as in the oil sands industry. Predicting tailings strength gain is an iterative process where predictions may improve as new information from the laboratory and field is gathered (Masala and Matthews 2010). For planning and feasibility stages, estimates of strength gain may be based on representative values and limited laboratory work. As laboratory testing, field pilot programs and commercial scale implementation progresses, back analysis of the deposit monitoring data can be used to improve and update the strength predictions (Masala and Matthews 2010).

When cohesive tailings are placed hydraulically, normally consolidated (NC) conditions will typically apply. Under these conditions, shear strength is expected to increase with depth and an estimate of the undrained shear strength (S_u) may be estimated by the ratio S_u/σ'_v, where σ'_v is the vertical effective stress (Holtz and Kovacs 1981). Therefore, changes in the deposit strength with time can be tracked by evaluating the consolidation of the deposit (change in σ'_v computed in the Stage 3 dewatering sub model) and using a representative estimate of S_u/σ'_v. Several relationships have been proposed for S_u/σ'_v based of laboratory and field data. Shiffman et al. (1988) proposed the following equation 3.26:

[3.26] $$\frac{S_u}{\sigma'_p} = 0.11 + 0.0037 \times I_p$$

Where I_p is the plasticity index of the soil. For soft soils, Mesri (1989) reasons the ratio is a constant, 0.22. Based on laboratory measurements, Masala and Matthews (2010) present a range of 0.17 to 0.33 for flocculated and thickened oil sand fine tailings with a slight increase to 0.35 based on field measurements. Jeeravipoolvarn (2010) investigated flocculated oil sand fine tailings in both field and laboratory experiments. He found at void ratios below 1.5, the ratio S_u/σ'_v was approximately 0.3, but increased linearly at higher void ratios.

For cohesionless tailings such as hard rock mine tailings or fines depleted coarse oil sand tailings like CT, the shear strength is usually represented by the Mohr-Coulomb effective stress parameters of *c'* (cohesion) and φ' (friction angle) (Bussiere 2007; Qiu 2000). The effective stress shear parameters are usually determined in the laboratory from triaxial or direct shear tests under various loading and drainage conditions (Holtz and Kovacz 1981). Bussiere (2007) reports that most hard rock tailings have an effective friction angle of 30^o to 42^o with cohesion close to zero for drained conditions. For undrained conditions, φ' varied from 14^o to 25^o and *c'* from 0 to 100 kPa. The development of strength in cohesionless tailings will also depend upon the initial density, stress path, and degree of saturation (Bussiere 2007).

To simplify the estimation of strength in fines depleted coarse oil sand tailings like CT, CNRL employs an undrained strength ratio (CNRL 2010). This negates the requirement to determine the pore pressure at failure for a particular scenario. Therefore, at high strains, they assume the strength can be normalized by the effective stress (before shearing was initiated). Based on field in situ measurements of rapidly loaded NST, they assume a ratio of S_u/σ'_v of 0.12.

To determine the shear strength of the tailings deposits for the TMSim model, the user must first specify the appropriate shear strength parameters (S_u/σ'_v, c' and φ') for the various tailings streams and deposits. Then, using the effective stress profiles determined in the Stage 3 dewatering sub-model, shear strength can be calculated for the deposit profile of interest using equation 3.27.

[3.27] $$S_u = c' + \sigma'_v * \tan\varphi'$$

For cohesive tailings or rapidly loaded cohesionless tailings, the user would specify c' as zero and replace $\tan\varphi'$ with the S_u/σ'_v ratio. If limited data are available on the strength parameters, the user can conduct sensitivity analyses by varying the parameters within reasonable ranges.

3.3.6 Impoundment Sub-Model

For each storage facility within the tailings management plan (i.e. external tailings facility [ETF] or in-pit deposition cells) there will be an impoundment sub-model. This sub-model consists of three interrelated components including:

- the containment dykes, beaches and constructed cells,
- the tailings deposit (tailings from extraction, and/or Stage 1 and 2 dewatering), and
- the water cap.

The impoundment sub model will keep stock of the volume of material (water and solids) stored within the impoundment over time. The various inputs to the impoundment sub model are depicted in Figure 3.17. Available storage ($Vol_{storage}$), maximum berm height ($H_{storage}$), and total impoundment area ($A_{storage}$) for the impoundment is calculated based on the volume of available construction material and pre-determined UDF stage curves (Q_{const_demand}) of the containment dyke/beach/cells (Equation 3.28).

[3.28] $VOL_{storage}, H_{storage}, A_{storage} = UDF(Q_{const\_demand})$

$$and\ Q_{const\_demand} = \sum Q_{beach}, Q_{CUF}, Q_{overburden}$$

3.3.6.1 Impoundment Seepage

Since the impoundment evolves as the simulation progresses, real-time, detailed simulation and evaluation of seepage is not practical. Therefore, simple, first order, theoretical estimations will be defined by the user. This level of detail is deemed sufficient for the needs of TMSim model. Recall, the tailings simulation model is used to compare tailings management technologies and practices and not for detailed design of tailings plans and impoundment design. Sensitivity analyses can be implemented by the user to determine if seepage may influence the performance of a particular strategy or technology. This information can be used to guide further detailed seepage modelling by the user (separate from TMSim). Additionally, tailings impoundments will likely include seepage collection measures (internal drains, perimeter collection ditches, etc) to reduce off site migration of potentially contaminate water (McRoberts, 2008). Therefore, majority of the seepage escaping from the impoundment will likely be returned.

For impoundments with constructed liners, it may be possible to estimate the seepage rates ($Q_{dyke\_seepage}$ and $Q_{basal\_seepage}$) *a priori* based on the methods discussed by Rowe (2005). The user can then specify the calculated seepage as a function of time or pond elevation.

An alternative first order estimate approach may be used to calculate $Q_{dyke\_seepage}$ and $Q_{basal\_seepage}$ using the Darcy equation shown below as equation 3.29 and represented in Figure 3.18 (Holtz and Kovacs, 1981). Rykaart (2002) successfully implemented this approach to estimate $Q_{dyke\_seepage}$ within 15% of measured rates for the Kidston gold mine as part of a mine site water balance study.

[3.29] $Q_{dyke\_seepage} = kiA = k\frac{\Delta h}{L}A$

Where k is the saturated hydraulic conductivity of the dyke (m/s), Δh is the headloss (m), L is the seepage path length (m) and A is the area of the seepage zone (m^2). In Rykaart's (2002) water balance, the area of the seepage zone was based on the area of the seepage drain normal to the direction of seepage (height of seepage drain * length of drain).

An alternative method presented by Chapuis and Aubertin (2001) can be used to estimate seepage from homogeneous dykes. A simplified expression can be developed (Equation 5.30) by generalizing the results of several seepage analyses with different geometries, saturated hydraulic conductivities and pond levels.

[3.30] $$\frac{Q_{dyke\_seepage}}{k_{dyke}} = \alpha 1 + \frac{\alpha 2 \Delta h^2}{L} + \alpha 3 \left(\frac{\Delta h^2}{L}\right)^2$$

Where Δh is the hydraulic difference in head between the pond level and the toe of the dyke or filter drain and L is the horizontal difference between the water level on the dyke and the nearest point to a toe drain or collection ditch. For saturated hydraulic conductivities ranging from 10^{-6} to 10^{-8} m/s and for dyke heights from 5 to 50 m, Chapuis and Aubertin (2001) determined the coefficients for equation 5.30 are α1 = 0, α2 = 0.60, and α3 = -0.006. If a dyke configuration is different than those modelled by Chapuis and Aubertin (2001), the coefficients should be used with caution, or the analyses should be repeated using the appropriate conditions.

Rykaart (2002) also used Darcy's equation to calculate seepage loss into the base of the impoundment ($Q_{basal\_seepage}$). Wels and Robertson (2003) also utilized this method for a conceptual water balances of a tailings impoundment. In both cases, k, represented the vertical saturated hydraulic conductivity below the pond and "i" is the gradient at the pond. For Rykaart's (2002) water balance at Kidston, the gradient was estimated at 1.

3.3.6.2 Tailings Deposit

Tailings material from the extraction process (Q_{tail}), Stage 1 dewatering (Q_{COF}, Q_{runoff}) or Stage 2 dewatering (Q_{treat_tail}) will make up the tailings deposit. The impoundment sub model can calculate the resulting surface profile of the tailings poured into the disposal area. The sub model will be linked with the Stage 3 (consolidation) sub model to ensure the current tailings surface incorporates volume change due to consolidation. Output from the sub model will include the topography of the tailings surface for the current time step (H_{tail}), elevation of the water pond (H_{water}), surface area of the tailings layer (A_{tail}) and water pond (A_{water}), and available free board volume ($Vol_{freeboard}$).

To simulate the deposition of tailings, the sub model requires the volume of tailings added for the current time step (Q_{tail}, Q_{COF}, Q_{runoff}, Q_{treat_tail}), volume of water in the impoundment (Vol_{water_cap}), the tailings slope above the water surface (i,BAW), the slope of the tailings below the water surface (iBBW), and finally the tailings discharge location coordinates (Xd, Yd). The discharge can be a single point (eg. central riser) or multi-spigot (eg. line deposition). To calculate the beaching slopes, the user can specify a static value, or a UDF such as Kupper's (1990) or Fitton's (2007) method for determining beach slope of various tailings streams. The user must also specify the topography of a base surface. This surface, for example represents a deposition cell or topography of the valley that the tailings will be deposited into. A schematic of the tailings deposition sub model is presented in Figure 3.19.

The following equation 3.31 is utilized to calculate the tailings surface elevation at each point (x,y) within the deposition area (Barrientos and Barrera, 2001):

[3.31] $$Z(x,y) = H_{tail} - \text{iBAW} * [(x - Xd)^2 + (y - Yd)^2]^{0.5}$$

Where Z(x,y) is the elevation at some point within the deposition area (x,y) and H_{tail} is the calculated deposition height of the tailings surface at the discharge location. If the computed elevation [Z(x,y)] is below the water level (H_{water}) it is then recalculated according to equation 3.32:

[3.32] $$Z(x,y) = H_{tail\_bbw} - \text{iBBW} * [(x - Xd)^2 + (y - Yd)^2]^{0.5}$$

where H_{tail_bbw} is the fictitious discharge height for the tailings below the water level. Using like triangles, H_{tail_bbw} is calculated according to equation 3.33:

$$[3.33] \qquad H_{tail\_bbw} = (\mathrm{iBAW}/iBAW) * (H_{tail} - H_{water}) + H_{water}$$

The above equations are represented in Figure 3.20 on a model cross section of a tailings deposited from a single discharge point.

To determine the final tailings surface elevations, a series of trial discharge elevations (H_{tail}) are computed. Then volume between the previous trial surface and the new "trial" tailings surface is calculated. If the difference between the calculated tailings volume and the specified tailings volume (e.g. Q_{tail}) is not within the prescribed tolerance, a new discharge elevation (H_{tail}) is calculated and the process repeats. A Secant method for determining roots (Chapra and Canale, 1998) was employed to calculate the updated value of H_{tail} for each trial (Equation 3.34).

$$[3.34] \qquad H_{tail(i+1)} = H_{tail(i)} - Z\left(H_{tail(i)}\right) * \frac{\left(H_{tail(i-1)} - H_{tail(i)}\right)}{Z\left(H_{tail(i-1)}\right) - Z\left(H_{tail(i)}\right)}$$

Where $H_{tail(i+1)}$ is the updated value, $H_{tail(i)}$ is the current value, and $H_{tail(i-1)}$ is the previous value. An initial guess for $H_{tail(i-1)}$ based on the user input or H_{tail} from the previous time step is used to determine the first trial surface. If the guess for $H_{tail(i-1)}$ is such that the trial surface is below the previous time step, H_{tail} is increased to ensure the new trial surface is greater than the previous time step. Once tolerance is achieved and a final tailings surface is generated, the water level, H_{water}, must be re-calculated to account for the displacement due to additional tailings in the deposition cell. The Secant method is again used for updating H_{water} for each trial water level. Using a new water level, the entire process repeats by creating a new trial tailings surface and continues until tolerance is met. A flow chart of the model logic is provided in Figure 3.21.

Once the tailings surface [Z(x,y)] is calculated for the given time step, the current tailings layer thickness (or volume of tailings/unit area) is passed to the consolidation sub model. The consolidation sub model then determines the amount of volume change (or settlement) for the current time step. The adjusted tailings height is then passed back to the deposition sub model to adjust the tailings surface based on the calculated amount of settlement. The amount of free board volume ($Vol_{freeboard}$) available between the tailings surface and the spill way elevation (or lowest point on the dyke) is then calculated with the updated tailings surface.

Once the current containment cell is at capacity and tailings deposition ceases, a cap (coarse tailings or overburden material) may be placed on the deposit. Based on the A_{tails} and required depth of cap (H_{cap}) defined by the user, the amount of capping material (Q_{cap}) can be calculated. Tailings streams (i.e Q_{CUF}) not used to construct containment structures can be diverted to fulfill the Q_{cap} demand.

3.3.6.3 Water Cap

Understanding the volume of free water within an impoundment is critical for a successful mine and tailings plan. In arid regions, sustaining sufficient recycle/reclaim water can be challenging. In wet climates, excess water accumulation may reduce the overall available tailings storage capacity. Therefore a water balance sub-model is utilized to calculate the total water volume stored within an impoundment (Vol_{water_cap}). The water balance components include (Figure 3.22):

- Tailings discharge water ($Q_{release}$) due to sedimentation (Q_{sed_water}) plus consolidation ($Q_{consol\_release}$),
- Direct precipitation onto the impoundment (rain and/or snow) ($Q_{precipitation}$),
- Runoff due to precipitation within the impoundment catchment (Q_{run_off}),
- Seepage collection return from dykes ($Q_{seepage\_return}$),
- Water separated during Stage 2 tailings dewatering treatment (Q_{treat_water}),
- UDF miscellaneous flows (may include mine pit dewatering/depressurization water, mine pit runoff, and other waste water streams) (Q_{misc}),
- Pond evaporation (Q_{pond_evap}), and

- Reclaim water for the extraction process ($Q_{reclaim}$).

The total volume of free water available within the impoundment water cap can be calculated using equation (3.35):

$$[3.35] \qquad VOL_{water\_cap} = Q_{release} + Q_{precipitation} + Q_{run\_off} + Q_{treat\_water} + Q_{seepage\_return} + Q_{misc} - Q_{pond\_evap} - Q_{reclaim}$$

Water liberated from the tailings stream in the Stage 2 dewatering process (Q_{treat_water}) may be discharged into the impoundment for storage and is calculated in the Stage 2 sub model. The amount of water released from the sedimentation and consolidation of the tailings deposit ($Q_{release}$) is calculated in the Stage 3 sub model. Climatic data including mean monthly precipitation (mm) as specified by the user is used to calculate the direct precipitation into the impoundment from equation 3.36.

$$[3.36] \qquad Q_{precipitation} = precipitation * A_{storage}$$

For impoundments that include catchments, the run off volume collected within in the impoundment is calculated with equation 3.37:

$$[3.37] \qquad Q_{run\_off} = precipitation * A_{catchment} * run\ off\ factor$$

Where the area of the catchment ($A_{catchment}$) and the run off factor for the catchment are defined by the user.

Seepage that is collected from the impoundment dykes may be returned to the impoundment by specifying the amount of $Q_{seepage\_return}$ either as a UDF or user defined fraction of $Q_{dyke\_seepage}$. It is assumed seepage from the base of the impoundment is not recoverable ($Q_{basal\_seepage}$), therefore not included in $Q_{seepage\_return}$.

Evaporation from the pond surface (Q_{pond_evap}) is calculated using the potential evaporation (Ep) rate as determined from climatic data, a conventional pan evaporation test or climatic based models (refer to Rykaart [2002] or Blight

[2010] for potential models) as specified by the user and the pond surface area (Equation 3.38).

[3.38] $$Q_{pond\_evap} = E_p * A_{water}$$

Water recycled from the impoundment for use if the extraction process ($Q_{reclaim}$) is determined by the process water demand from extraction ($Q_{process\_water}$). The $Q_{reclaim}$ may be limited based on the available water within the impoundment (Vol_{water_cap}) and/or the depth of the water cap (H_{water}). For example, a barge may require a minimum depth of water to operate in, therefore the water level may not be reduced below a certain depth (or volume as determined from UDF stage curves). The function used to determine $Q_{reclaim}$ is presented in equation 3.39:

[3.39] If (Vol_{water_cap} - $min\_VOL_{water\_cap}$) > $Q_{process\_water}$

Then $Q_{reclaim} = Q_{process\_water}$

Else $Q_{reclaim} = Vol_{water\_cap}$ - $min\_VOL_{water\_cap}$

For instances when the $Q_{reclaim}$ is not sufficient to satisfy $Q_{process\_water}$, make up water (Q_{make_up}) is required to fulfill the extraction process water demand. The make-up water can be sourced from fresh water or other ponds on site.

3.3.6.4 Water Cap Chemistry

The chemical composition of the fluid streams in a TMS can have a profound effect on extraction efficiency and tailings behaviour such as the settling characteristics and suspension rheology (Mikula and Omotoso 2006). There are four main chemical mechanisms that govern the distribution of ions between the water phase and solid phase (exchangeable surfaces or mineral precipitates) (Wallace et al., 2004):

- Mixing. During extraction, mixing will occur between the water phase in the ore, process water and any additives. Mixing may also occur during tailings treatment and within the impoundment. Ions initially distributed separately in each stream, mix rapidly and completely to achieve a new chemical equilibrium. Essentially, mixing of the fluid streams will lead to dilution of mass in various components of the TMS. The kinetics of mixing are assumed to be much greater than the other three processes.

- Ion Exchange. Cations may exchange between the water phase and exchange sites on suspended clay minerals. Initially, there is an equilibrium distribution of ions between the water phase and exchange sites on clay surfaces. After mixing of streams, aqueous ion concentrations change and a new distribution will be established. Ion exchange kinetics are relatively rapid for clays with well defined surface exchange sites. Since this process only occurs on surfaces with active exchange sites (clays), it is likely not significant for hardrock mining operations.
- Precipitation. Aqueous ion concentrations after mixing and ion exchange will determine the extent of mineral precipitation. Precipitation is relatively slow compared to ion exchange.
- Dissolution or degassing of CO_2. This process is extremely slow compared to the other three processes. Equilibration with CO_2 will impact pH of the solution and solubility of mineral precipitates.

Integration of all these geochemical processes can be quite complex and data intensive (equilibrium constants and kinetic rates for each process and chemical species). Therefore, the TMSim model will only look at mixing of ions which is assumed to occur instantaneously. Simulating mixing should be sufficient to understand the major trends of chemical mass transfer between the many components of the TMS.

To determine the concentration of species "i" within the impoundment water pond, a mixing function is used (Equation 3.40).

[3.40] $$Ci_{pond}(t+\Delta t) = \frac{Ci_{pond}(t) Vol_{water\_\mathrm{cap}}(t) + \sum_{j=1}^{n} c_{ij} Q_j}{Vol_{water\_cap}(t) + \sum_{j=1}^{n} Q_j}$$

Where $C_{i\ pond}$ is the concentration of chemical species 'i' in the pond, t is the current time step, t+Δt, is the next time step, C_{ij} is the concentration of species "i" in the water stream "j", and Q_j is the flow rate of stream "j" (for example Q_{run_off} or Q_{treat_water}). The concentration of the chemical species in $Q_{reclaim}$ is assumed to be equivalent to the concentration within the water cap ($C_{i\ pond}$).

3.3.7 User Decisions and Logic Conditions

The TMSim model was developed using a suite of sub-models to represent the individual components of a tailings management system such as the extraction plant, tailings dewatering Stage 1, 2 and 3, the impoundment (containment, water cap etc.) and the environment (Figure 3.6). Each of these sub-models were constructed from process-based, empirical or even qualitative formulations based on tentative relationships between parameters. Many of the sub models were also developed using a conditional UDF that must be defined *a prior* by the user. Utilizing conditional UDFs adds flexibility to the TMSim allowing for simple or complex numerical modelling of the TMS and dewatering processes. A summary of the inputs and outputs from each of the sub-models is provided in Figure 3.23. The transfer of material and information between each sub-model presented in Figure 3.23, follow the paths identified in Figure 3.6.

During each time step, single or multiple dewatering technologies (i.e. Stage 1 cyclone, Stage 1 beaching, Stage 2 dewatering, dredging, and Stage 3 dewatering) can be employed during a simulation. The user also has the option to deposit the various tailings streams into more than one location (i.e external tailings facility, beach/cell of containment dykes, or in-pit containment cells). Therefore, the user must not only define the UDFs and variables for each of the sub-models (Figure 3.24), they must also provide input to the various decisions and logic required to allow for the switching between dewatering technologies and deposition points. These decisions and logic points are identified on Figure 3.24 and described in the following sections.

3.3.7.1 Decision 1

The user must specify the first deposition point for the tailings. For example, tailings (Q_{tails}) are to be deposited into an external tailings facility until mining operations have advanced sufficiently and storage space is available in-pit. After determining the location point, the user must now specify if Stage 1 dewatering (cyclone/beaching) will be incorporated in the TMSim and if one or both of the technologies will be used. If Stage 1 dewatering is to be used, the user must define the logic on when it is utilized. For example, when first constructing an impoundment, the tailings may be always beached to the build-up the containment dykes. Therefore, all of the extraction tailings undergo Stage 1 dewatering prior to deposition. Alternatively, only a fraction of the tailings may undergo Stage 1 dewatering with the remainder to Stage 2 or straight deposition. The fraction must be predefined by user or can be calculated (Equation 3.41) based on the required storage ($Vol_{storage}$ of current impoundment).

[3.41] $$Vol_{storage} > \sum_{n=1}^{x}(Q_{tails_{t-n}} + Q_{tail\_water_{t-n}})$$

Where x is the amount of storage required defined by the user (i.e. 3 months, 6 months, etc.) and $Q_{tails\ t-n}$ is the tailings and water from the previous time steps. If the above condition is true, no tailings undergo Stage 1 dewatering. If false, the required amount of Stage 1 tailings (Q_{beach}, Q_{CUF}) are used to construct dykes until the condition is true.

3.3.7.2 Decision 2

If Stage 1 cyclone dewatering is utilized, the user must also specify where the overflow (Q_{COF}) will be deposited or if it will undergo further dewatering by a stage 2 technology (i.e. thickening). For example, the user may specify that all Q_{COF} is deposited in the external tailings facility.

3.3.7.3 Decision 3

Following Stage 1 cycloning, the model must determine if the underflow (Q_{CUF}) is to be used for cell construction (i.e. increase $Vol_{storage}$), dewatered by a Stage 2 technology (CT) or used for capping a tailings deposit. The user can specify the priority to fulfil demand (i.e. containment > capping > Stage 2) or a static fraction is always diverted to capping throughout the mine life.

3.3.7.4 Decision 4

If no Stage 1 dewatering is utilized the user must determine if Stage 2 dewatering will be used prior to deposition. For example, a thickener or filtration technology may be used. If no Stage 2 dewatering is used, the tailings will be deposited straight into an impoundment.

3.3.7.5 Decision 5

Tailings deposited into a settling pond (i.e. beach runoff and cyclone overflow into an ETF) are typically dredged and re-treated due to their low density and slow settling rate. Therefore, the user must decide if Stage 2 dewatering will be utilized once dredged (i.e. to make CT from MFT or to centrifuge MFT) or if the tailings will be transferred to an in pit impoundment cell for long term containment (i.e. end pit lakes).

3.3.7.6 Decision 6

Following Stage 2 dewatering (or tailings without any dewatering), the final deposition point must be identified. If there are multiple impoundments, the user must define the filling sequence/priority (i.e. ETF > in-pit 1 > in pit 2 = in pit 3) and the appropriate stage curves/construction demands for each impoundment. First, the user must also specify the trigger to initiate in-pit impoundment construction. The trigger can be a function of the ore (Q_{ore_feed}) and overburden ($Q_{overburden}$) that has been mined or a static value (i.e. 10 years). Construction of the subsequent impoundments would only initiate once the current impoundment reaches it maximum height (based on stage curves). Filling of subsequent impoundments will only initiate once the current impoundment reaches its maximum capacity based on the stage curves.

3.4 CONCLUSIONS

Several options are available to a mine operator for management of their dry waste and tailings streams. Evaluating all the options available to develop a sound management system or understanding the implications of modifications (new technology) or upsets to an existing TMS can be complex, time consuming and expensive. Therefore, a high level simulation model (TMSim) was developed using publically available data and tailings plans. The simulation model utilizes a multitude of process-based, empirical and qualitative formulations. User inputs (data and parameters) can be static, stochastic or defined as functions of time depending on the level of detail available to the user or required by the simulation exercise. Conditional UDFs are also used to empower the user an option to apply simple or complex numerical modelling of the dewatering processes. The model incorporates the major components of a tailings management system such as the extraction plant, tailings dewatering Stage 1, 2 and 3, the impoundment (containment, water cap etc.) and the environment. A summary of each is provided below:

- The extraction sub-model utilizes the mining plan, ore characteristics, and feedback (process water quantity and quality) from the other sub-models to calculate the quantity of concentrate extracted and tailings produced.
- Stage 1 dewatering sub-model incorporates both cyclone separation and beach deposition based on empirical formulations. The model also provides flexibility for the user to implement new Stage 1 formulations.
- The Stage 2 dewatering sub-model is highly flexible allowing the user to utilize the built in dewatering models or again implement new dewatering models. These models can be simple built in functions or complex 3rd party software models, depending on the level of detail available to the user and the requirement of the simulation exercise.
- The mining industry is adopting management plans that avoid creating large volumes of fluid tailings, therefore, the Stage 3 sedimentation sub-model was simplified to focus resources to the consolidation/deposition

modelling components. The model is based on empirical formulations derived from experimental data.

- The Stage 3 consolidation sub-model incorporates a 3rd party software (FSConsol) as the consolidation modeling engine. The linkage between the TMSim and FSConsol allows the model to dynamically capture evolving tailings properties at each time step. Profiles of void ratio, σ', and S_u with depth are computed at each time step.
- Environmental dewatering processes are also incorporated into the Stage 3 dewatering process. Currently, freeze-thaw dewatering coupled with the Stage 2 consolidation sub-model is included in the TMSim. The user has the flexibility to update the model with other dewatering models including desiccation.
- The impoundment sub-model requires the user to specify the stage curves *a priori*. Using the stage curves and tailings properties, a three dimensional deposition model coupled with the one dimensional consolidation sub-model delivers a representative tailings surface at each time step.
- The impoundment sub-model also tracks the various flows of process water in and out of the impoundment including Stage 2 and 3 release water, precipitation/evaporation, seepage, miscellaneous flows, and extraction reclaim. The chemical quality of the stored process water is also evaluated using a mixing model.
- During each time step, single or multiple dewatering technologies and deposition locations can be employed. The TMSim model provides significant flexibility in deciding when and where tailings are dewatered and deposited based on user input.
- Finally, performance measures were defined to provide a means of evaluating the simulations.

Inherently, there will be limitations to the capabilities of this model. Simplifications and assumptions were required in absence of available data and to facilitate modeling. Additionally, due to the numerous systems and networks involved in the simulation model, the level of detail may be somewhat limited for ease of use. However, the TMSim model will allow the user to identify the properties and processes (i.e. consolidation, solids content, treatment options) that have the greatest influence on the performance measures. Understanding these significant processes and the overall success of a proposed tailings management system will allow for targeted research, additional design and value-added insight which will enhance the tailings management plan.

3.5 TABLES

Table 3.1 Tailings Roadmap Key Indicator – Objective Ranking (CTMC 2012).

Medium Ranking	**High Ranking**
Applicability to Fresh tailings	Fluid fine tailings formation
Applicability to MFT	Tailings Footprint
Impact to Bitumen extraction	Design complexity
System flexibility	Construction complexity
Consolidation	Geotechnical product quality
Water treatment: long-term	Process product quality
Technical uncertainty	Water recovery
Erosion and sedimentation	Water treatment: short-term
Fresh water usage	Geotechnical and seismic hazards/risk
Surface water quality	Groundwater quality
Release of Contaminant of Concern into the environment	Corporate reputation
Ecosystems: long-term	Operating and energy cost
Closure and post-closure cost	
Standards, laws and regulations	

Table 3.2 TMSim Performance Measures.

Performance Measure	Description
$Q_{concentrate}$	Amount of concentrate from extraction
$Q_{process\_water}$	Amount of water required for extraction
$Vol_{storage}$	Available storage volume of current impoundment
Q_{const_demand}	Material required for impoundment construction
Vol_{water_cap}	Volume of water in the impoundment
$C_{[pond]}$	Chemical concentration within the water cap
$Q_{reclaim}$	Volume of water available for use in extraction
H_{tails}	Height of tailings deposit within impoundment
$\sigma'_v = f(z)$ $e = f(z)$ $S_u = f(z,t)$	Vertical effective stress, void ratio and strength profile within the impoundment
$Vol_{freeboard}$	Volume of freeboard
$Q_{dyke\_seepage}$	Seepage from the impoundment

3.6 FIGURES

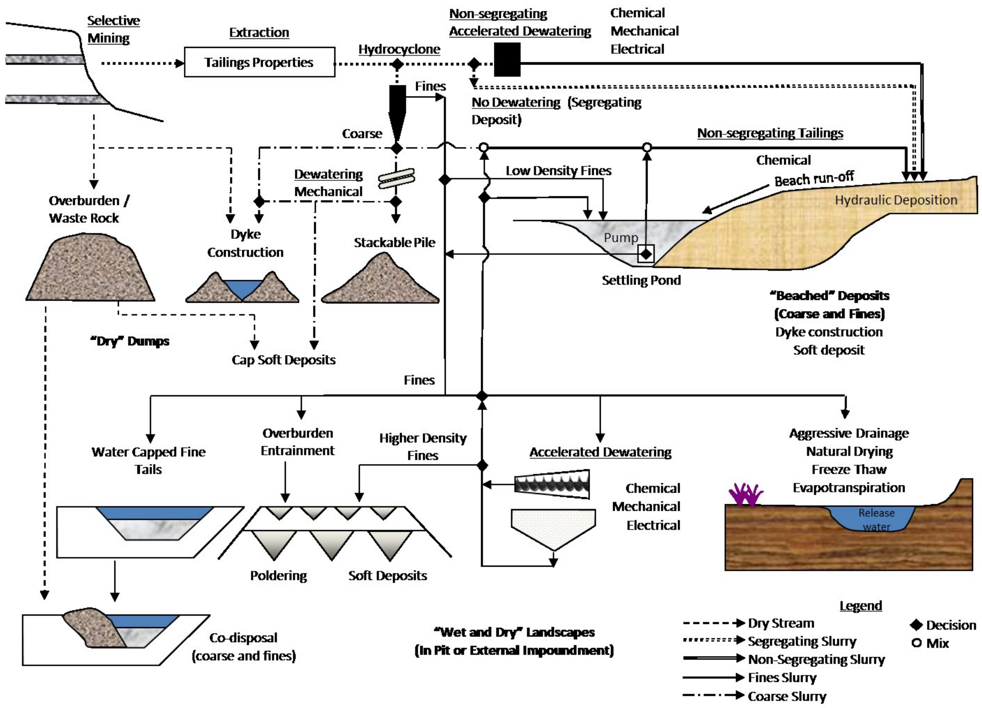


Figure 3.1 Mine Waste Management Strategies (Modified from Sheeran 1993).

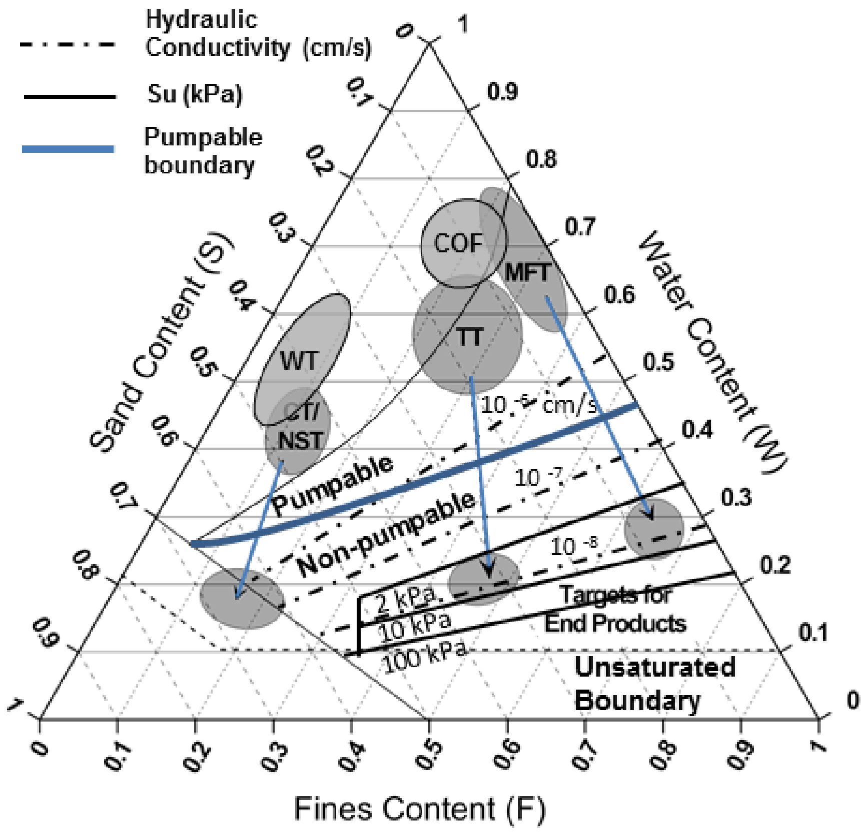


Figure 3.2 Ternary diagram for oil sands tailings (modified from Sobkowicz and Morgenstern 2009). [WT-whole tailings; CT/NST- componsite tailings/non segregating tailings; COF-cyclone overflow tailings; TT-thickened tailings; MFT-mature fine tailings]

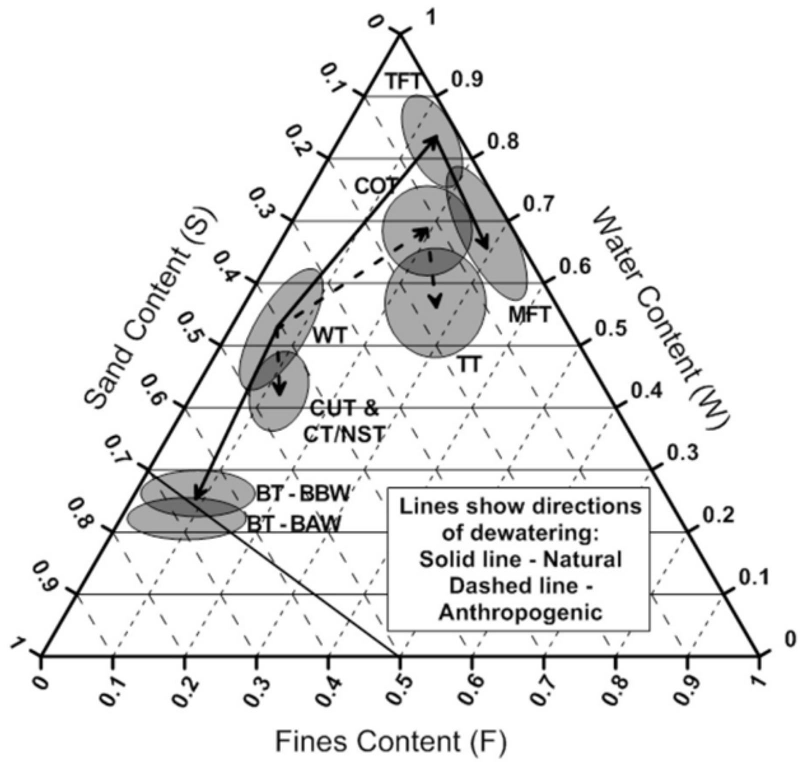


Figure 3.3 Stage 1 dewatering processes for oil sand tailings (modified from Boswell and Sobkowicz 2010). [WT-whole tailings; CUT-cyclone underflow tailings; CT/NST- componsite tailings/non segregating tailings; COT-cyclone overflow tailings; TFT-thin fine tailings; TT-thickened tailings; MFT-mature fine tailings; BT-BBW – beach below water tailings; BT-BAW – beach above water tailings]

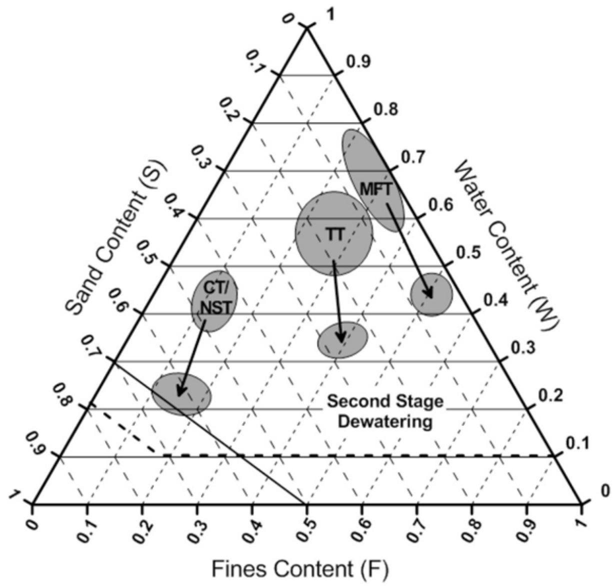


Figure 3.4 Stage 2 dewatering processes for oil sand tailings (modified from Boswell and Sobkowicz 2010). [CT/NST- componsite tailings/non segregating tailings; TT-thickened tailings; MFT-mature fine tailings]

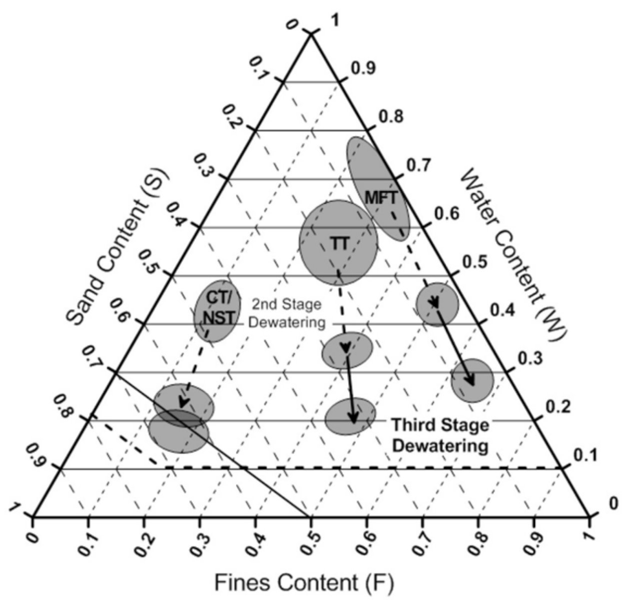


Figure 3.5 Stage 3 dewatering processes for oil sand tailings (modified from Boswell and Sobkowicz 2010). [CT/NST- componsite tailings/non segregating tailings; TT-thickened tailings; MFT-mature fine tailings]

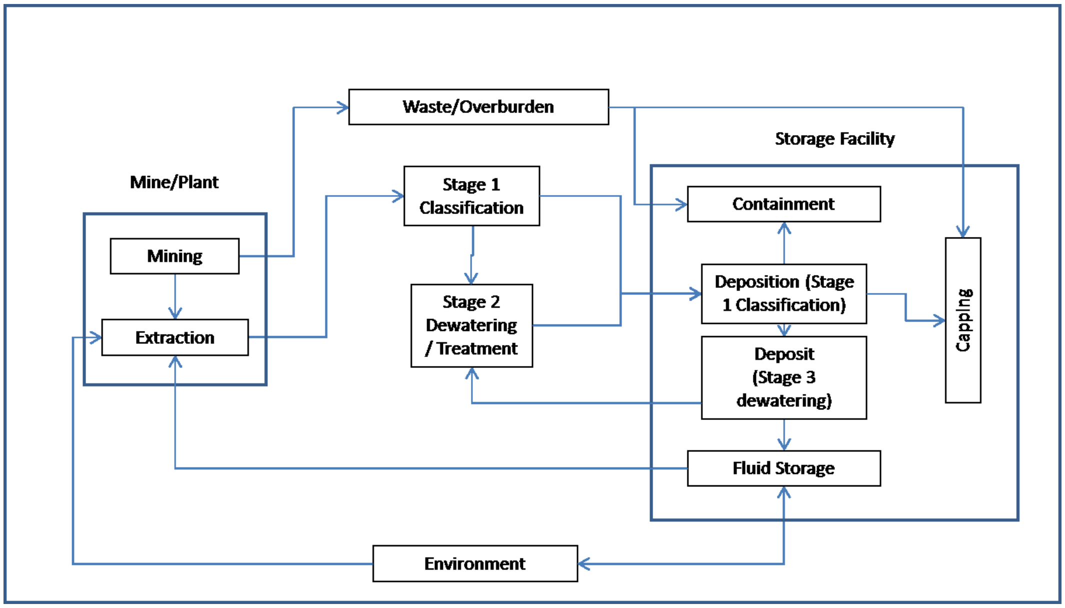


Figure 3.6 Tailings management systems conceptual model.

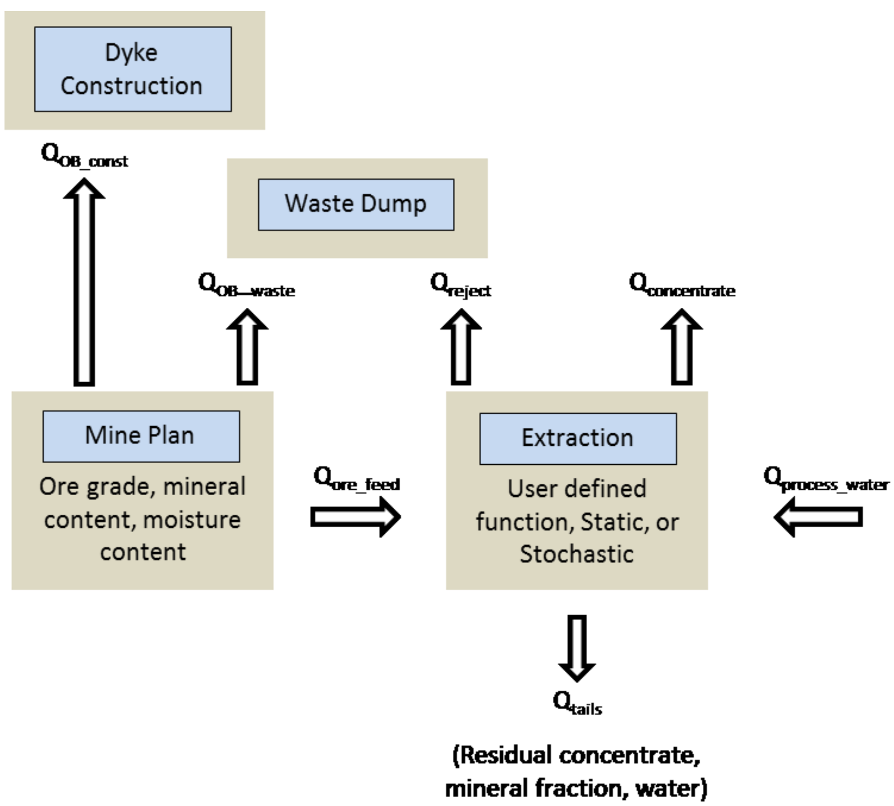


Figure 3.7 Extraction sub-system model.

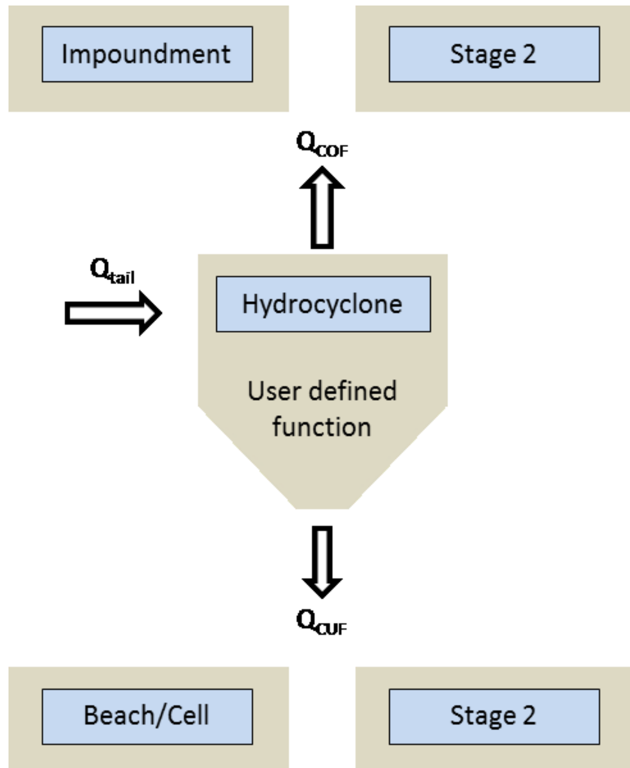


Figure 3.8 Stage 1 anthropogenic classification sub-system (hydrocyclone).

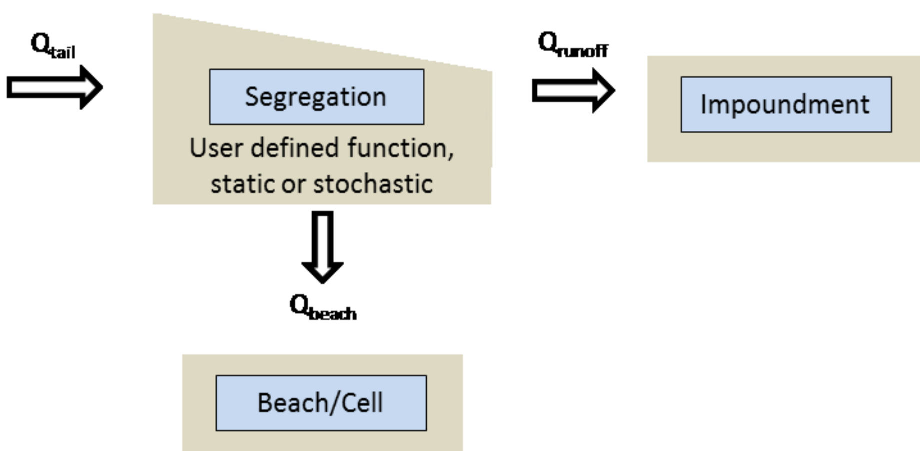


Figure 3.9 Stage 1 natural classification sub-system (segregation upon deposition).

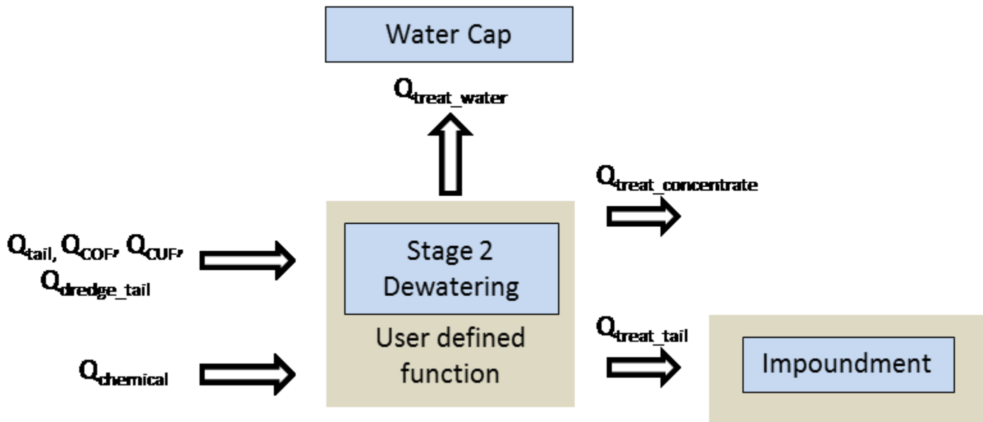


Figure 3.10 Stage 2 Dewatering Sub Model.

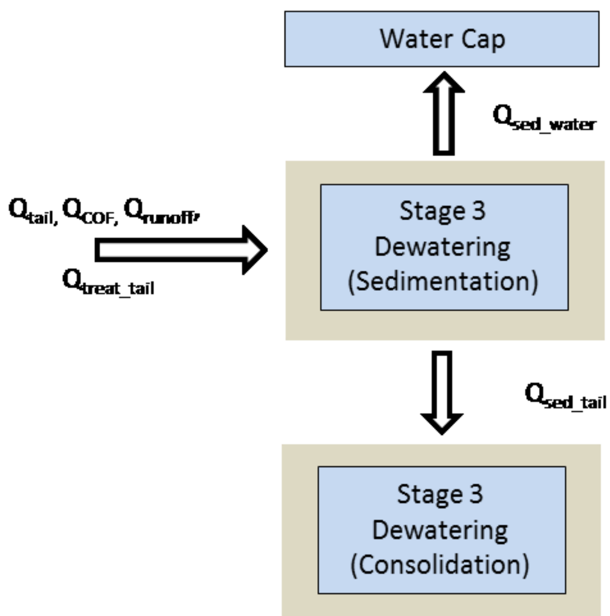


Figure 3.11 Stage 3 Sedimentation sub model.

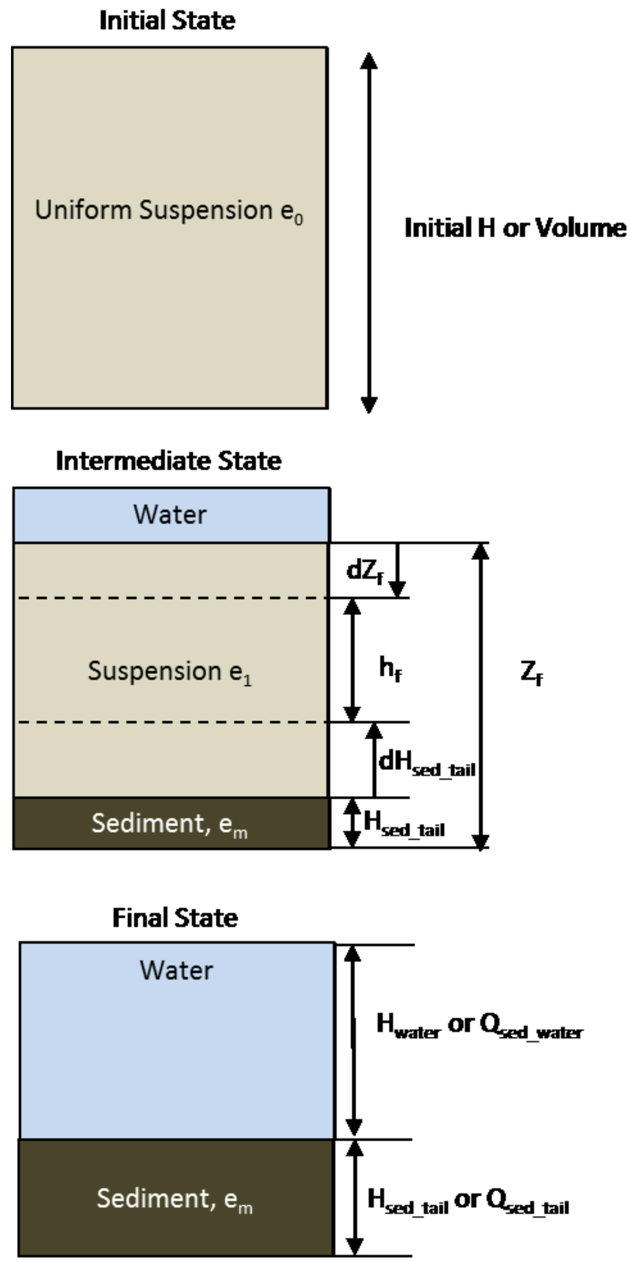


Figure 3.12 Sedimentation modeling – initial, intermediate and final states (modified from Masala 1998).

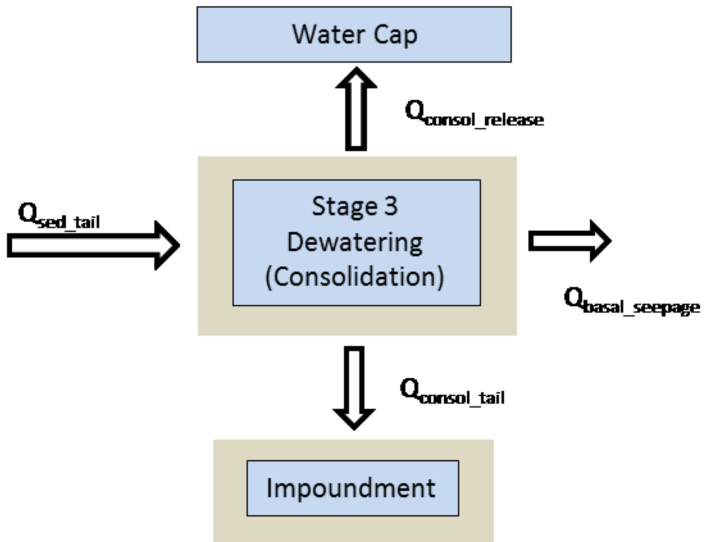


Figure 3.13 Stage 3 Consolidation sub-model.

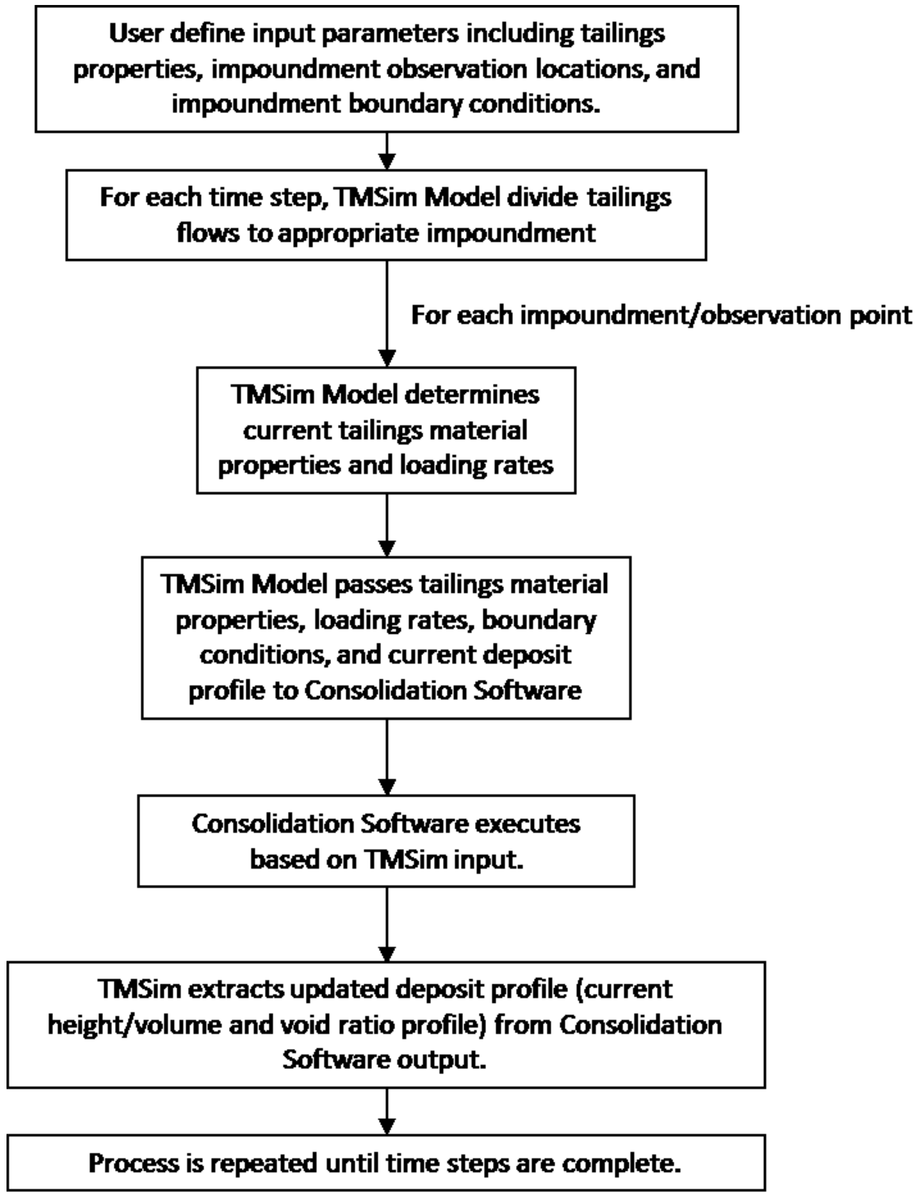


Figure 3.14 Consolidation modeling steps.

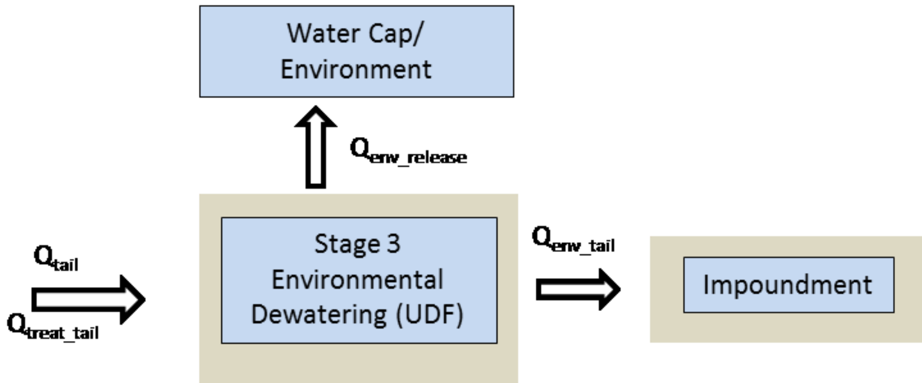


Figure 3.15 Stage 3 Environmental Dewatering sub model.

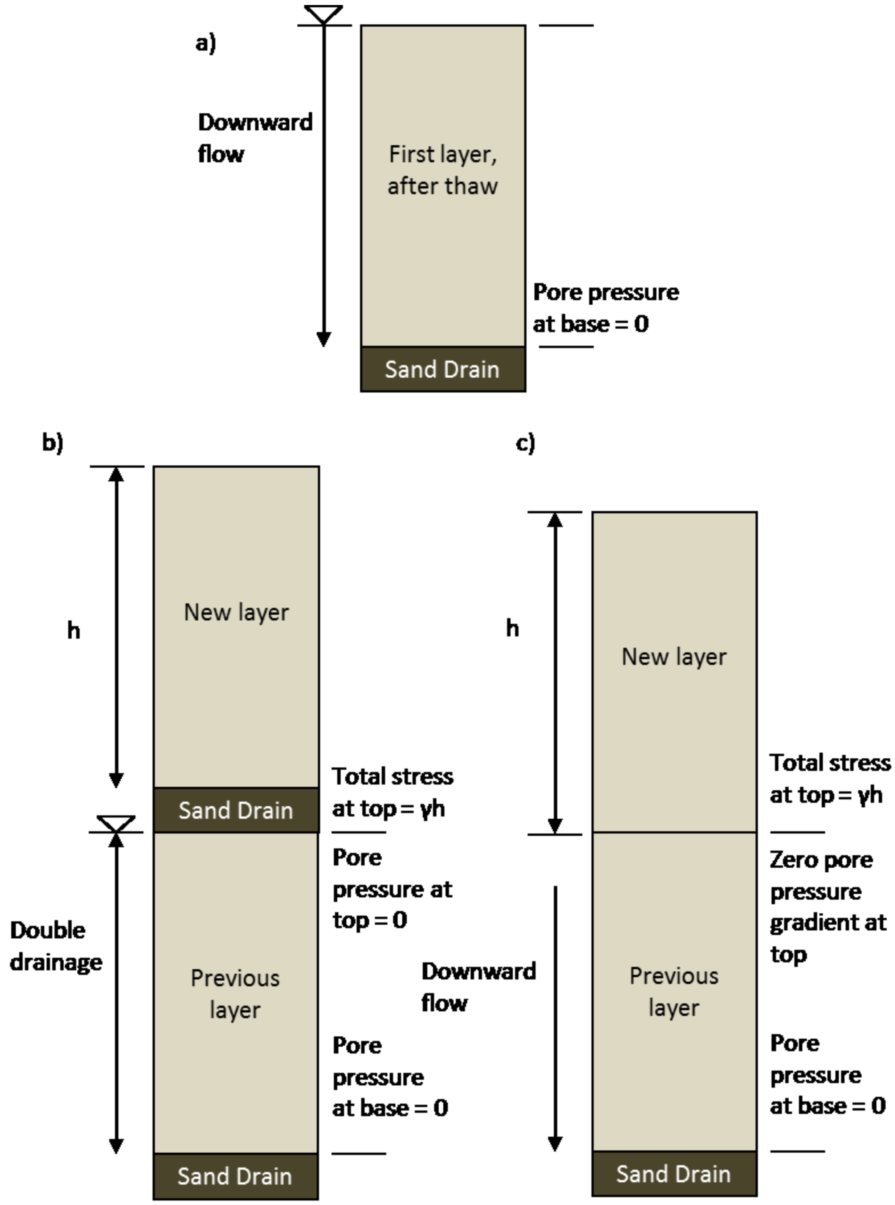


Figure 3.16 Freeze-Thaw consolidation model (modified after Dawson et al 1999). A) Boundary conditions after placement and thawing of first layer. B) Boundary conditions during placement of new layer **WITH** intermediate sand drainage layer. C) Boundary conditions during placement of new layer **WITHOUT** intermediate sand drain and prior to thaw of new layer. Following thaw, boundary conditions are same as A.

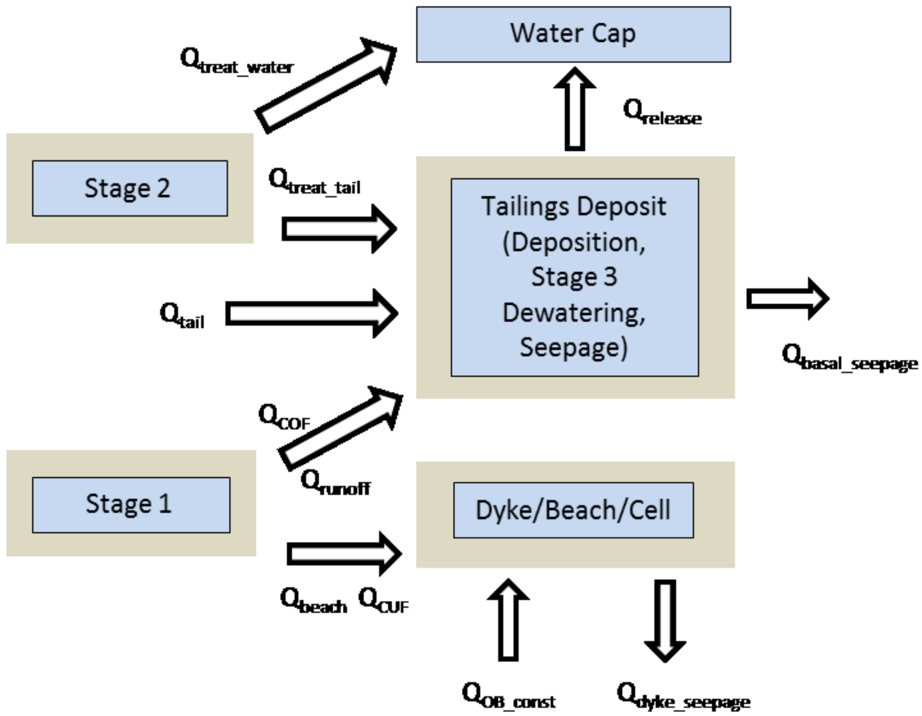


Figure 3.17 Typical impoundment sub-system.

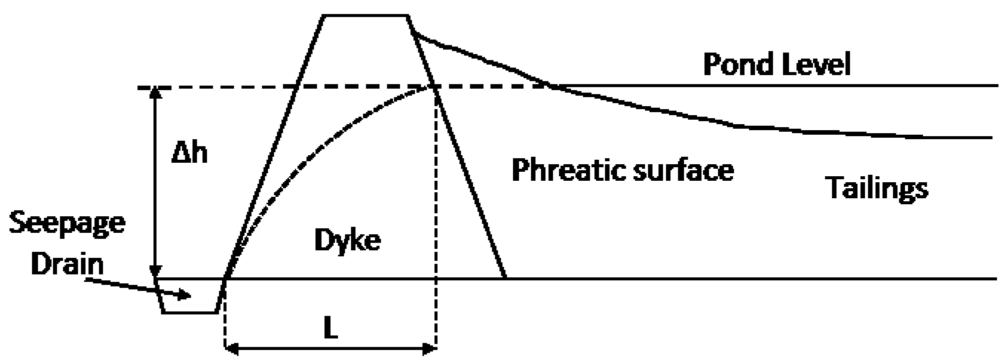


Figure 3.18 Simplified cross section of an impoundment dyke for seepage calculations (modified from Rykaart 2002).

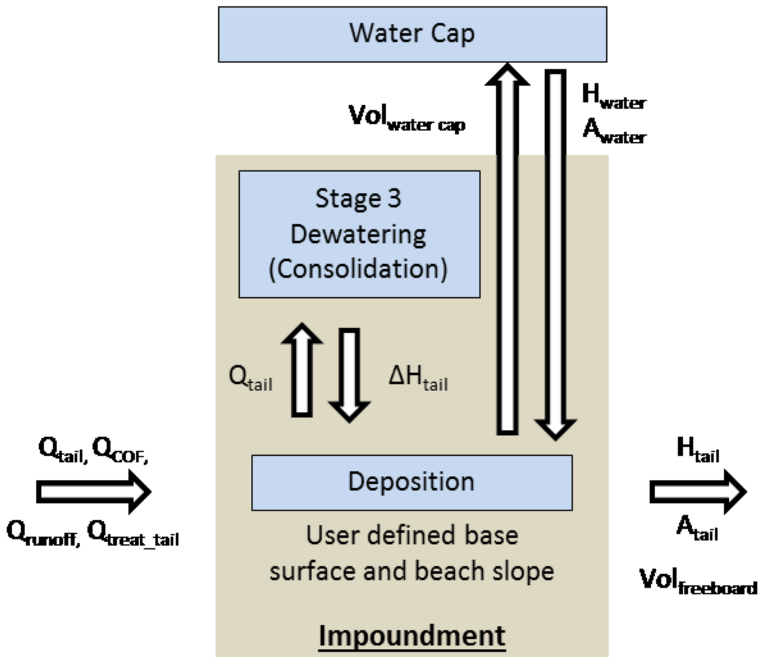


Figure 3.19 Deposition sub model

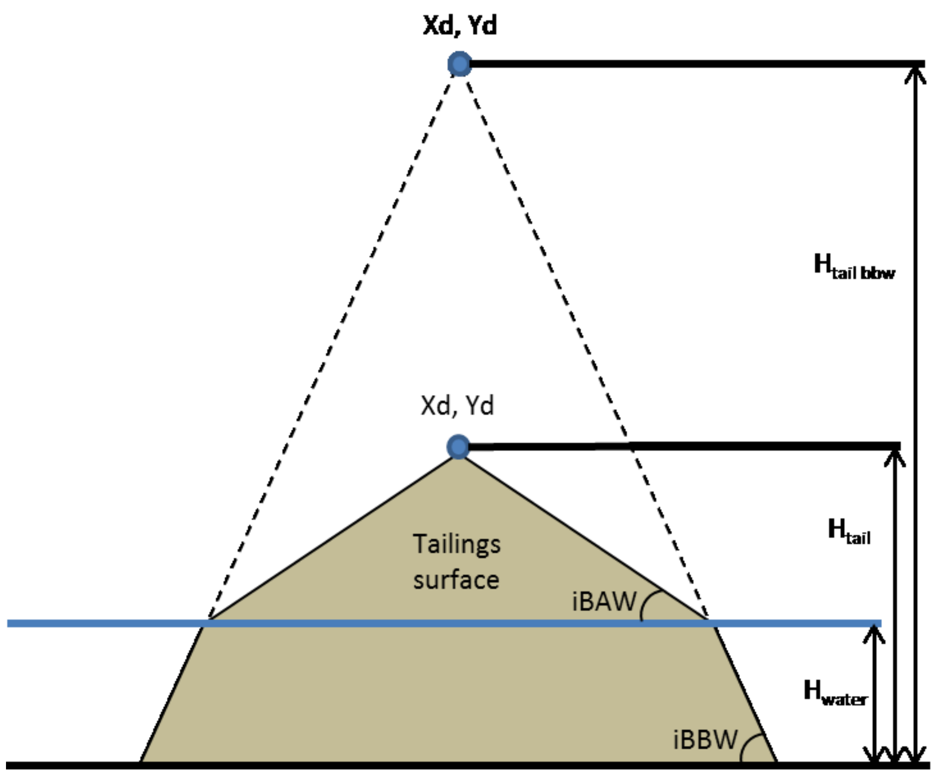


Figure 3.20 Tailings deposition surface profile.

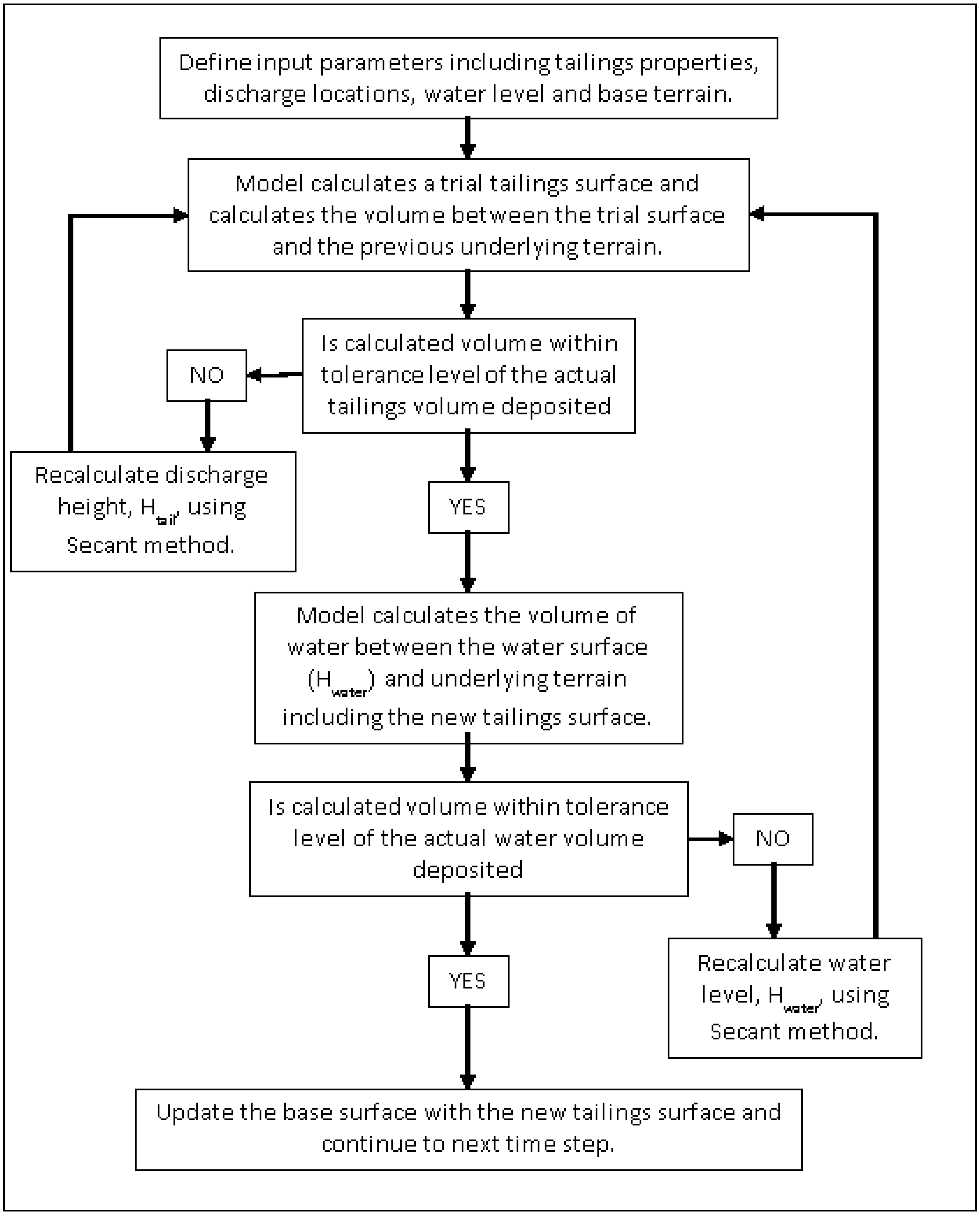


Figure 3.21 Flow chart of algorithm for the tailings depositional model.

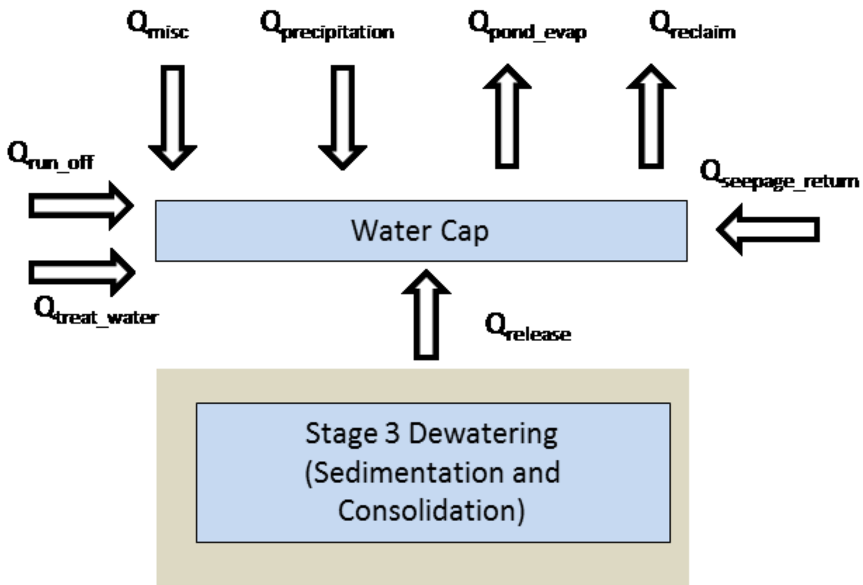


Figure 3.22 Impoundment Free-Water Balance.

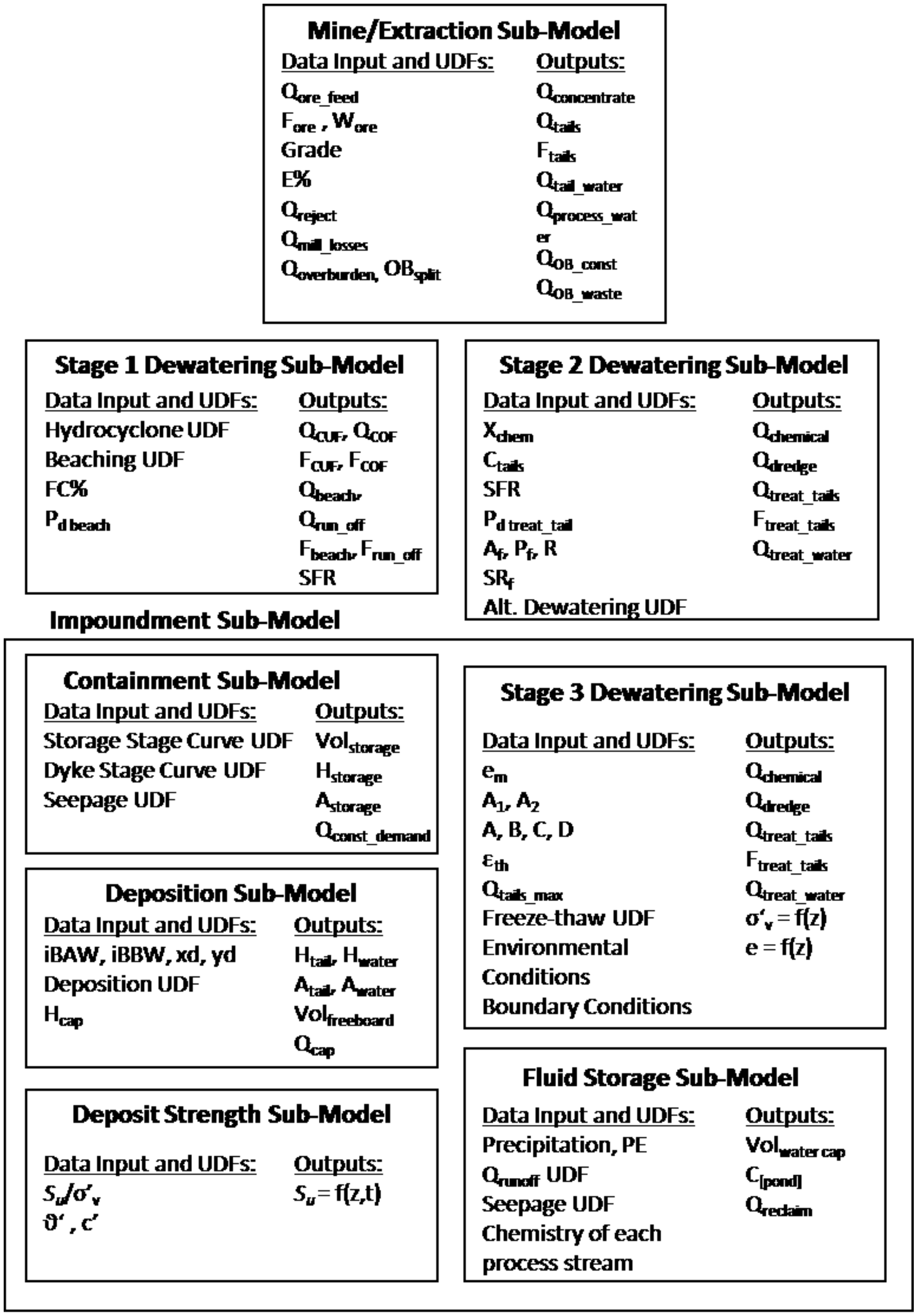


Figure 3.23. Data inputs, UDFs and outputs for each TMSim sub-model.

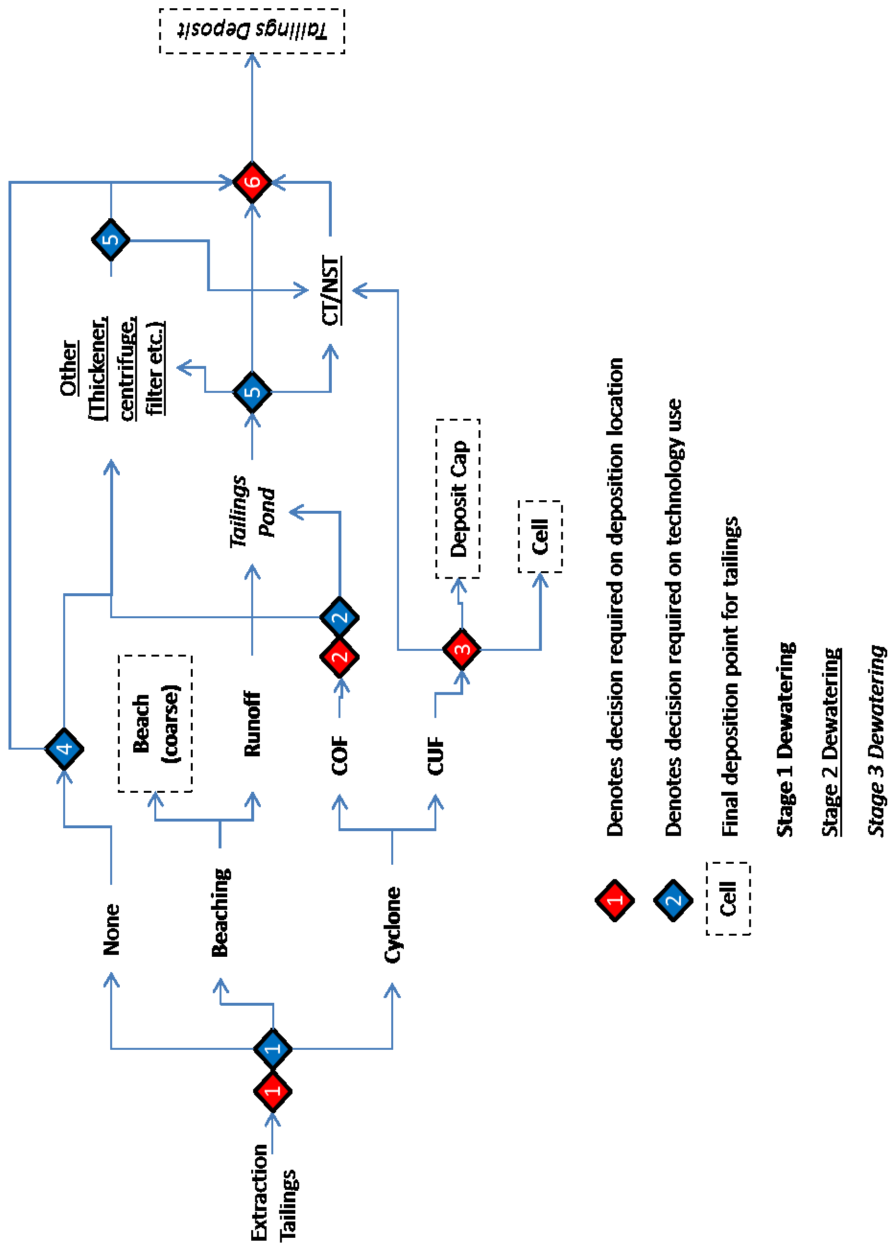


Figure 3.24 TMSim user decision and logic locations.

3.7 REFERENCES

Australian Government. 2007. Tailings Management. Leading practice sustainable development program for the mining industry. Commonwealth of Australia. www.ret.gov.au/sdmining , accessed Aug 8, 2007.

Badiozamani, M. and Askari-Nasab, H. 2012a. Towards integration of oil sands mine planning with tailings and reclamation plans. Proceedings of Tailings and Mine Waste 2012. Keystone, Colorado, October 14-17, 2012, 609-622.

Badiozamani, M. and Askari-Nasab, H. 2012b. An integration of long-term mine planning, tailings and reclamation plans. 3rd international oil sands tailings conference. 103-110.

Barrientos, M. and Barrera, S. 2001. Modeling of tailings impoundment raising. 8th International Conference of Tailings and Mine Waste 2001. Colorado State University, Colorado. Pg 163-170.

Beier, N.A., Wilson, G.W., Dunmola, A., and Sego, D.C. 2013. Impact of flocculation-based dewatering on the shear strength of oil sands fine tailings. Canadian Geotechnical Journal **50**(9): 1001-1007.

Ben-Awuah, E. and Askari-Nasab, H. 2011. Oil sands mine planning and waste management using mixed integer goal programming. International Journal of Mining, Reclamation and Environment, **25**(3);, 226-247,
DOI: 10.1080/17480930.2010.549656

Ben-Awuah, E. and Askari-Nasab, H. 2012. A mathematical programming model for oil sands production scheduling and waste management. Proceedings of Tailings and Mine Waste 2012. Keystone, Colorado, October 14-17, 2012, 593-609.

Ben-Awuah, E., Askari-Nasab, H., and Awuah-Offei, K. 2012. Production scheduling and waste disposal planning for oil sands mining using goal programming. Journal of Environmental Informatics, **20**(1), 20-33.

Blight, G. 2010. Geotechnical Engineering for Mine Waste Storage Facilities. CRC Press, Taylor and Francis Group. Pp 634.

Boswell, J.E.C. and Sobkowicz, J.C. 2010. Environmental assistance for tailings disposal. 63rd Canadian Geotechnical Conference, Calgary, Alberta, 12-16 September 2010, pp. 665-669.

Bussiere, B. 2007. Colloquium 2004: Hydrogeotechnical properties of hard rock tailings from metal mines and emerging geoenvironmental disposal approaches, Canadian Geotechnical Journal, **44**(9): 1019-1052.

Canadian Natural Resources Ltd. (CNRL). 2010. Horizon tailings management plan. 13-RPT-TA-0001 Revision 2. November 2010.

Chapra, S.C. and Canale, R.P. 1998. Numerical Methods for Engineers. 3rd Edition. Mcgraw Hill. 924 pages.

Chapuis, R. and Aubertin, M. 2001. A simplified method to estimate saturated and unsaturated seepage through dikes under steady-state conditions. Canadian Geotechnical Journal, **38**: 1321-1328.

Consortium of Tailings Management Consultants (CTMC), 2012. Oil sands tailings technology deployment roadmaps: Project report. Report prepared for AI-EES and the Oil Sands Tailing Consortium (OSTC). June 29, 2012. 5 Volumes.

Coulson, J. M., J. F. Richardson, J. R. Backhurst and J. H. Harker, "Particle Technology and Separation Processes,"Chem. Eng. Vol. 2, 4th ed. Pergamon, Chapter 7 (1991).

Dawson, R.F., Sego, D.C., Pollock, G.W., 1999. Freeze–thaw dewatering of oil sands fine tails. Canadian Geotechnical Journal **36**(4): 587–598.

Devenney, D.W. 2009. A screening study of oil sand tailing technologies and practices. Report prepared for Alberta Energy Research Institute. May 2009.

Fitton, T. 2007. Tailings beach slope prediction. PhD Thesis, School of Civil, Environmental and Chemical Engineering, RMIT University, Melbourne, Australia.

Gibson, R.E., England, G.L. and Hussey M.J.L. 1967. The Theory of One-dimensional Consolidation of Saturated Clays. I. Finite Non-linear Consolidation of Thin Homogeneous Layers. Géotechnique, **17**(3): 261–273.

Halog, A. and Chan, A. 2008. Developing a dynamic systems model for the sustainable development of the Canadian oil sands industry. International Journal of Environmental Technology and Management, **8**(1): 3-22.

Holtz, R.D. and Kovacs, W.D. 1981. An Introduction to Geotechnical Engineering. Prentice-Hall, Englewood Cliffs, NJ, 733 p.

Hyndman, A. and Sobkowicz, J. 2010. Oil Sands Tailings: Reclamation Goals and the State of Technology. 63rd Canadian Geotechnical Conference, Sept 12 - 16, 2010, Calgary, Alberta, 642 – 655.

Jeeravipoolvarn, S. 2010. Geotechnical behavior of in-line thickened oil sands tailings. PhD Thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, Alberta. pp. 431.

Kalantari, S. 2011. Linkage of Annual Oil Sands Mine Plan to Composite Tailings Plan. MSc Thesis, Department of Civil and Environmental Engineering, University of Alberta.

Kupper, A. 1991. Design of Hydraulic Fill. PhD thesis. University of Alberta. Edmonton, Alberta, Canada.

Kynch, G.J. 1952. A theory of Sedimentation. Transactions of Faraday Society, **48**: 166-176.

Martel, C.J. 1988. Development and design of sludge freezing beds. U.S. Army Cold Regions Research and Engineering Laboratory, CRREL Report 88-20.

Masala, S. 1998. Numerical Solution of sedimentation and consolidation of fine tailings, M.S.c. thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, AB.

Masala, S. and Matthews, J. 2010. Predicting development of undrained shear strength in soft oil sands tailings. Proceeding of the Second International Oil Sands Tailings Conference 5-8 December 2010, Edmonton, Alberta, Canada. pp 31-41.

Masliyah, J., Xu, Z., and Czarnecki, J. 2011. Handbook on theory and practice of bitumen recovery from Athabasca oil sands – Volume 1: Theoretical Basis. Kingsley Publishing Services. 468 pp.

McKenna, G. 2008. Landscape design for oil sands tailings. First International Oil Sands Tailings Conference, December 7-10, 2008, Edmonton, AB, Canada, pp. 51.

McRoberts, E. 2008 Considerations in oilsands tailings containment. First International Oil Sands Tailings Conference. Edmonton, Alberta. December 2008. 13-23.

Mesri, G. 1989. A reevaluation of $S_{u(mob)} = 0.22\sigma'_p$ using laboratory shear tests. Canadian Geotechnical Journal **26**: 162-164.

Mikula, R. and Omotoso, O. 2006. Role of clays in controlling tailings behaviour in oil sands processing. Clay Science, **12**(2): 177-182.

Newson, T.A. and Fahey, M. 2003. Measurement of evaporation from saline tailings storages. Engineering Geology, 70(2003): 217-233. doi:10.1016/S0013-7952(03)00091-7

Oliveira-Filho, W. and Van Zyl, Dirk (2006) Modeling Discharge of Interstitial Water from Tailings Following Deposition – Part 2: Application. Brazilian Journal on Soil and Rocks, Vol. 29, No. 2, August.

Proskin, S.A. 1998. A geotechnical investigation of freeze–thaw dewatering of oil sands fine tailings. Ph.D. Thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, AB.

Qiu, Y., 2000. Optimum deposition for sub-aerial tailings disposal. PhD thesis, University of Alberta.

Qiu, Y. and Sego, D. C. 2006. Optimum deposition for sub-aerial tailings disposal: concepts and theories. International Journal of Mining, Reclamation and Environment, **20**:4, 272 — 285

Rowe, R.K. 2005. Long-term performance of contaminant barrier systems. Geotechnique, **55**(9): 631-678, November 2005.

Rykaart, E.M., 2002. A methodology to describe spatial surface flux boundary conditions for solving tailings impoundment closure water balance problems. PhD thesis, Department of Civil and Environmental Engineering, University of Saskatchewan, Saskatoon, Canada.

Scaffo-Migliaro, A. 2007. Modeling and simulation of an at-face slurry process for oil sands. MSc thesis, Department of Mechanical Engineering, Univeristy of Alberta, Edmonton, Alberta

Scott, M.D. and Lo, R.C. 1992. Optimal tailings management at highland valley copper, CIM Bulletin, **85**: 85-88.

Sheeran, D. 1993. An improved understanding of fine tailings structure and behavior. Fine Tailings Symposium, Environment Canada, Edmonton, Alberta, 4-7 April 1993, 11 pp.

Schiffman, R.L., Vick, S.G. and Gibson, R.E. 1988. Beahviour and properties of hydraulic fills. Hydraulic Fill Structures, Eds: DJ Van Zyl and SG Vick. American Society of Civil Engineers, Colorado State University, pp. 166-202.

Silva, M. J.. 1999. Plant dewatering and strengthening of mine waste tailings. PhD thesis. University of Alberta. Department of Civil and Environmental Engineering. 271 p.

Simms. P. and Grabinsky, M. 2004. A simple method for estimating rates of drying and desaturation of paste tailings during surface deposition. Tailings and Mine Waste 2004, 11-13 October 2004, pp 287-292.

Sobkowicz, J.C. 2012. The oil sands tailings technology roadmap and action plan: Introduction and key recommendations. Third International Oil Sands Tailings Conference. Edmonton, Alberta. December 2-5, 2012. Pp. 13-23

Sobkowicz, J.C. and Morgenstern, N.R. 2009. A geotechnical perspective on oil sands tailings. 13th International Conference on Tailings and Mine Waste, Banff, Alberta, 1-4 November 2009. University of Alberta, pp. xvii-xli.

Suthaker, N.N and Scott, J.D. 1997. Thixotropic strength measurement of oil sand fine tailings. Canadian Geotechnical Journal, **34**(6): 974-984.

Syncrude Canada Ltd. (Syncrude). 2010. 2010 Annual tailings plan submission Syncrude Aurora North (Leases 10, 12 and 34). 30 September 2010.

Syncrude Canada Ltd. (Syncrude). 2012. 2012 Annual tailings plan submission Syncrude Aurora North (Leases 10, 12 and 34). 30 September 2012.

Vick, S.G. 1990. Planning, design, and analysis of tailings dams. Wiley, New York.

Wallace, D., Tipman, R., Komishke, B., Wallwork, V., and Perkins, E. 2004. Fines/water interactions and consequences of the presence of degraded illite on oil sands extractability. The Canadian Journal of Chemical Engineering, **82**: 667-677.

Wang. T., Feng, X., Xu., Z. and Masliyah, J. 2010. Polymer aids for settling and filtration of oil sand tailings. The Canadian Journal of Chemical Engineering, **88(**6): 403-410.

Wels, C. and Robertson, M. 2003. Conceptual model for estimating water recovery in tailings impoundments. Tailings and Mine Waste ’03. Pp. 87-94.

Xu, Y., Dabros, T., and Kan, J. Filterability of oil sands tailings. Process Safety and Environment Protection, **86**: 268-276.

4 TAILINGS MANAGEMENT SIMULATION MODEL IMPLEMENTATION AND VALIDATION

4.1 INTRODUCTION

4.1.1 Tailings Management

Inherent to the extraction of mineral commodities and hydrocarbon resources through mining, is the production of tailings. Tailings, a by-product of the extraction process, are typically low density, high water content slurries. With no treatment or further processing, tailings require fluid containment upon deposition and generally have poor dewatering and strength gain behavior (Vick 1990). Therefore, raw extraction tailings have proven to be troublesome for mining operations to meet closure and reclamation goals. Storage of slurried tailings also poses a significant environmental risk if the containment structure were to fail. In lieu of the recent Mount Polley tailings storage facility breach, the expert engineering review panel reported that mining operations reduce their risk by moving toward adoption of best available technologies for dewatering tailings (i.e. create highly dewatered deposits and eliminate surface water from the impoundment) (Government of British Columbia 2015).

In the oil sands industry, a low density, high fines content tailings called mature fine tailings (MFT) is formed after deposition of whole tailings. The MFT requires long term containment and further dewatering is expected to take decades (Sobkowicz and Morgenstern 2009). In order to meet reclamation and regulatory objectives suitable for reclamation, tailings must undergo significant dewatering. During dewatering, tailings will undergo changes in strength, saturated hydraulic conductivity, and compressibility of several orders of magnitude (Boswell and Sobkowicz 2010). With sufficient dewatering, tailings can develop sufficient long term stiffness and strength (50 to 100 kPa) to support reclamation activities (Boswell and Sobkowicz 2010).

There are three stages of dewatering tailings may go through before they meet their end reclamation targets (Boswell and Sobkowicz 2010). The first stage involves anthropogenic or natural classification of the tailings stream. Mechanical separators such as hydrocyclones may be used to separate a tailings slurry stream into a low density, fine grained overflow and coarse, dense underflow. Tailings may also undergo natural segregation/dewatering when they are sub-aerially discharged. In this case, a coarse beach deposit and a low density, fine grained slurry run off are formed. The beach run off collects within the impoundment and may settle with time. The second stage of dewatering includes the various mechanical, chemical and electrical methods described in Chapter 2. These technologies will typically dewater the tailings streams to near, but still wet of their liquid limit. Upon deposition, the tailings deposits will typically have strengths of a few hundred pascals (Boswell and Sobkowicz 2010). The final stage of dewatering following deposition (Stage 3) includes time dependent and environmental dewatering processes. Stage 3 dewatering includes sedimentation/consolidation processes, and environmental dewatering processes such as freeze/thaw dewatering, desiccation and evapotranspiration. With an appropriate deposition and management strategy, Stage 3 dewatering can be maximized in order to dewater the tailings and achieve the strength required to meet reclamation targets.

Management of tailings also includes the construction and operation of tailings storage facilities (i.e impoundments). Impoundment may be constructed from the tailings (hydrocyclone underflow, or sub-aerial beach deposits), from other mine waste or natural soils. The construction of the impoundments must be coordinated with the deposition and storage requirements of the tailings and associated process water to ensure sufficient storage capacity and freeboard is available. The required capacity of the impoundment is a function of the tailings dewatering processes (described above), the interaction with the environment (i.e. seepage, precipitation, evaporation), and process water demands from the extraction process. Mine operators manage their tailings through the implementation of a tailings management system (TMS) that incorporates all aspects of the tailings dewatering and their associated storage facilities.

4.1.2 Tailings Management Simulation

The underlying processes in a TMS presented above (and discussed in Chapter 2) are typically modeled separately using complex, analytical tools. There are few models or approaches available to the public that incorporate the tailings management process as a whole system over the life of the mine. The following models attempt to examine various aspects of tailings management through deterministic, stochastic and optimization models.

In 2009, the Alberta Energy Research Institute published a study and developed a deterministic, material balance and economic model using excel spreadsheets to screen and evaluate oil sand tailings technologies and practices (Devenny 2010). The model determined the fresh water make-up needs, time to reclamation, and included high level cost estimates. Only one technology could be simulated at a time (no concurrent technologies). The model was driven by static ore body components, tailings properties and extraction parameters. Operational issues such as interim capacity requirements and process water management were not considered in the model.

A simulation model linking long term mine plans with a CT production planning model was developed by Kalantari in 2011. Deterministic and stochastic simulations were used to establish a mining schedule that takes into account the tailings produced (CT), required storage impoundments, and quantify uncertainties with the CT production process. Both Visual Basic for Applications (VBA) code and Matlab code were implemented to run the simulations. The CT production model assumed a constant density of the feed fine tailings and final product (CT) and the material composition of the cyclone underflow. Stochastic simulations were based on probabilistic distributions for ore rejects (total sand content), target sand to fines ratio and on-spec CT.

A complementary model to Kalantari's (2011) was developed by Ben-Awuah and Askari-Nasab (2011 and 2012). They integrated waste material allocation and dyke construction requirements with long term mine plans to optimize the net present value (NPV). Dyke requirements were determined from the final tailings volumes and a rigid dyke construction design using Devenny's (2010) "base case" oil sand ore body. No tailings management technology was specified for the model (i.e. CT, thickened tailings, end pit lakes, etc.). Additionally, process water storage requirements were not explicitly calculated or included. The optimization model was implemented in TOMLAB/CPLEX environment.

Another NPV optimization model was developed by Badiozamani and Askari-Nasab (2012a and 2012b) based on estimated reclamation material requirements and associated handling costs. The model was constrained by the required tailings capacity and reclamation material needs (i.e coarse tailings and overburden for capping). Tailings volumes were based on Suncor's model for cyclone underflow (coarse tailings) and overflow (fine tailings). The optimization model was implemented in MATLAB/CPLEX environment.

In 2012, a large project was undertaken to screen and evaluate tailings technologies and practices to assist in the creation and implementation of technology solutions for tailings reclamation in the oil sands industry. The project was called the Oil Sands Tailings Technology Development Roadmap (Tailings Roadmap) for the AI-EES (Sobkowicz 2012). Over 550 potential technologies were initially identified which were subsequently consolidated to 101 unique types and variations having potential application in the oil sands industry. To evaluate the 101 tailings technologies, a set of tailings reclamation objectives were defined. Based on qualitative input from relevant professionals, the tailings technologies were then assessed against the objectives. Groups of 8 to 10 professionals ranked each technology for its ability to meet the specified reclamation objective. The data was compiled into a spreadsheet to organize the rankings. A total of nine technology roadmaps were developed based on the best ranked technologies that could improve tailings management practices. The technology suites and ranking were solely based on qualitative opinions, therefore, they should be assessed for applicability to individual mine sites to ensure site specific needs are included. Additionally, there is an ongoing need to provide updates and assessment of technologies as they develop.

The above publically available tailings and mine planning models either do not take into account the dynamic nature of the tailings management process are attempt to through probabilistic estimates of parameters. Devenney (2009), Ben-Awuah and Askari-Nasab (2011 and 2012), and Badiozamani and Askari-Nasab (2012a and 2012b) based their tailings volumes on static forecast models and parameters. Kalantari (2011) incorporated probabilistic distributions for some tailings forecast parameters, but kept other parameters rigid. The mine plan optimization models looked at theoretical scenarios where the NPV of a mine plan could be maximized based on implementation of CT technology Kalantari (2011). However, the use of the complementary models is required to assess containment and reclamation requirements (Awuah and Askari-Nasab 2011 and 2012; Badiozamani and Askari-Nasab 2012a and 2012b). The above models also assumed final storage volume requirements and did not assess required operational stage curves/storage requirements. Substitution of alternative tailings technologies or utilization of multiple technologies concurrently was not possible either. These models were focused solely on optimizing the mine plan rather than assessing the merits of particular tailings management technology. The recent initiative by government and industry (Tailings Roadmap) provided a foundation for assessing alternate tailings technologies. However, the process was based solely on qualitative assessments. Future evaluation of technologies using the Tailings Roadmap may require significant effort.

There currently is no single simulation/assessment model available for evaluating tailings technologies and management options quickly and efficiently. A simulation model is needed that incorporates the mine plan, various stages of dewatering including classification, pre- and post- deposition dewatering, and an impoundment material balance including tailings, process water, construction material and capping materials. This research aims to develop a simulation modelling tool, called TMSim that can be used to evaluate tailings technologies and incorporate the above processes. A dynamic systems approach was adopted for the simulation. An object orientated, systems dynamic modeling software called GoldSim was used as the "simulation engine". The previous chapters detailed the tailings management components and approach taken to develop the simulation model. This chapter will focus on the TMSim model structure, implementation and validation.

4.2 TMSIM DEVELOPMENT

4.2.1 Modeling Software

Mining operations are constantly evolving with time due to inherent changes within ore bodies and subsequent extraction processes as well as economic conditions. Dynamic system analysis and modeling or dynamic simulation modeling is therefore appropriate to capture and represent the evolving system through time. The modeling system becomes dynamic when feedback processes are captured through time (Halog and Chan, 2008). In addition to dynamic uncertainties, a TMS is highly multidisciplinary. In a typical TMS, sub models for each discipline or component are not usually linked. Therefore, a user may lose sight of the "big picture" (the ultimate problem that the modeling is trying to address). The interdependencies of the different elements of the subsystems are often ignored or poorly represented.

There are a suite of general-purpose, systems based software tools (STELLA, GoldSim, Simulink, etc.) that are becoming increasingly popular (Rizzo et al. 2006). These modeling packages provide easy to use, graphical interfaces that can be understood and used by novice modelers. Whereas, traditional model development using high level programming languages (Matlab) may take years of training to fully implement (Rizzo et al. 2006). A system-based software is appropriate for simulating a TMS because it allows the greatest flexibility and application for users of all skill levels. Rizzo et al. (2006) compared four system simulation packages and Matlab. They translated a canopy surface wetness model into GoldSim, Madonna, Simulink, and STELLA modelling platforms. GoldSim was slightly less user-friendly than Madonna and STELLA, but provided greater computational capabilities and equation/function editing. GoldSim was also more sensitive to changes in input parameter variability.

GoldSim was ultimately chosen as the simulation package for TMSim because of its ability to simulate dynamic systems such as a tailings system, balance of computational power with ease of use, and other features such as ability to link with external software. GoldSim allows flexible inputs, outputs, time stepping, and coupling of processes (Wickham et al. 2004). The Goldsim platform is like a "visual spreadsheet". 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. Probability can even be incorporated within the model to represent uncertainty in processes, parameters and events. To integrate all of the TMS components, Goldsim is highly suited for top down, total system modeling approach, keeping the "big picture" in focus. This facilitates documentation of the complex model and allows one to see how all the components of the model fit together, ensuring they are reasonable and logical. Goldsim also has the ability to dynamically link with external programs such as excel to enhance the simulation capability (Kossik and Miller, 2004). The modeling approach adopted, using GoldSim, will assist the planner to understand the TMS and its boundaries, identify key variables and clarify complex interrelationships.

4.2.2 Model Structure

The TMSim will use the object orientated, systems dynamic modeling software called GoldSim as the "simulation engine" and will be linked with external modeling software including spreadsheet models (deposition topography), 3rd party consolidation models (FSConsol) and user defined design data/charts to develop the tailings simulation model (Figure 4.1). A typical GoldSim sub-model is created by constructing an influence diagram (Figure 4.2) using the built in modeling elements and programming equations representing the relationships between the elements.

The TMSim model tracks the stocks (accumulation in a containment facility) 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 dewatering Stage 1, 2 or 3, impoundment and the environment (Figure 4.3). Critical processes (such as consolidation, Stage 2 dewatering, seepage, etc) within each component will dictate mass transfer between the sub-models as discussed in Chapter 3. An example of a GoldSim sub-model used in the TMSim model is included as Figure 4.4. This model represents the influence diagram between the Stage 3 dewatering (sedimentation/consolidation), seepage and water cap which all impact the calculation of required storage volume (ETF1_volume), a performance measure of the TMSim. A summary of all the GoldSim sub-models and detailed variable/function influences are included in Appendix 2. It is obvious that a TMS is multifaceted, interrelated and complex. Therefore, the TMSim will be a simplification of the real mining system and include only the important user defined processes. The accuracy of the model will depend on the level of detail and parameters available to the user.

4.2.3 Implementation of Model

The TMSim model process is illustrated in Figure 4.5 and contains three components. The first component the data input requirements. 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 UDF. The UDFs can be simple or complex numerical models, depending on the level of detail available and objective of the modelling. Implementation of user specific models/data would be completed either in the data input spreadsheet or the Goldsim model directly (Beier et al 2009). A summary of the inputs requirements from each of the sub-models is provided in Figure 4.6. In addition to the variables and UDFs for each of the sub-models (Figure 4.6), the decision logic required to allow for the switching between dewatering technologies and deposition points must also be provided by the user. These decisions and logic points are identified on Figure 4.7.

Various methods, ranging from empirical estimates to direct measurements and theoretical formulations are available to the modeler for use as inputs. Their use depends on the degree of accuracy required and cost for implementation. A hierarchical level of detail is proposed for the simulation modeling which fits well with the “top down systems” approach of the TMSim model structure. For example, the lowest level of detail would include “database mining” of the literature or professional judgment for a quick estimate of a particular parameter. The next level would base estimations of a particular parameter or property on some direct measurement of simple properties. This approach is commonly used for most preliminary tailings and mine waste studies (Fredlund et al., 2003). Finally, direct measurement of properties and parameters may be conducted either in a laboratory or in situ and used in conjunction with theoretical or empirical formulations. The challenge to the modeler is then to apply the appropriate level of detail that will result in a suitable engineering simulation satisfying the objective of the model process. Attempt should be made to use the highest possible level of detail that is necessary. Therefore, nearly any parameter or process can be determined whether it is based on crude empirical correlations or direct in situ measurements (Fredlund et al. 2003). This approach is well suited for application in GoldSim because of its ability to construct models from process-based, empirical or even qualitative formulations based on a tentative relationship between two parameters.

The second component of the TMSim is the simulation engine. The GoldSim software coupled with Excel VBA code and FSConsol software will execute the sub-models for the extraction, Stage 1, 2, and 3 dewatering, impoundment and environment (Figure 4.3) as presented in Chapter 5. Using the input data from Part 1, the flow of information (decisions) and material (solids, water, chemistry) between each component will be tracked accordingly.

Model output (Part 3) include the performance measures described in Chapter 3. For each deposition area (i.e. external tailings facility), the required volume of storage for both tailings and process, the available storage volume, the strength of the tailings deposit and the quality of the process will be calculated at each time step. Additionally, surface profiles of the tailings deposit will be calculated and can be further examined/incorporated into geographic information system (GIS) software. By utilizing the built in sensitivity analyses in GoldSim, the robustness of a particular technology under evaluation to uncertainties can also be assessed.

4.3 VALIDATION OF SUB SYSTEMS

The introduction of numerical instability, truncation and round-off errors are potential issues for any numerical modeling exercise. Therefore, the validity of output from a numerical model should be compared with data known to be true. The data may arise from an analytical solution or from experimental data. If the model output is within a specified tolerance to the "exact solution", the model is deemed valid. Given the nature of the TMSim model (i.e. mass balance of a large system), an analytical model is not available, however, several components of the model can be compared with analytical or experimental data. Therefore, the TMSim model components will tested individually for validity where available.

Due to flexibility of the TMSim model, individual UDFs must be validated to ensure the numerical approach is correct and verified to ensure the UDFs are coded properly, by the user at time of implementation. Given the numerous variations possible for the UDFs, only the processes currently coded into the TMSim (Chapter 3) will undergo a validation step. Currently, the sedimentation, consolidation, and tailings deposition model are included in the TMSim and will be validated individually. To ensure validity of the entire model as a whole, a metal mine tailings plan will be simulated and compared to the findings of a feasibility report.

4.3.1 Sedimentation Sub-Model

The sedimentation sub model incorporates Masala's (1998) approach to solve Kynch's (1952) theory. The model is a simplification to Kynch's theory and provides a computing advantage due to its simplicity and independence of material models. To utilize the sedimentation model, the user must first specify a representative function for change in suspension/water interface with time. This can easily be extracted from sedimentation columns tests. To test the sedimentation model simplification, data from Toorman and Gudehus (1998) (also used by Masala (1998) as a validation data set) is compared with the TMSim model output. Spherical glass beads, (67 μm diameter), initial solids content of 30.2 %, specific gravity of 2.45, and initial height of 31.2 cm were used in Toorman and Gudehus's (1998) test. The interface settlement velocity was calculated as 0.16 m/s from the data set. The sedimentation profile diagram for Toorman and Gudehus's (1998) data is plotted on Figure 4.8 as points. The TMSim model output are presented at lines on Figure 4.8 and show very good agreement with the Toorman and Gudehus (1998) data. Therefore, the modeling approach is deemed valid for batch settling or quiescent conditions (no active filling).

However, since tailings management plans will also include continuous filling, the sub-model must be allow for continual addition of slurry to the deposit. The sedimentation sub-model treats tailings deposited in a given time step as an individual layer. It then tracks the material properties and water released for each layer as time progresses. The total water released or sediment created at the base is the sum from each layer for the given time step. To validate this modeling approach, a known volume of tailings will be modelled in quiescent conditions and under filling conditions. According to Carrier et al. (1983), the final sediment height of non-consolidating slurries is dependent on the dry weight of the solids placed in the disposal area. Therefore, if the disposal area is filled rapidly and left to settle (i.e. quiescent conditions modelled above) or filled slowly over a long period of time, the final sediment height should be the same.

To validate the filling conditions, sedimentation data for fine tailings from CNRL's tailings plan (CNRL 2010) will be used. They assume fine tailings are deposited at 12 % solids (Sg = 2.57) and will reach a solids content of 30 % after 30 months. Therefore, for each tonne of solids deposited, 0.167 m^3/month of water will be released. Using this sedimentation data, two scenarios were simulated, one where 10 tonnes of fine tailings are deposited instantaneously and allowed to settle with time and a second where 10 tonnes of fine tailings are deposited at a rate of 1 tonne/month and allowed to settle. The sedimentation diagram for both scenarios are included as Figure 4.9. Respecting Carrier et al.'s (1983) assumption, the final sediment height in both cases is the same at 27.2 m^3. Therefore, the sedimentation sub-model for filling conditions is performing as expected.

4.3.2 Consolidation Sub-Model

The TMSim model utilizes a "black box" concept to simulate the consolidation process, employing a commercially available software called FSConsol utilizing Gibson et al. (1967) finite strain theory. The FSConsol software is dynamically linked with the GoldSim software via Excel VBA code. It is assumed the FSConsol software is employing a valid numerical representation of the finite strain theory, therefore, validation of the finite strain component is not required. The linkage with FSConsol currently does not include simulation of the time dependent, creep process. Jeeravipoolvarn (2010) assessed the FSConsol software and determined that while it did not consider creep, it was reasonably capable of predicting interface settlement of a sand dominated tailings deposit (Syncrdue CT). The finite strain consolidation theory employed by FSConsol is capable of capturing coarse grained tailings consolidation behavior and the behavior of high solids content fine grained tailings. It did, however, have difficulties predicting the sedimentation consolidation behavior of low solids content fine grained oil sand tailings (i.e. MFT). Therefore, low solids content, fine grained materials will be modelled approximately using the sedimentation sub-model and only coarse-grained and higher solids content materials will employ the FSConsol sub-model. Given the trend in the mining industry to move from storage of large volumes of low solids content tailings towards depositing high solids content tailings, this trade-off is deemed appropriate.

A validation step however, is required to ensure the dynamic linkage and transfer of data from GoldSim to FSConsol is acceptable. 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 4.10). Tailings material properties and typical loading rates from a metal mine scenario as well as an oil sand thickened tailings stream (Masala and Matthews 2010) were utilized to conduct the validation run (Table 4.1. Consolidation sub-model parameters.). 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.

4.3.3 Deposition Algorithm

4.3.3.1 Validation

To ensure the deposition algorithm and model code (GoldSim and VBA macro) function as expected, a validation step was conducted. A simple scenario consisting of depositing a known volume of tailings at a constant slope in ten sequential steps was conducted. The model parameters are summarized in Table 4.2. Deposition sub-model validation data. The actual volume and associated heights of the deposited cone can be confirmed using the analytical formula for volume of a cone. A tolerance level for the solver algorithm was set at 0.5%.

[6.1] $$Vol = \frac{1}{3} * \pi * r^2 * h$$

Where r is the radius of the cone and h is the height of the cone. Three scenarios were simulated, representing three potential spatial discretizations (0.5, 1, and 2 m) for both the x and y directions. The calculated heights and volumes for each case are also summarized in Table 4.3. Deposition sub-model demonstration scenarios.

The relative percent error of height and volume is presented in Figure 4.11 and Figure 4.12. Case 1 had the lowest error for both height and volume due to the finer discretization. For Case 2 and 3, the relative error was nearly indistinguishable. The maximum error in height for all cases was 0.08% occurring at the final step. The maximum error in volume for all cases was 0.4% from the initial step. These errors are minimal and within the specified tolerance level of 0.5%.

4.3.3.2 Deposition Scenarios

To demonstrate the potential wide range of applications of the deposition sub model, four scenarios are presented. The first scenario demonstrates deposition of a tailings slurry with nearly flat deposition slope. This is representative of pouring low solids content tailings into a deposition cell or external tailings facility. The second scenario represents a central discharge of high solids content tailings into a deposition cell with little to no ponded water. The third scenario consists of depositing a tailings slurry from a multi-spigot line along a beach into a deposition cell containing ponded water. The final scenario repeats scenario 3 but varying the iBAW at each step. A total of twelve steps were simulated. A summary of the tailings properties and deposition cell dimensions are presented in Table 4.3. Deposition sub-model demonstration scenarios.. Figure 4.13 to Figure 4.18 represent cross sections of the tailings surface profiles for each scenario over the twelve steps.

The deposition sub-model was able to produce the expected profiles for each scenario. For scenario 1, the tailings surface reflected the deposition slope of 0 and the surface elevation increased approximately 0.15 m per each step. The final tailing surface was calculated at 1.65 m (Figure 4.13). In scenario 2 (Figure 4.14 and Figure 4.15), the final tailings peak elevation at the central riser was 6.76 m. The tailings surface profiles increased outward with diminishing layer thickness as the cone of tailings grew in volume. In scenario 3 (Figure 4.16), the tailings surface profiles grew with each step and the layer thicknesses decreased with increase in volume. Also evident in Figure 4.16, are the pond water elevations. They are the represented by the “knee” in the tailings profiles. The final pond elevation was calculated at 3.48 m. Finally, in Figure 4.17 and Figure 4.18, the deposition model was capable of calculating tailings profiles with changing tailings properties (iBAW). The slope used for each step can be found adjacent to the corresponding tailings profile. The final water elevation was calculated at 3.30 m, less than in scenario 3. A difference in water volume elevations is expected due to the separate displacements each scenario produced in the water pond. Based on the minimal calculation errors for surface profiles and material volumes and the versatility to simulate several common depositional scenarios, the deposition sub-model is deemed acceptable for use in the tailings management simulation package.

4.3.4 Impoundment Mass Balance

To ensure the TMSim model as a whole will produce valid results, a simple simulation was undertaken using a metal mine tailings management scenario. The data used in the validation was taken from an engineering feasibility study. The complexity and scale of the proposed tailings management plan was deemed suitable for the development and validation stages of the TMSim model structure and components.

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/year over the 29 year life of the mine. Extraction tailings consist of non-plastic sandy silt with 65 % geotechnical fines content (<

75 μm). 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 a data set generated during an engineering feasibility analysis (Figure 4.19 and 4.20). The solid lines represent the TMSim model data and the dashed line represent the feasibility report data sets. As can be seen from the figures, 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 TMSim model were digitized from a report and not the original detailed design data set. 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 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.

4.4 CONCLUSIONS

Management of tailings includes all aspects of tailings dewatering, deposition and the construction and operation of tailings storage facilities. The implementation and performance of dewatering technologies and depositional management practices are crucial in order to meet reclamation and regulatory objectives. Additionally, the construction of storage impoundments must also be coordinated with the deposition and storage requirements of the tailings and associated process water to ensure sufficient storage capacity and freeboard is available throughout the mine life. Typically complex, numerical models are utilized to design and evaluate the individual components of the tailings management plans. Few models are available that assess the tailings management process as a system and they do not take into account the dynamic nature of the tailings management process.

A simulation model called TMSim was developed that incorporates the mine plan, various stages of dewatering including classification, pre- and post-deposition dewatering, and an impoundment material balance included tailings, process water, construction material and capping materials. It will be used to evaluate tailings dewatering technologies and management strategies to ensure reclamation and closure goals are met while balancing ongoing storage demand needs.

An object orientated, systems dynamic modeling software called GoldSim was used as the "simulation engine" for TMSim coupled with Excel VBA, spreadsheet models and FSConsol. The implementation will allow great flexibility because user inputs can range from empirical estimates to direct measurements and theoretical formulations. Therefore, nearly any parameter or process can be determined whether it is based on crude empirical correlations or direct in situ measurements. Their use depends on the degree of accuracy required and cost for implementation.

Individual TMSim model components for sedimentation, consolidation, and deposition were validated using experimental, analytical and numerical data sets. Using a metal mine tailings plan, a comparison between TMSim predictions and a design data set was also presented to demonstrate the application of the model as a whole. The mass balance and performance measures predicted by the TMSim agreed well with the design data set. The TMSim model developed has demonstrated its validity for application to simulate tailings management systems and technologies.

4.5 TABLES

Table 4.1. Consolidation sub-model parameters.

Model Parameter	Metal Mine	Oil Sands Mine
Loading rate (kg/d/m^2)	24	3.289
Solids Content (%)	72.8	45
Sg	4.09	2.65
A	1.2649	1.8059
B	-0.111	-0.233
C	1e-06	1e-9
D	7.4377	3.0713

Table 4.2. Deposition sub-model validation data.

	Analytical Cone		Case 1 (0.5 m)		Case 2 (1.0 m)		Case 3 (2.0 m)	
Step	Volume (m^3)	Height (m)	Volume (m^3)	Height (m)	Volume (m^3)	Height (m)	Volume (m^3)	Height (m)
1	3000	1.371	3010.46	1.372	3011.47	1.372	3011.86	1.372
2	6000	1.728	6001.18	1.728	6002.19	1.728	6002.56	1.728
3	9000	1.978	9010.59	1.978	9011.61	1.978	9012.02	1.978
4	12000	2.177	12001.05	2.177	12002.31	2.177	12002.70	2.177
5	15000	2.345	15014.87	2.345	15016.13	2.345	15016.52	2.345
6	18000	2.492	18026.55	2.493	18027.82	2.493	18028.22	2.493
7	21000	2.623	21037.24	2.624	21038.51	2.624	21038.92	2.625
8	24000	2.742	24051.66	2.744	24052.91	2.744	24053.33	2.744
9	27000	2.852	27062.56	2.854	27063.81	2.854	27064.21	2.854
10	30000	2.954	30075.72	2.957	30076.97	2.957	30077.35	2.957

Table 4.3. Deposition sub-model demonstration scenarios.

Scenario	Description	Tailings Vol./step (m^3)	Cumm. Water Vol. (m^3)	iBAW / iBBW	Cell Size (m)	Berm Crest Elevation (m)	Berm Slope
1	Low density tailings	5,000	0	0 / N.A.	200 x 200	2	4:1
2	High density, central discharge	30,000	0	0.03 / N.A.	500 x 500	2	4:1
3	Medium density, multi-spigot discharge	30,000	100,000	0.015 / 0.04	500 x 200	10	4:1
4	Medium density, multi-spigot discharge	30,000	100,000	0.01-0.02 / 0.04	500 x 200	10	4:1

4.6 FIGURES

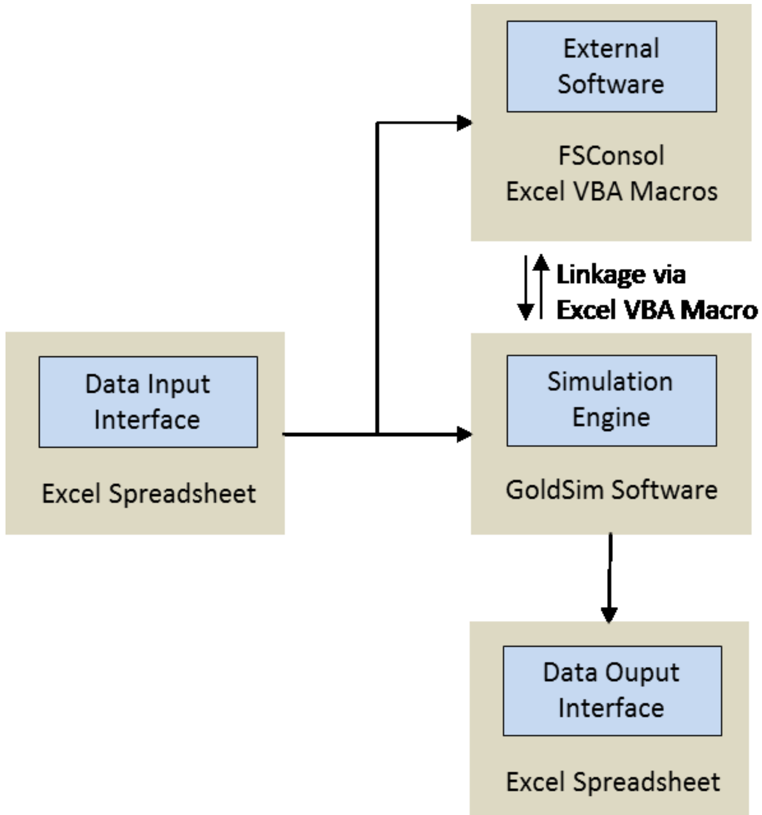


Figure 4.1. TMSim modeling environment.

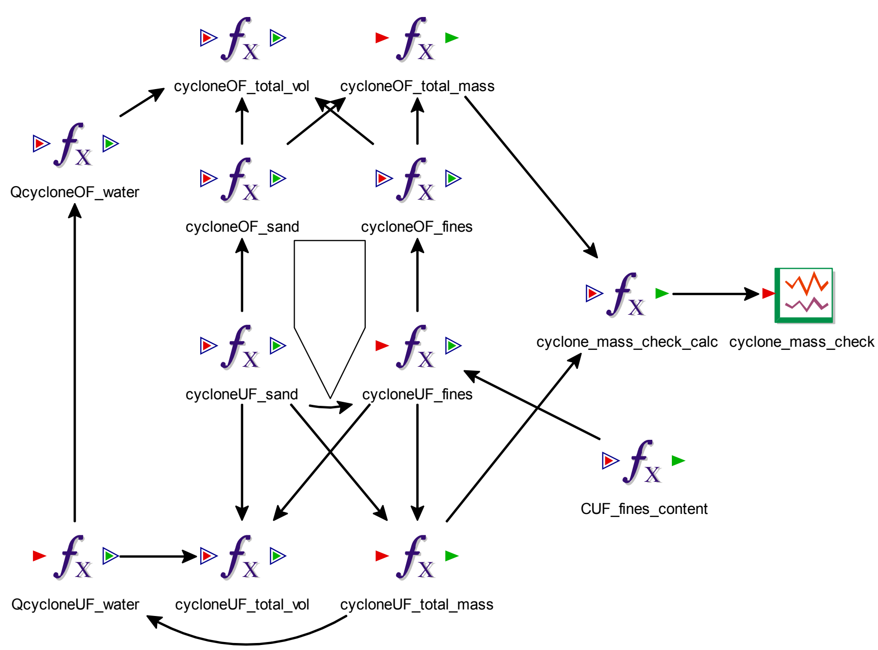


Figure 4.2. Influence diagram of the Stage 1 Goldsim cyclone model.

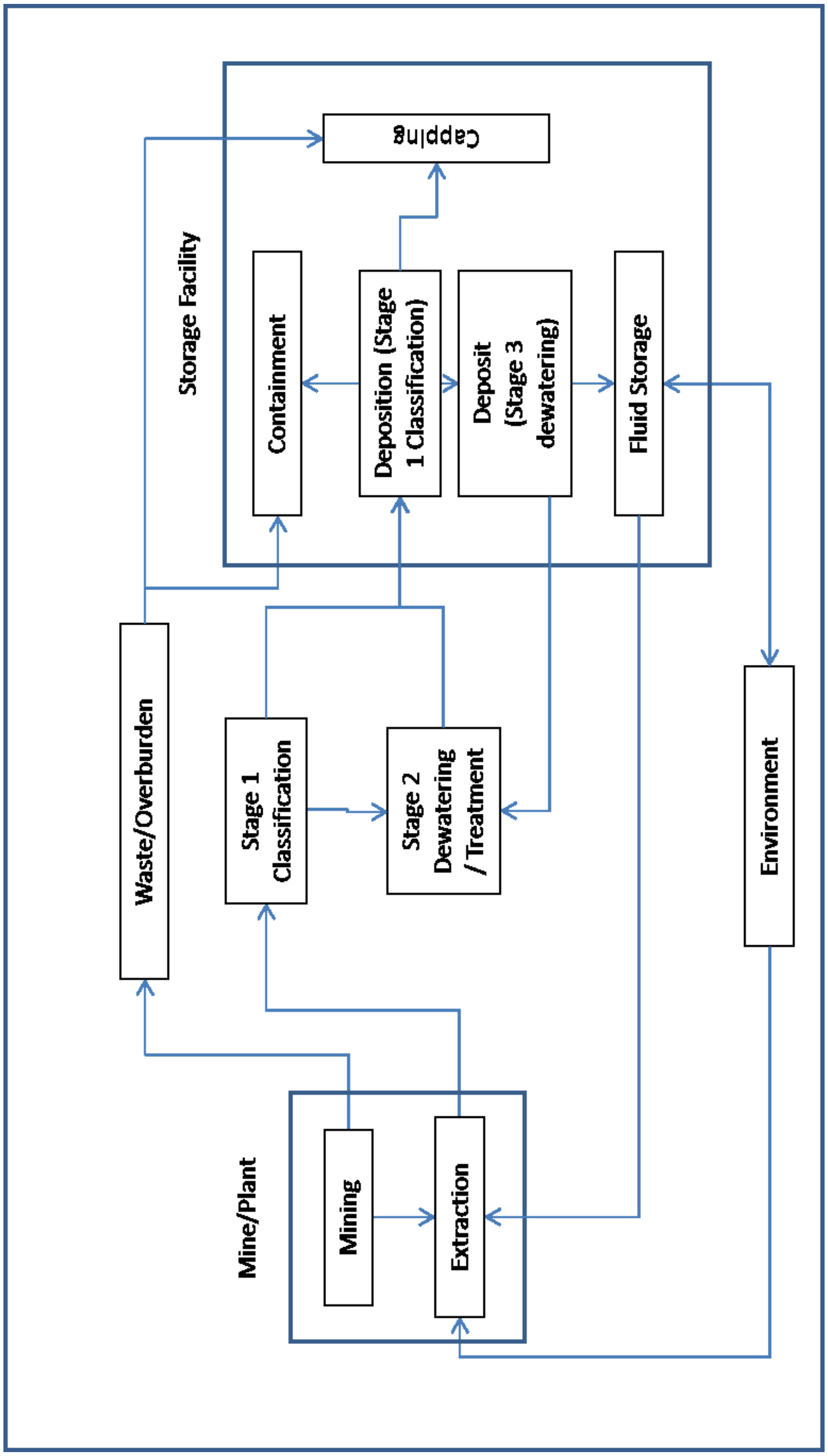


Figure 4.3 Tailings management systems conceptual model.

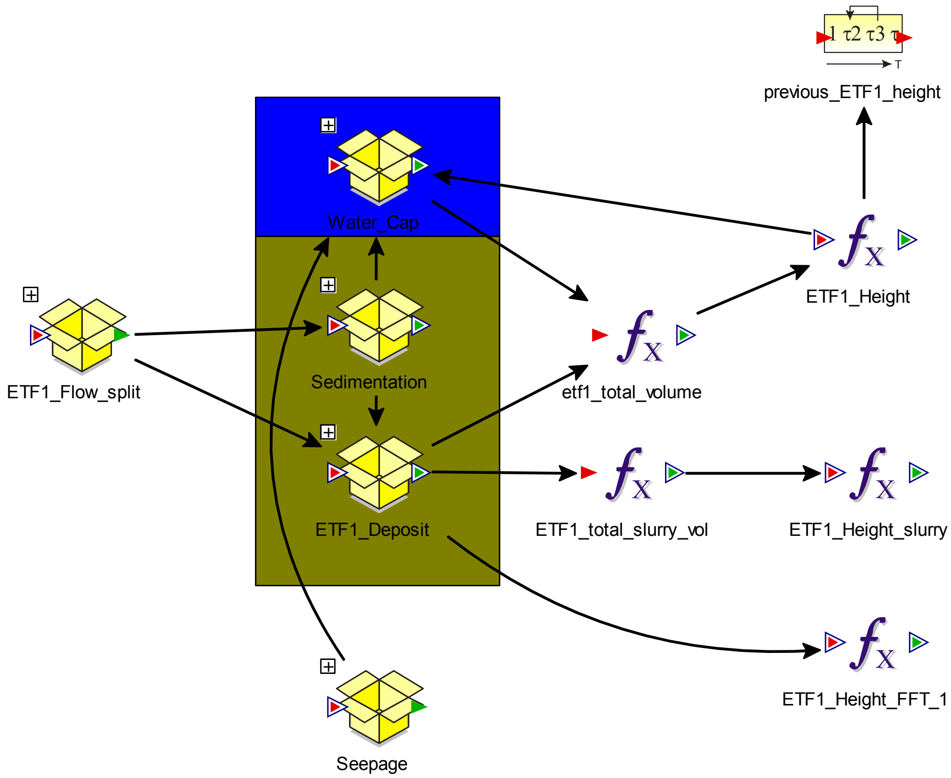


Figure 4.4. Influence diagram of the impoundment GoldSim code.

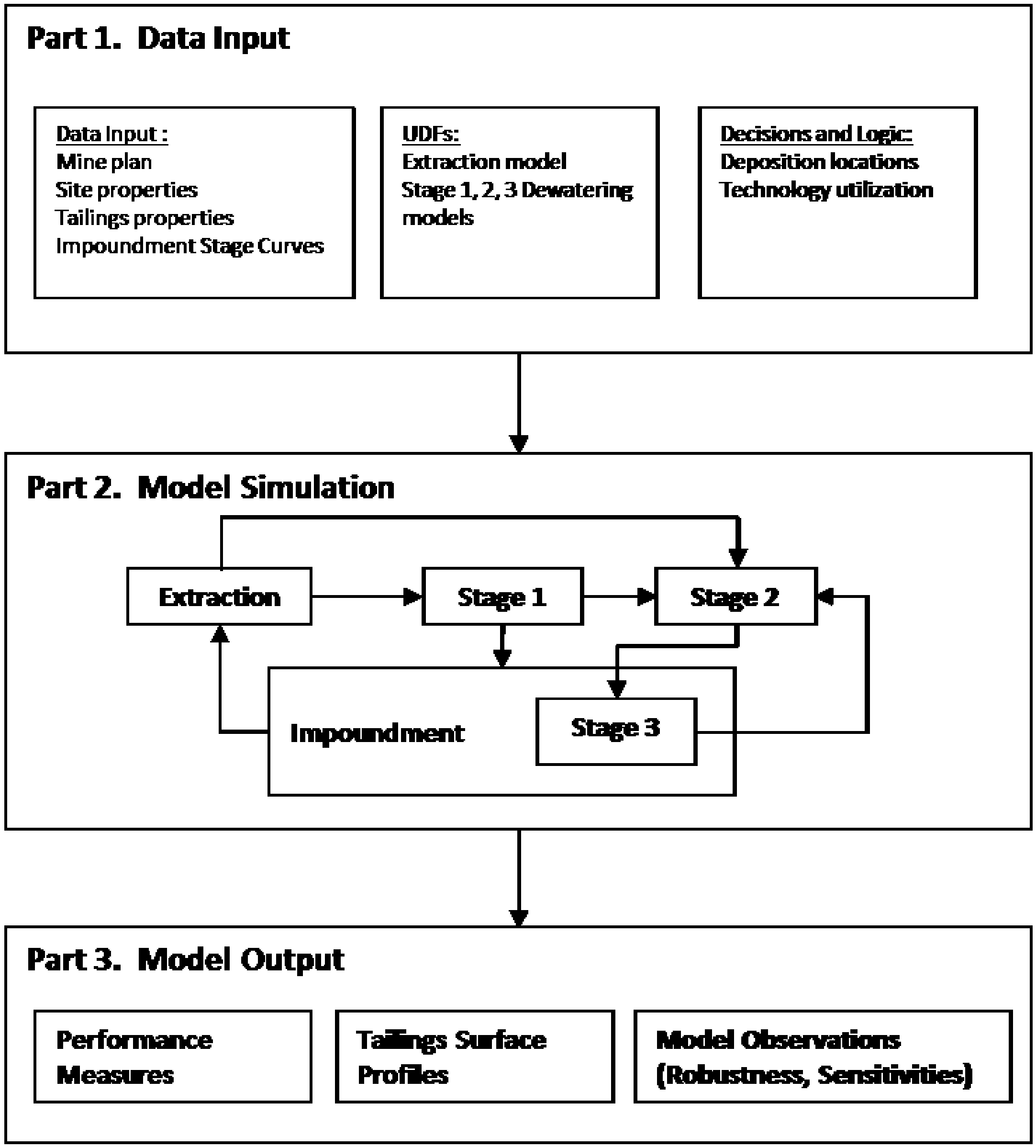


Figure 4.5. Flow chart of TMSim modeling process.

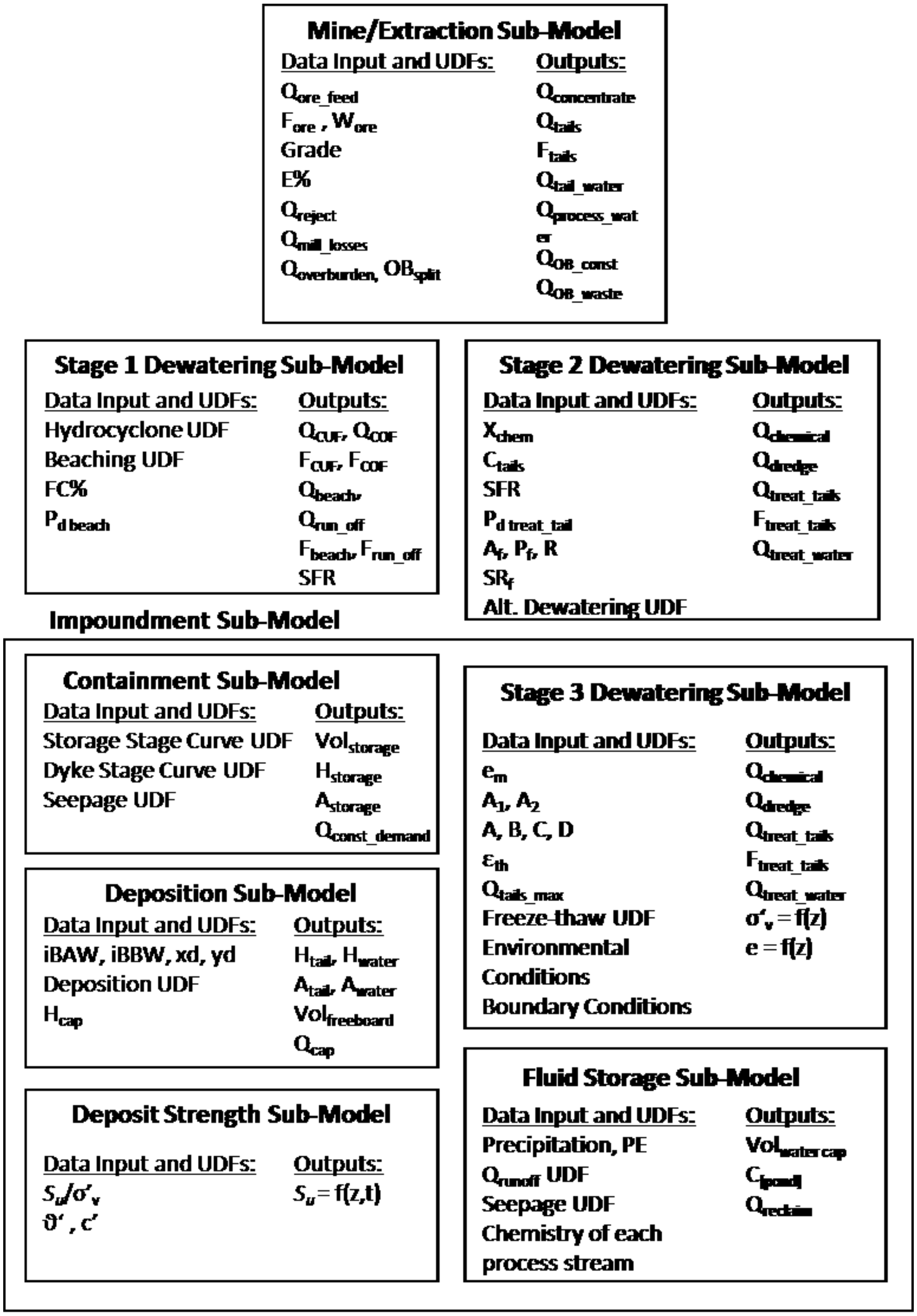


Figure 4.6. Data inputs, UDFs and outputs for each TMSim sub-model.

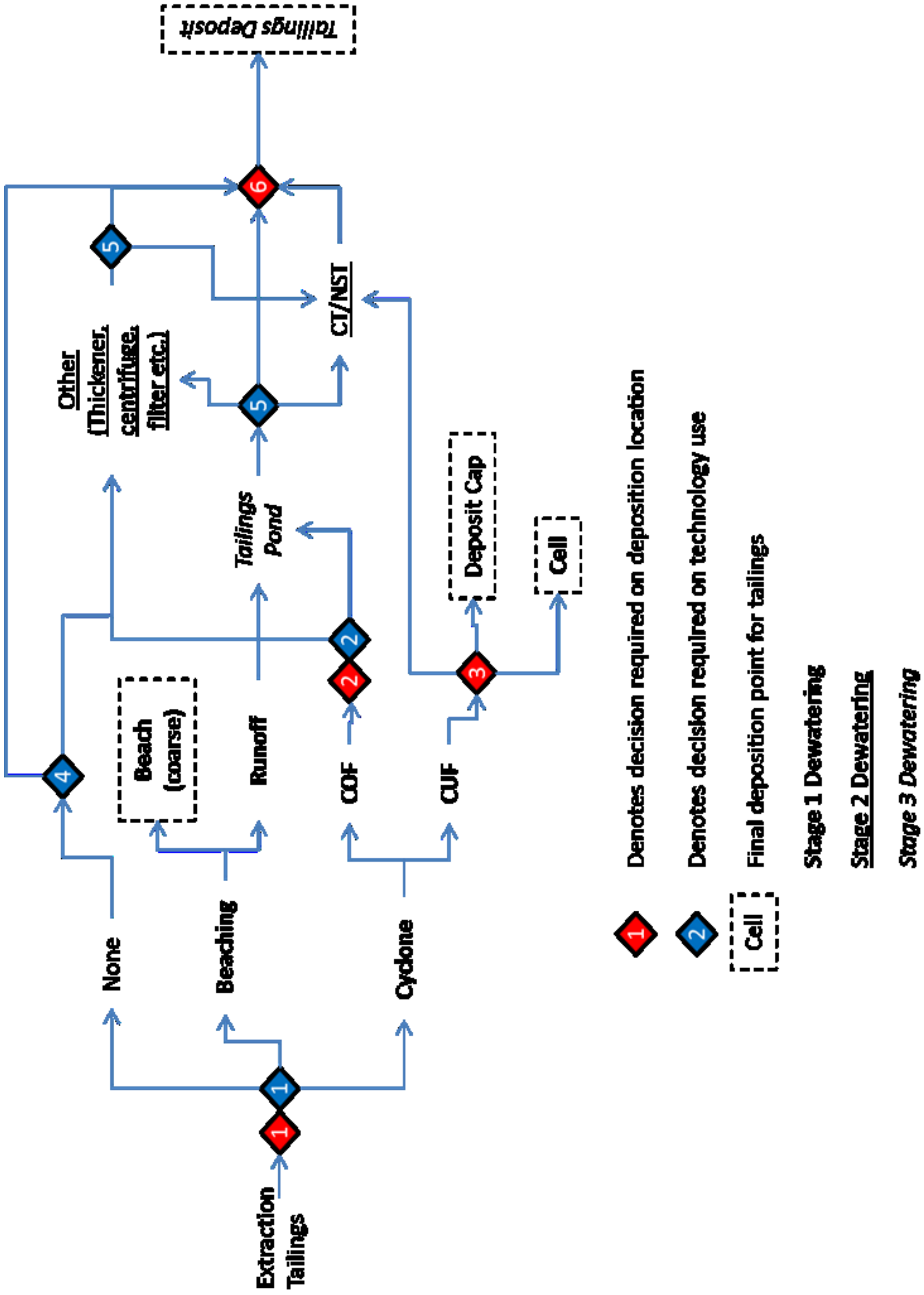


Figure 4.7. TMSim user decision and logic locations.

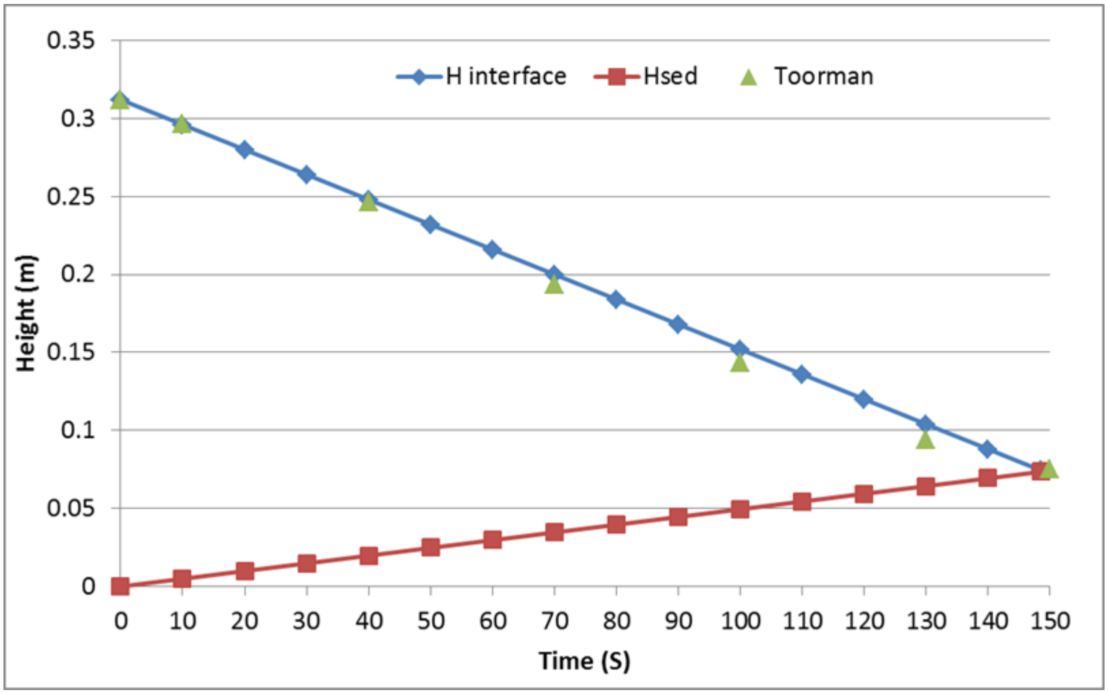


Figure 4.8. Sedimentation algorithm validation with Toorman and Gudehus (1998) data.

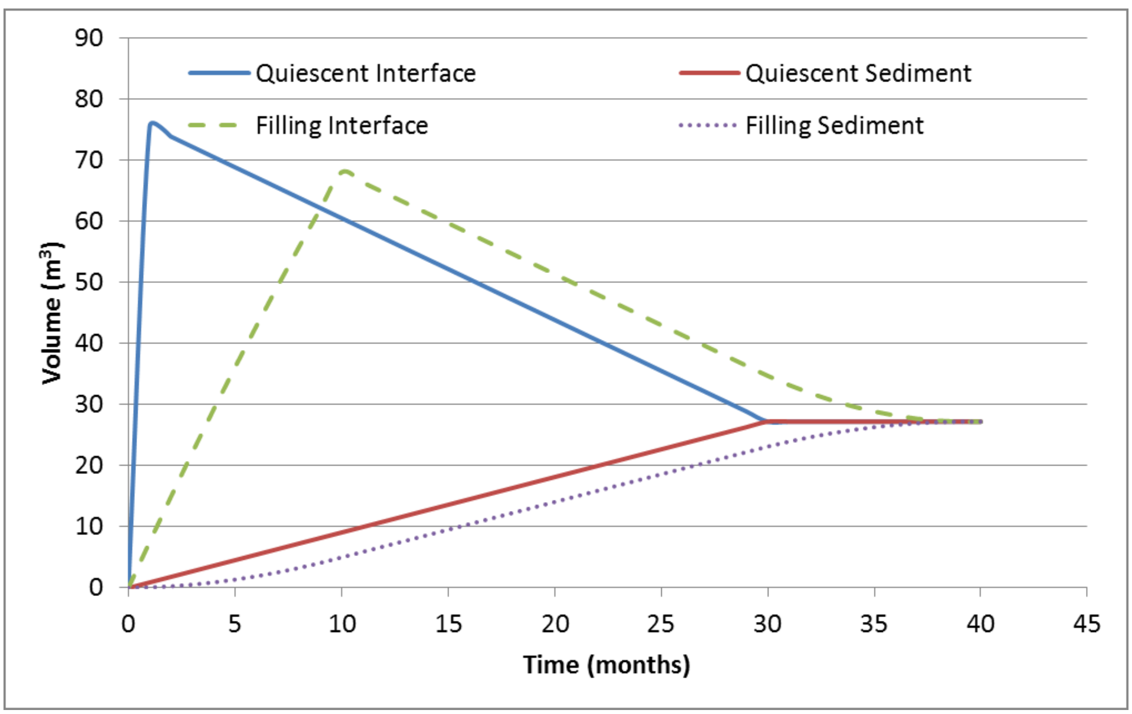


Figure 4.9. Sedimentation algorithm comparison of quiescent and filling conditions for oil sands tailings.

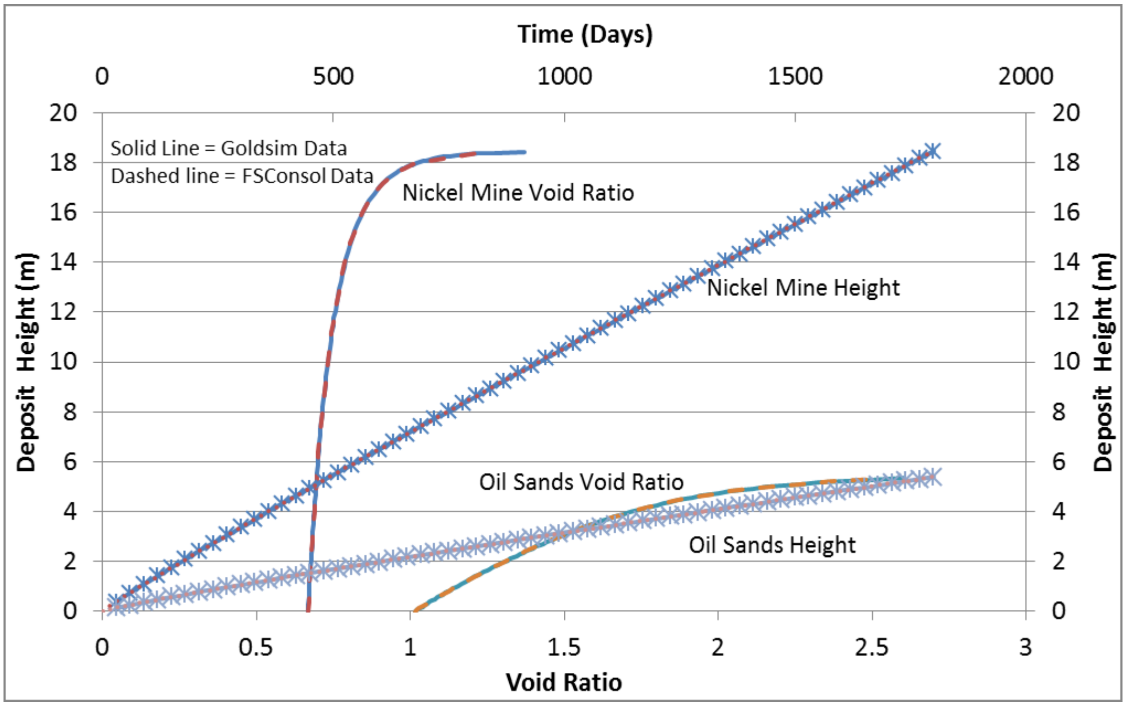


Figure 4.10. Validation of FSConsol linkage with Goldsim.

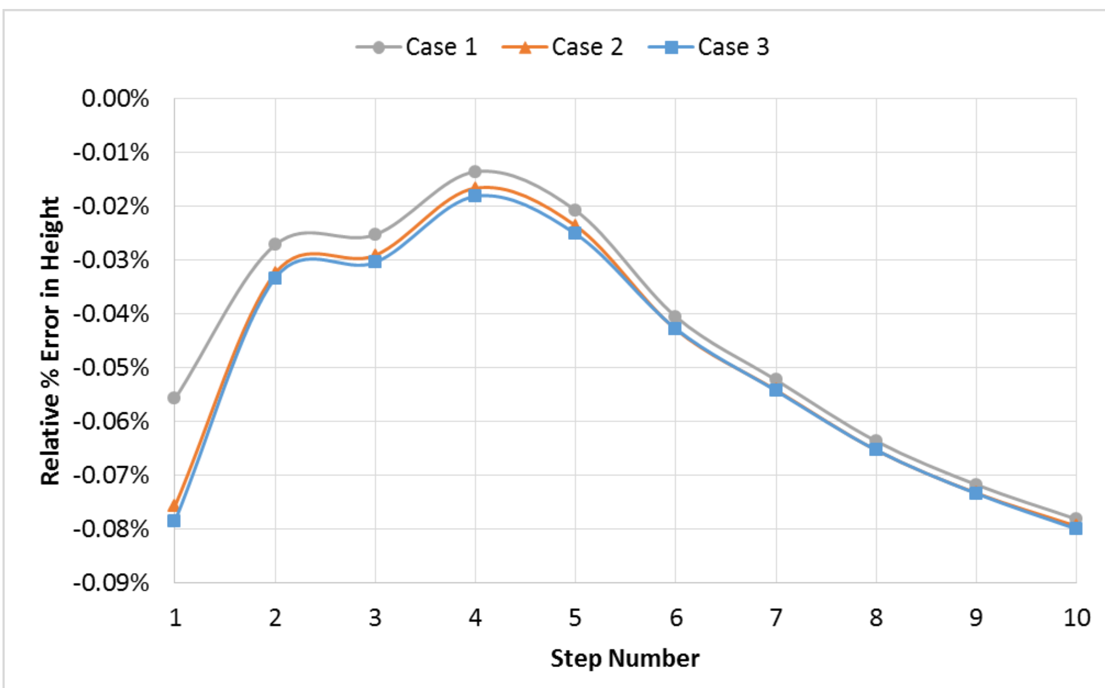


Figure 4.11. Relative percent error in height calculation for the deposition algorithm.

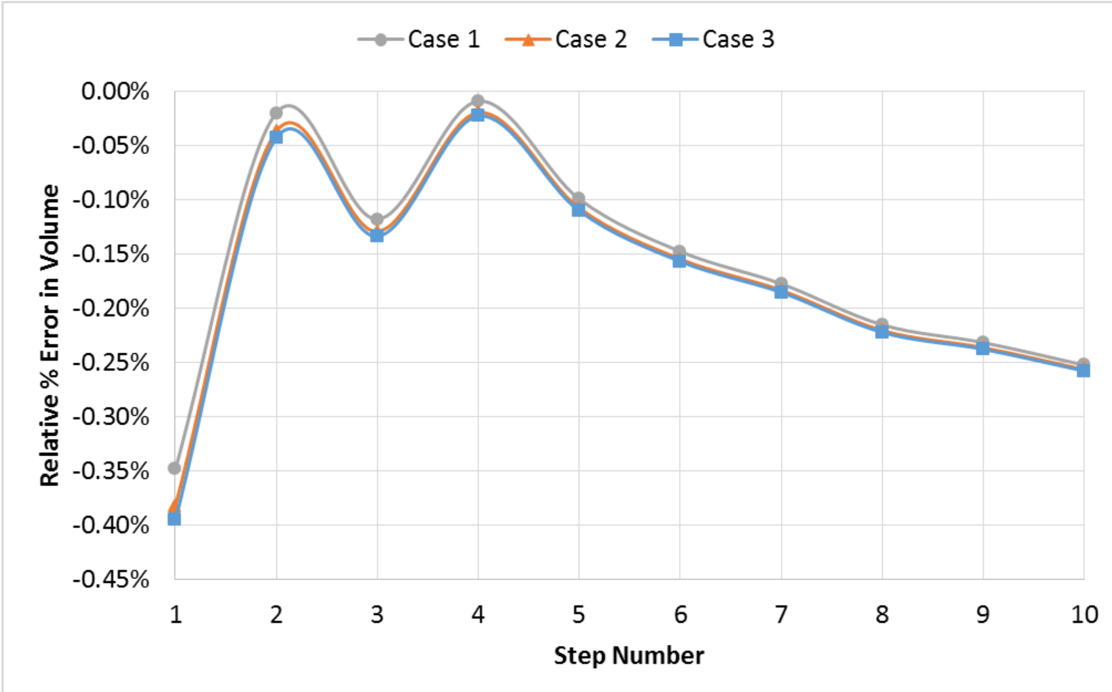


Figure 4.12. Relative percent error in volume calculation for the deposition algorithm.

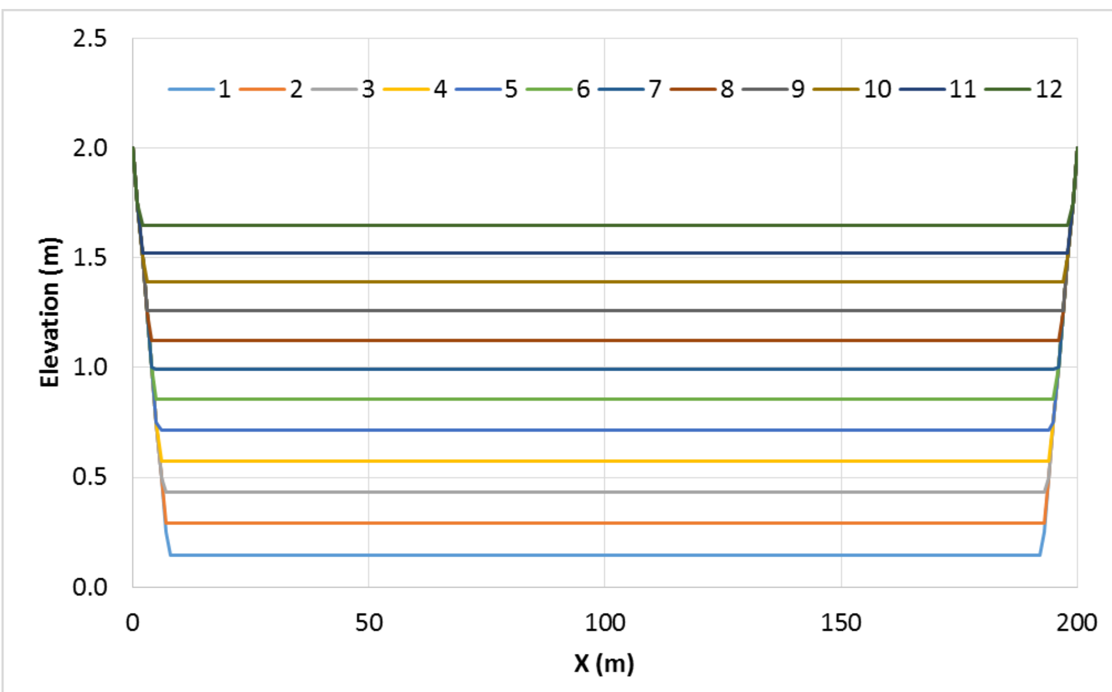


Figure 4.13. Tailings surface profiles for scenario 1, low solids content tailings.

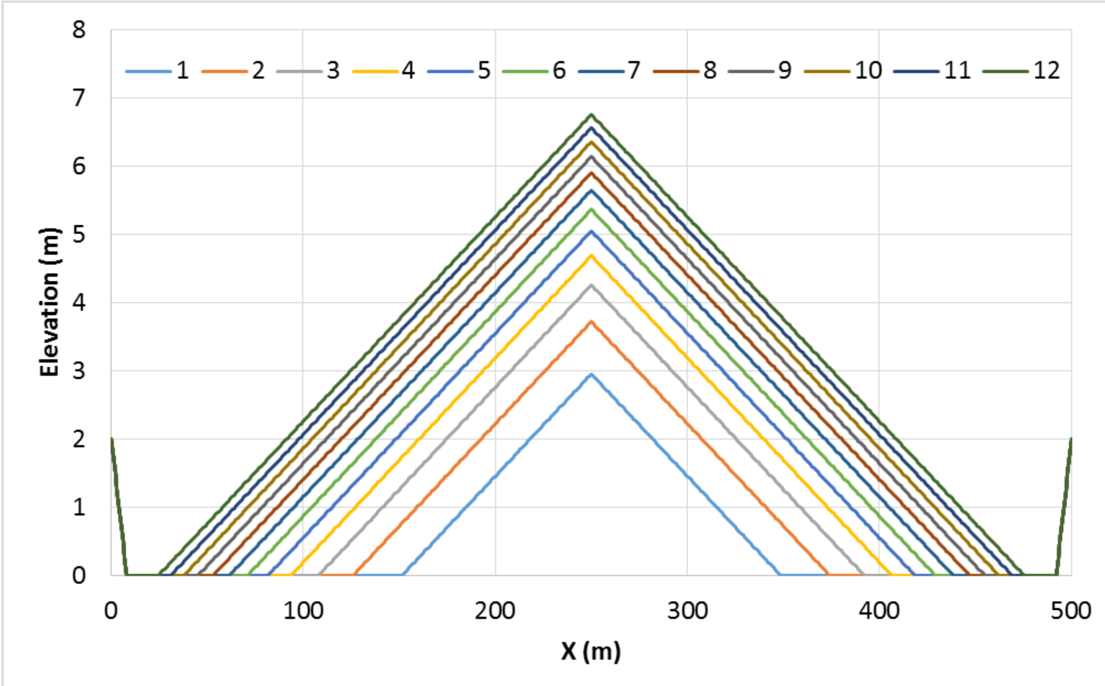


Figure 4.14. Tailings surface profiles for scenario 2, high density tailings from a central discharge.

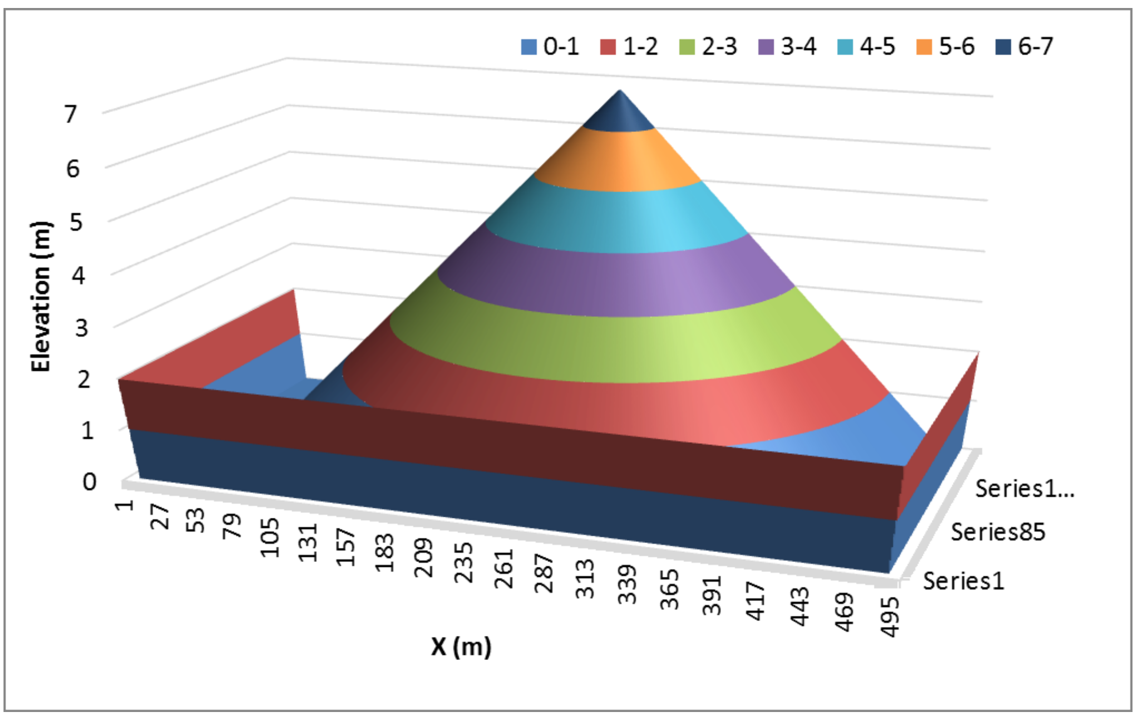


Figure 4.15. Three dimensional tailings surface profile for scenario 2, high density tailings from a central discharge.

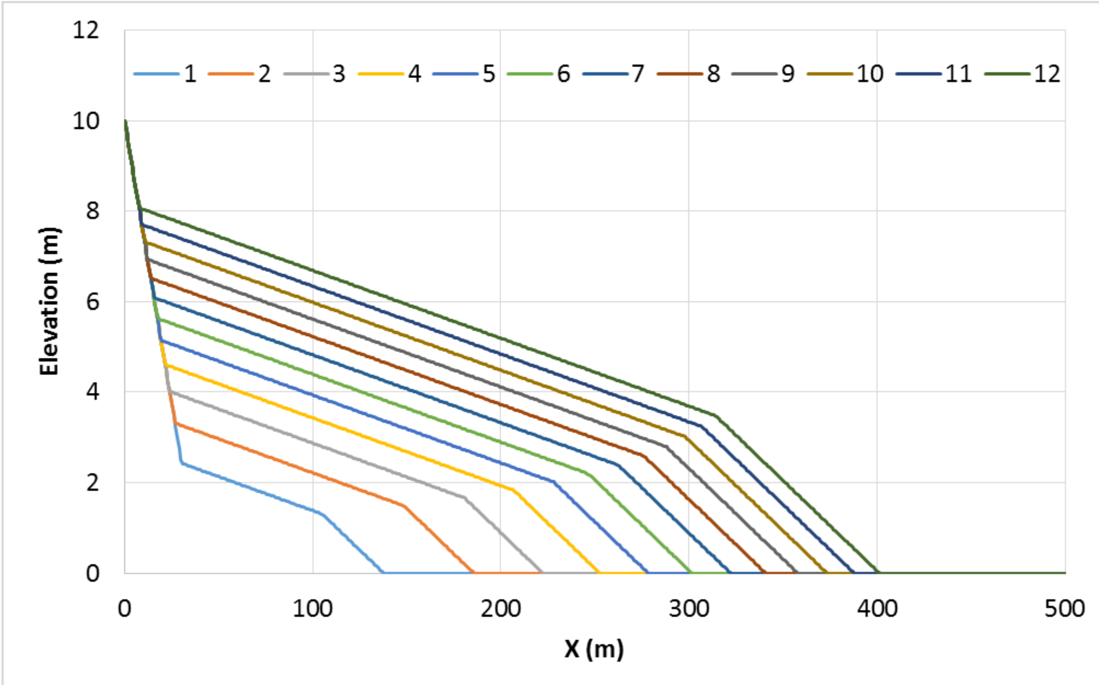


Figure 4.16. Tailings surface profiles for scenario 3, medium density tailings from a multi-spigot deposition pipe.

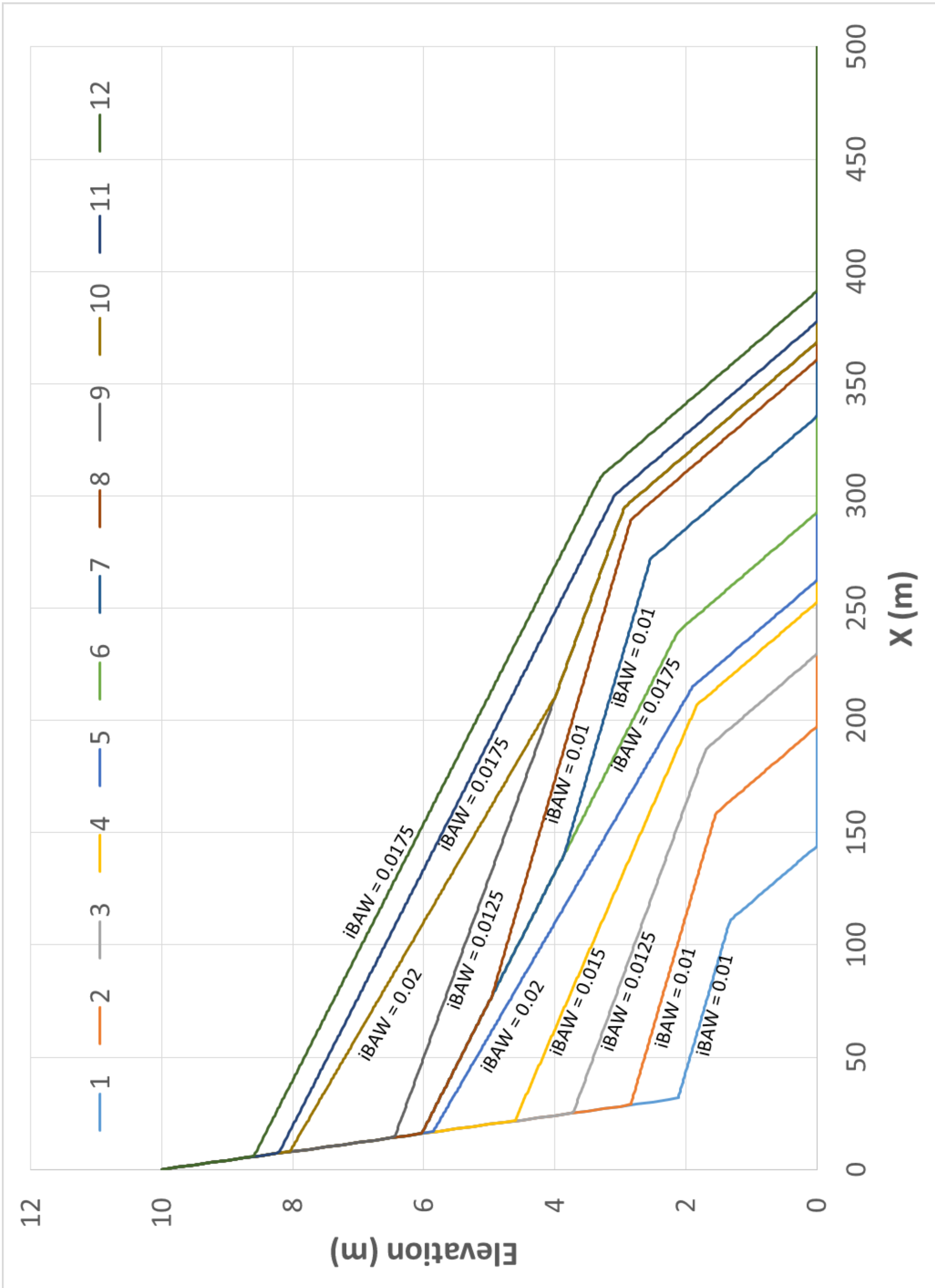


Figure 4.17. Tailings surface profiles for scenario 4, medium density tailings from a multi-spigot deposition pipe and varying iBAW (iBAW for each step is identified on the figure).

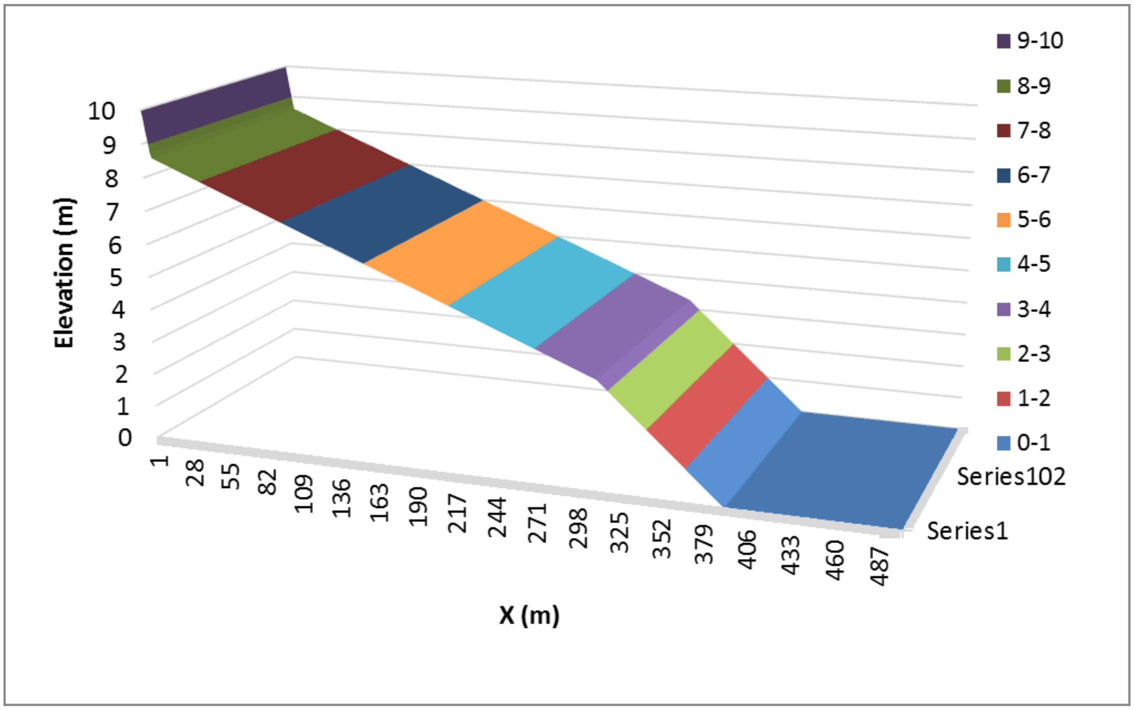


Figure 4.18. Three dimensional plot of the final tailings profile for Scenario 4.

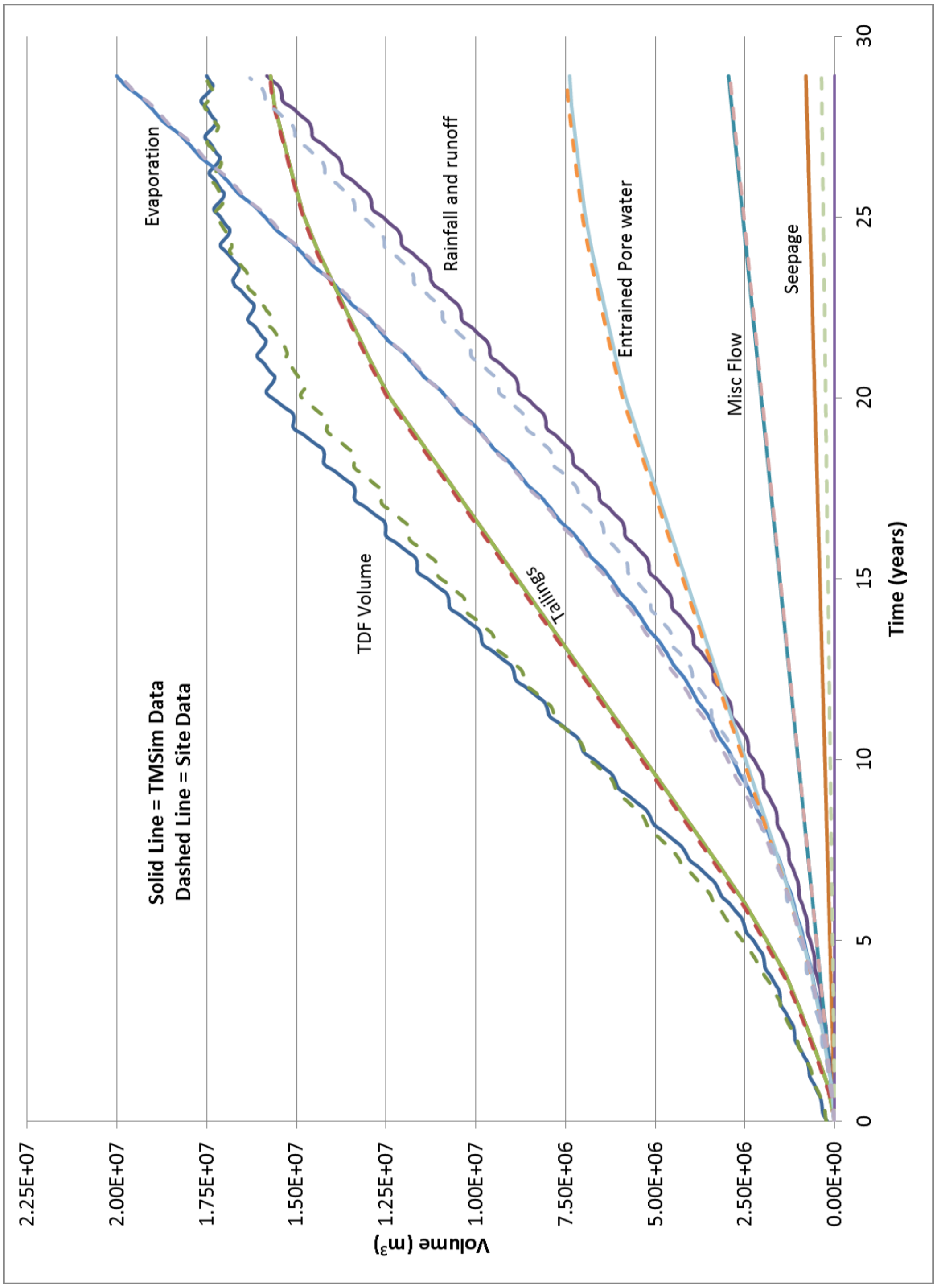


Figure 4.19. Validation of TMSim volume data with a metal mine site data set.

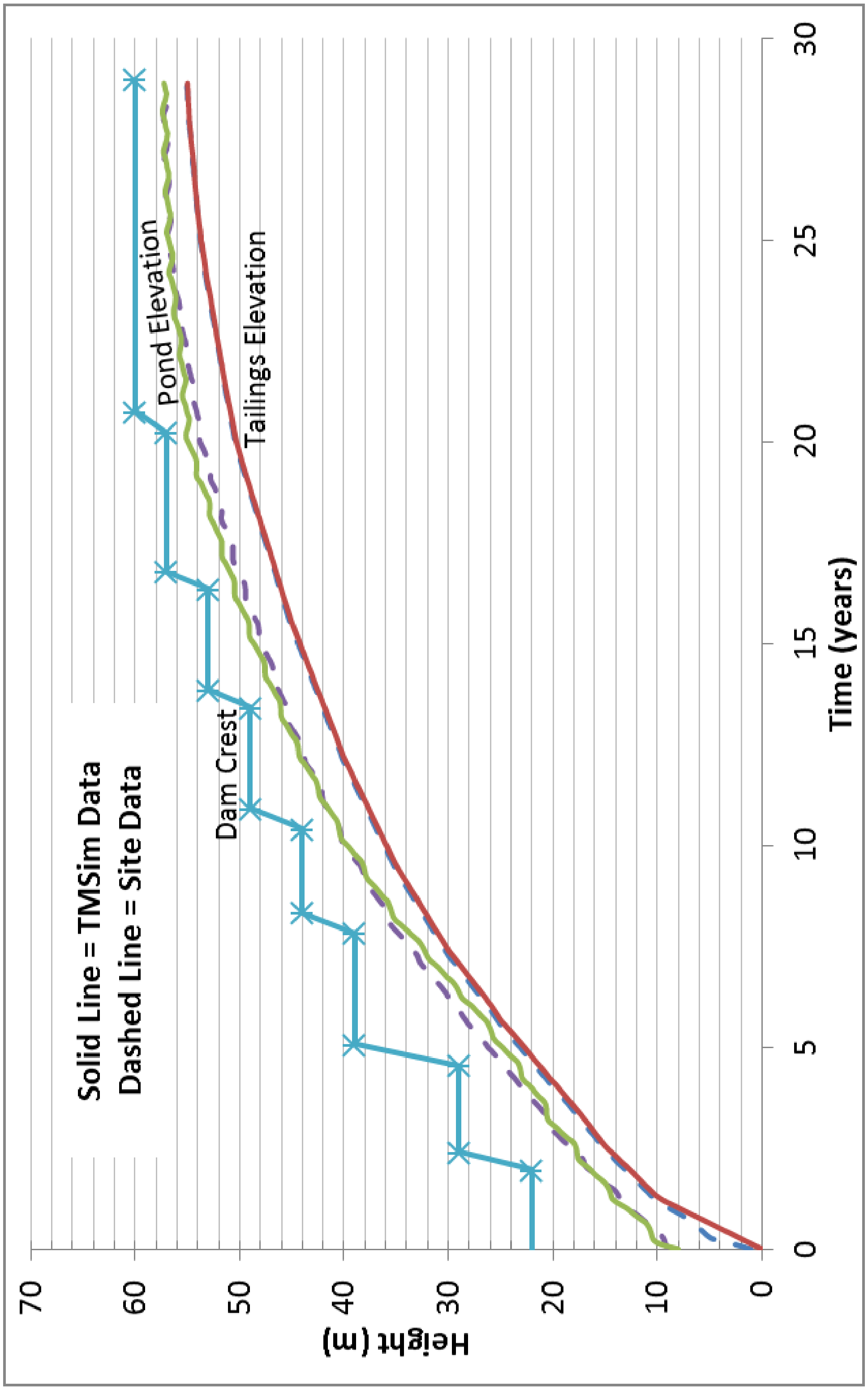


Figure 4.20. Validation of TMSim elevation data with a metal mine site data set

4.7 REFERENCES

Badiozamani, M. and Askari-Nasab, H. 2012a. Towards integration of oil sands mine planning with tailings and reclamation plans. Proceedings of Tailings and Mine Waste 2012. Keystone, Colorado, October 14-17, 2012, 609-622.

Badiozamani, M. and Askari-Nasab, H. 2012b. An integration of long-term mine planning, tailings and reclamation plans. 3rd international oil sands tailings conference. 103-110.

Beier, N., Sego, D., and Morgenstern, N. 2009. Dynamic Simulation of tailings Management Options. Tailings and Mine Waste 2009 Conference, Banff, Alberta, November 1-4, 2009. 441-450.

Ben-Awuah, E. and Askari-Nasab, H. 2011. Oil sands mine planning and waste management using mixed integer goal programming. International Journal of Mining, Reclamation and Environment, **25**(3): 226-247, DOI: 10.1080/17480930.2010.549656

Ben-Awuah, E. and Askari-Nasab, H. 2012. A mathematical programming model for oil sands production scheduling and waste management. Proceedings of Tailings and Mine Waste 2012. Keystone, Colorado, October 14-17, 2012, 593-609.

Boswell, J.E.C. and Sobkowicz, J.C. 2010. Environmental assistance for tailings disposal. 63rd Canadian Geotechnical Conference, Calgary, Alberta, 12-16 September 2010, pp. 665-669.

Canadian Natural Resources Ltd. (CNRL). 2010. Horizon tailings management plan. 13-RPT-TA-0001 Revision 2. November 2010.

Carrier, W.D., III, Bromwell, L.G., and Somogyi, F. 1983. Design Capacity of Slurried Mineral Waste Ponds. Journal of Geotechnical Engineering, **109**(5): 699-716.

Devenney, D.W. 2009. A screening study of oil sand tailing technologies and practices. Report prepared for Alberta Energy Research Institute. May 2009.

Fredlund, M.D., Fredlund, D.G., Houston, S.L., and Houston, W. 2003. Assessment of unsaturated soil properties for seepage modeling through tailings and mine waste. Tailings and Mine Waste '03. October 12-15, 2003, Vail, Colorado, USA. pp. 149-157.

Gibson, R.E., England, G.L. and Hussey M.J.L. 1967. The Theory of One-dimensional Consolidation of Saturated Clays. I. Finite Non-linear Consolidation of Thin Homogeneous Layers. Géotechnique, **17**(3): 261–273.

Government of British Columbia, 2015. Report on Mount Polley tailings storage facility breach. Independent expert engineering investigation and review panel. Province of British Columbia.

Halog, A. and Chan, A. 2008. Developing a dynamic systems model for the sustainable development of the Canadian oil sands industry. International journal of Environmental Technology and Management, **8**(1): 3-22.

Jeeravipoolvarn, S. 2010. Geotechnical behavior of in-line thickened oil sands tailings. PhD Thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, Alberta. pp. 431.

Kalantari, S. 2011. Linkage of Annual Oil Sands Mine Plan to Composite Tailings Plan. MSc Thesis, Department of Civil and Environmental Engineering, University of Alberta.

Kossik, R and Miller, I. 2004. A probabilistic total system approach to the simulation of complex environmental systems. Proceedings of the 2004 Winter Simulation Conference. Ingalls, R.G., Rossetti, M.D., Smith, J.S., and Peters, B.A. eds. December 5-8, 2004, Washington, D.C. 5 pp.

Kynch, G.J. 1952. A theory of Sedimentation. Transactions of Faraday Society, **48**: 166-176.

Masala, S. 1998. Numerical Solution of sedimentation and consolidation of fine tailings, M.S.c. thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, AB.

Masala S. and Matthews, J. 2010. Predicting development of undrained shear strength in soft oil sand tailings. Second International Oil Sands Tailings Conference. Edmonton, Alberta, December 5-8 2010, pg 31-41.

Rizzo, D.M., Mouser, P.J., Whitney, D.H., Mark, C.D., Magarey, R.D., and Voinov, A.A. 2006. The comparison of four dynamic systems-based software packages: Translation and sensitivity analysis. Environmental Modeling and Software, **21**: 1491-1502.

Sobkowicz, J.C. 2012. The oil sands tailings technology roadmap and action plan: Introduction and key recommendations. Third International Oil Sands Tailings Conference. Edmonton, Alberta. December 2-5, 2012. Pp. 13-23.

Sobkowicz, J.C. and Morgenstern, N.R. 2009. A geotechnical perspective on oil sands tailings. 13th International Conference on Tailings and Mine Waste, Banff, Alberta, 1-4 November 2009. University of Alberta, pp. xvii-xli.

Toorman E. and Gudehus G. 1998. Sedimentation and self-weight consolidation: general unifying theory, Discussion. Geotechnique, **48**: 295-298.

Vick, S.G. 1990. Planning Design and Analysis of Tailings Dams. BiTech Publishers Ltd. Pp. 369.

Wickham, M., Johnson, B., and Johnson, D. 2004. Dynamic systems-modeling approach to evaluating hydrologic implications of backfill designs. Mining Engineering, **56**(1): 52-56.

5 OIL SANDS TAILINGS MANAGEMENT

5.1 INTRODUCTION

Previous chapters have outlined the various options for managing, dewatering and reclaiming tailings streams generated from mining operations. The review was applicable to a wide range of mineral commodities including iron and copper, precious metals and stones, and hydrocarbons such as coal and oil sands. Although each of these operations involve excavation of the parent ore, an extraction process, and production and subsequent management of a wet tailings stream, oil sands mining operations have unique challenges and differences from metal mine operations. A summary of these differences are included in Table 5.1. The scale of oil sands mining operations can be an order of magnitude greater than a typical metal mining operation. Additionally, oil sands tailings management operations are dominated by a poor settling, clay-rich, fine tailings stream that has been accumulating since the industry began in the late 1960s. The following chapter will provide a summary of the historical tailings management practices and challenges in the oil sand industry and the proposed future tailings management plans based on publically available sources of information. Due to a delay in publically releasing the tailings management plans, the information presented only includes data that was available at the time of writing. The following Chapter 6 summarizes updated technologies under investigation or implemented for managing fine grained oil sand tailings that were not covered in this Chapter.

5.2 OIL SANDS TAILINGS

The Athabasca region of northern Alberta, Canada, is home to massive deposits of oil sands with an estimated reserve of approximately 170 billion barrels (bbl) of recoverable bitumen. Production of the bitumen in this region is based on open pit mining and extraction using warm to hot water based processes. Five operating oil sands companies (Suncor Energy, Syncrude Canada Ltd., Canadian Natural, Albian Sands and Imperial Kearl Lake) are currently producing bitumen at approximately 890,000 bbls/day with several new mines and expansions planned that would double production by 2025 (CAPP 2012; OSDG 2013). Oil sands are typically composed of bitumen (~12 % by mass), sand, silts, clays (mineral content ~85 % by mass) and water (3 to 6 % by mass). The clay component is comprised of mainly kaolinite (50-60 %) and illite (30-50 %) with some montmorillonite (Chalaturnyk et al. 2002, FTFC 1995, and Kaminski et al. 2009). At Suncor, the clays in the parent ore body are typically less than 1%, however they make up to 50 % of shale bed lenses within the deposit (Wells and Riley 2007). Due to their proximity, these shale beds are excavated and processed with the ore as they are too thin to be effectively segregated.

In the oil sands industry, fines denote the material passing 45 μm and will include silt, clay and residual bitumen unless otherwise noted. Solids content is calculated as the dry mass of solids (mineral solids and residual bitumen) divided by the total mass of the tailings (mineral solids, residual bitumen and fluid); sand content (>45 μm) is the dry mass of sand divided by the dry mass of solids; fines content is the dry mass of fines (silts and clays plus residual bitumen) divided by total dry mass of solids; and clay content is the dry mass of clay divided by the dry mass of fines (Boratynec 2003).

Bitumen extraction begins with crushing of the excavated ore. The crushed ore is then conditioned with warm to hot water, steam, and process aides such as caustic (NaOH) or citrate (Albian only) and hydrotransported via pipeline to the extraction plant (Chu et al. 2008, FTFC 1995, Masliyah et al. 2004, and Wells and Riley 2007). Bitumen is separated from the coarse fraction as a floating froth in large gravity separation vessels. The bitumen froth is further processed to remove fine solids. Typical bitumen recoveries range from 88 to 95 % depending on oil sands grade and origin. Tailings from the extraction process include a mixture of water, sand, silt, clay and residual bitumen and are referred to as "whole tailings" (Sobkowicz and Morgenstern 2009). The whole tailings slurry is typically discharged at a solids content of approximately 55 % C_w. The sand fraction of the tailings comprises approximately 82 % by dry mass of solids and fines at approximately 17 % dry mass of solids.

The oil sands industry uses a ternary plot (Azam and Scott 2005) of sand, fines and water content to characterize and explain tailings behavior. This plot is illustrated in Figure 5.1 Ternary diagram of oil sands tailings streams (modified from Beier and Sego 2013). and explained below (Beier and Sego 2013). The sand content (S) represents the mass ratio of sand to the total mass of the tailings (sand, fines, water and bitumen). Similarly, fines content (F) is the mass ratio of fines to the total mass. Finally, the water content (*W*, represented by the horizontal lines) is mass of water over total mass (Sobkowicz and Morgenstern 2009). Typical ranges of whole tailings and MFT are plotted on the ternary plot (Figure 5.1) for reference.

5.2.1 Historical Tailings Management

Historically, tailings were pumped into large, above grade, settling basins (external tailings facilities [ETF]) where the sand fraction settled out rapidly to form beaches. Some 50 % or more of the available fines were trapped within the sand matrix of the beaches. However, the remaining thin slurry of fines, residual bitumen and water (6 to 10 % solids content) flowed into the settling basin where the solids settled gradually at depth. After a few years, the fines settle to a solids content of 30 to 35 % and are referred to as MFT or FFT. Further consolidation of the MFT is expected to take centuries (Chalaturnyk et al. 2002, FTFC 1995). Water released as the fines slowly settled was recycled back to the extraction plant. Figure 5.2 shows schematically a conventional oil sand extraction and tailings disposal system (Beier and Sego 2007). On average, approximately 0.266 m^3 of MFT and 0.91 m^3 of sand are produced for every 1 m^3 of mined ore (Devenny 2010). These historical tailings management practices have resulted in an estimated 850 million m^3 (Mm_3) of MFT stored among the operating mine sites (Fair and Beier 2012). This volume of MFT is referred to as “Legacy Tailings”.

Water management is also an issue for these mine sites in addition to managing large volumes of fluid fine tailings. The mines are currently operating under a zero-effluent discharge policy preventing release of accumulated water from their site. Continual recycle of process water (tailings release water) to the extraction plant has led to a buildup of dissolved ions within the recycle water. Elevated ion concentrations can lead to various operational issues including poor extraction recovery and scaling/fouling of piping and equipment.

In an effort to deal with the Legacy tailings and provide a stable landscape in a timely manner, the industry implemented non-segregating tailings technology [NST at Albian (Matthews 2008) and CNRL (Chu et al. 2008), Composite tailings or CT at Syncrude (Matthews et al. 2002), and Consolidated tailings or CT at Suncor (Shaw 2008)]. CT tailings are a mixture of coarse sand, coagulant (gypsum at Syncrude/Suncor/Albian or carbon dioxide at CNRL) and MFT at SFR gravimetric ratios of approximately 4:1. To produce NST or CT, total tailings from the extraction plant are passed through a hydrocyclone with the overflow (COF; mainly fines) pumped to a settling basin to form MFT. In some cases (i.e. Albian/CNRL) the COF or floatation tailings are sent to a thickener to further dewater the fines and recover heated water, now referred to as thickened tailings (TT), prior to deposition in the settling basin or to be used directly in NST. Coarse CUF is then be combined with dredged MFT or TT and a coagulant to form the CT or NST. The resulting mixture is pumped to the disposal area. If the NST/CT mixture was prepared and deposited according to the design specifications it should not segregate during transport, discharge, or deposition. Production of NST or CT provides an opportunity to consume the Legacy tailings and also releases water rapidly for reuse (Matthews et al. 2002). A typical NST process is shown schematically in Figure 5.3 (Beier et al. 2009).

Disposal areas for NST or CT mixtures include constructed cells within the mined out pit or an ETF. As the in-pit mine face progresses, additional deposition cells may be constructed with available coarse CUF, overburden materials or lean oil sands. The NST/CT deposits may then be capped with CUF (sand) and overburden prior to final reclamation. Under the current operating approvals, all runoff water from the NST/CT deposits and remaining fine tailings not consumed by NST/CT production will be transferred and stored in pit below a water cap in what the oil sands industry refers to as end pit lakes (EPL).

When NST was initially implemented, it was anticipated that the deposits would reach a geotechnically stable state in a timely manner so reclamation activities could proceed. However, reduced dewatering rates are preventing the deposit from reaching the strength required to support reclamation activities. Also segregation of the NST was occurring upon deposition allowing fines to re-suspend in the pond. Variable clay content in the tailings stream due to heterogeneous oil sands and/or difficult to control tailings management and deposition techniques are factors that contribute to segregation of NST. Production of NST and consumption of fines also relies on a continuous supply of coarse sand from the extraction plant. When coarse tailings are needed for dyke construction or upsets in the extraction process reduce sand production, fine tailings have continued to accumulate.

Over the past 10 years, only Syncrude and Suncor have produced CT at the commercial scale. To date, the technology has not performed as expected and no CT deposits have been fully reclaimed. Additionally, plans to place fluid tailings remaining at the end of project life in end-pits capped with water have yet to be demonstrated on a commercial scale (Houlihan and Mian 2008). The ERCB has expressed concerns over the past years regarding the current tailings management practices, continual accumulation of fine tailings and associated risk to reclamation activities. As such, they decided 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). Based on the challenges of implementing NST/CT experienced at Suncor and Syncrude, the recent ERCB Directive 074, and increasing public awareness, the industry is looking at alternative and complementary tailings management options to reduce their inventory of fluid fine tailings and expedite the reclamation process.

5.2.2 Fluid Tailings Management

Essentially, there are three general methods to reclaim fine tailings (i.e MFT) into geotechnically-stable landforms or functional aquatic ecosystems (EPLs) (Fair and Beier 2013). Where "engineered above grade landforms" are planned for the closure landscape, the fines tailings can be re-combined with the coarse tailings sand (i.e. sequestering fines within the sand matrix as NST or CT), or dewatered separately from the coarse tailings creating a cohesive, silty-clay deposit. In the case of aquatic ecosystems, fine tailings can be placed in an engineered mine pit under a water cap to form a lake referred to as EPL.

The oil sands industry is trending toward managing the fine stream using chemical additives in addition to physical and environmental dewatering techniques in order to meet the Directive 074. Fines tailings dewatering techniques under investigation include mechanical centrifuges, large thickener vessels, and chemical dewatering in combination with strategic deposition. This process has been termed "*in-line* flocculation" or ILTT, atmospheric fines drying at Shell (AFD) or Tailings Reduction Operations (TRO) at Suncor. The processed tailings may be discharged in thin lifts onto gently sloped beaches where further dewatering is achieved via a combination of settlement, seepage and environmental dewatering (desiccation and freeze/thaw). Alternatively, the tailings may be discharged continuously (thick lifts) into large depositional cells (>10 m deep) to promote self-weight consolidation, seepage, and environmental dewatering via evaporation at the surface only. The fines dewatering processes utilizing chemical amendments to manage fine tailings and their associated implications on tailings management are included in Chapter 6 and not discussed further in this Chapter.

5.2.3 Site Specific Conditions Encountered in the Oil Sands

Site specific conditions may preclude certain technologies from being applied at a particular mine site (CTMC 2012). Site specific conditions that may influence the chosen tailings management plan at an oil sands mine are included in Table 5.2 and summarized below in detail. The geology of a particular mine site will have the greatest influence on a tailings management plan (foundation conditions, stratigraphy, ore body configuration and quality). Foundation conditions at a site will have implications on the geotechnical stability of the onsite structures both external and in-pit. Foundations may be strong and competent to weak and very soft. This will impact the dimensions and design of containment dyke structures which ultimately will impact the storage capacity of constructed impoundments. Poor foundation conditions will require a greater amount of suitable construction material for containment structures (flatter slopes) and possibly slower rise rates (to allow pore pressures to dissipate). There may be circumstances where dyke construction (and capacity) is constrained due to suitable construction material supply. The saturated hydraulic conductivity of the foundation is also an issue related to location of impoundment structures. Highly permeable foundations may provide migration pathways for process affected waters to escape the containment facility. The location, configuration, and design (engineered liner or interception technologies) of impoundment structures will be impacted.

The stratigraphy of a particular site will impact the suitability and supply of mine waste materials for construction purposes. This will include overburden, intraburden, and coarse tailings. Implications of insufficient quantity of suitable construction material include slower rise rate of impoundments (reduced capacity), limit the options of available tailings technologies (if competition for sand becomes an issue). The geometry of the actual ore body can influence several components of a tailings plan. The amount of ex-situ area available for storage (footprint of impoundment), time until in-pit space is available, storage efficiency (amount of dyke construction needed per unit of tailings storage) are dependent on the ore body configuration. The quality of the ore body may also limit the tailings technologies available at a particular site. In scenarios where there are high fines content ore bodies, more water may be required for extraction (potentially straining recycle water or increasing storage requirements) and require supplemental fines tailings management technologies. Other factors include topography and proximity to nearest industrial facilities (lease boundaries). These factors will impact the amount of area available for containment facilities and configuration of impoundment structures.

5.3 2011 Oil Sand Tailings Management Plans

There are four oil sands mining operations currently operating with several new mines and expansions under development. The following sections will provide an overview of the tailings management plans for each of the sites (operating and planned) in no particular order. This data was compiled based on the ERCB's Directive 074 2011 tailings reports from each company, the AERI Screening Study of Oil Sands Tailings Technologies and Practices (Devenny 2010), and other public literature. The information contained herein was based on publically available materials up to December 31, 2011.

Tailings plans that were reviewed for this study only include sites that have a single external tailings impoundment (ETF) with eventual deposition into the mined-out pit. Syncrude Mildred Lake and Suncor operations were not included due to the complexities of their deposition plans (several interconnected tailings facilities and deposition points for tailings and water streams). Typically, the submitted ERCB Directive 074 tailings plans contained detailed information over the period of 2011 to 2029. Beyond 2029, the information was estimated with a lower degree of certainty and on 5 year intervals. Therefore, available ore body and waste material balances were only compiled up to 2029.

5.3.1 Shell Canada

5.3.1.1 Muskeg River Mine

The Muskeg River Mine (MRM) currently operates at an average production of 150,000 bbls/day of bitumen (Shell 2010a). Shell's chosen technologies for tailings management at the MRM site include composite tailings CT and AFD until 2019 switching to NST from 2019 and beyond. The CT process will use MFT recovered from the ETF as feedstock and the NST process will use TT as a fines feedstock. Prior to 2012, extraction tailings will be deposited in the ETF until sufficient space is available in-pit. In-pit deposition of CT will commence in

2012 transitioning to NST in 2018. Segregated fines will be removed from the in-pit deposition cells and re-cycled to the tailings system for consumption within the NST process. Any off-spec coarse deposits (low SFR) will be remediated in place with soft tailings capping technologies. Up to 6.8 million m^3 of fines tailings per year are expected to be treated by the AFD process, however, yearly storage efficiency (m^3 of fines/m^2 area) was not provided. Remediation techniques for off-spec materials arising from the AFD process were not specified. Fluid fine tailings are currently deposited into an in-pit cell and the ETF is used for process water clarification and storage. At the end of mining (2059) there is a residual of 172 million m^3 of MFT stored within the pit and 5.8 million m^3 within the ETF. Reference to end-pit-lakes are not included in the 2011 Directive 074 report.

5.3.1.2 Jackpine Mine

The Jackpine mine site (JPM) is operating at 100,000 bbls/day, half the approved capacity (Shell 2010b). Bitumen froth produced at JPM is transferred to MRM for processing. Fines associated with the JPM froth are managed at the MRM mine site. Whole tailings from the JPM process are segregated in a hydrocyclone to produce a coarse tailings stream (CUF) and a fines dominated COF. The COF is then dewatered in a thickener to produce TT. During upsets in the process, whole tailings will be deposited onto a beach or used for cell construction. Runoff from the whole tailings deposition are expected to form MFT. Tailings will initially be managed as separate streams in the ETF. The ETF is separated into cells for TT drying, sand storage and fluid fines/water storage. The ETF will be constructed initially from mine waste and then compacted cell sand (coarse CUF). TT will be deposited into alternating cells (summer and winter operations). Dewatering will rely on strategic deposition, evaporation, freeze-thaw, and subsequent mechanical manipulation to enhance dewatering when required. TT deposited in the summer months (March to October) will be transferred to a second drying area during the winter months. The material will then be re-handled in the fall and placed into dumps, dykes or in-pit cells. Any shortfalls from the TT deposition and drying within the ETF will be treated with a centrifuge system. After 2027, tailings deposition will move in-pit and utilize NST technology by combining underflow sand, TT and MFT. At the end of mining, an estimated 29.8 million m^3 of fluid fines tailings will be stored within the pit and 6 million m^3 within the ETF.

5.3.2 CNRL Horizon

The CNRL Horizon mine is operating at 270,000 bbl/day. CNRL plans to utilize NST technology and manage any residual fines with a thin lift drying operation (CNRL 2010). Up to 2015, all tailings will be deposited in the ETF and will segregate to form MFT and coarse beaches. NST will then be placed in the ETF until sufficient in-pit space is available starting in 2020. After NST is initiated, all in pit dykes structures will be constructed with mine waste, to ensure sufficient sand is available for the NST process. The NST will be made from cyclone underflow coarse tailings, thickener underflow fine tailings, and carbon dioxide (CO_2) as the coagulant. CNRL assumes that the NST process will operate at 85% (defined as 85% of time the fines in NST equals amount of fines in ore body). Following 2015, thin lift deposition of polymer and CO_2 amended fluid fines will be used to manage MFT not used in NST. Thin lift deposition of fines will occur in an external disposal area. Residual MFT/fluid fines from the NST process will be managed with the thin lift drying operation. The current design is treat 2.0 m^3 of fine tailings/m^2 per year via thin lift drying in external disposal area which equates to 5.7 million dry tones of fines/year. However, CNRL estimates the actual thin lift production will likely be closer to 1.2 m^3/m^2. During beaching operations, up to 60% of fines can be captured in the sand pore space, but the planned design value is 45%. For planning and design purposes, the generation of MFT is based on a simple relationship between fluid fine tailings (beach run off) and time (Table 5.3). When fluid fines tailings are generated (beaching and runoff conditions), CNRL estimates approximately 0.18 m^3 of MFT is created per tonne of ore. They assume this value is conservative based on historical data from other operators which ranges from 0.11 to 0.16 m^3 of MFT/tonne of ore. Any residual MFT at the end of mining (180 million m^3) will be placed in an in-pit lake.

5.3.3 Imperial Oil Canada

The Imperal Oil (Imperial) Kearl ERCB Directive 074 tailings plan was developed for production capacity of up to 260,00 bbls/day (Imperial 2010). The Imperial mining and extraction process will feed oil sands ore to a primary separation vessel (PSV) where coarse underflow will be deposited in an ETF. PSV middlings will be further processed in a floatation unit. Floatation tailings will also be deposited initially into the ETF. Additionally, tailings solvent recovery unit, (TSRU) tailings will be deposited into the ETF. Kearl assumes MFT will take 2-3 years to reach a solids content of 30 weight %. All extraction and process tailings will be placed into the ETF up to 2018. After which, TSRU and sand deposition will commence in pit starting 2018. MFT from ETF will be removed and treated with thin lift drying. MFT treatment will occur in an above grade disposal area. Overburden and interburden mine waste will be used for dyke construction or stockpiled in dumps. Some coarse sand will be used for dyke cell construction. TSRU tailings will be co-disposed in the ETF with PSV tailings until 2018, then 80% of the TSRU will go to the in-pit facility. Floatation tailings have low SFR and will form MFT upon deposition. Floatation tailings will be deposited in the ETF until 2034 when in pit space is available and the ETF is full. MFT in the ETF will be treated with thin-lift or other unspecified technology after 2018. Imperial estimates approximately 2.25 m^3 of MFT per m^2 of area can be placed per year in the disposal area. The disposal areas will consist of coarse sand beaches or mine waste disposal areas. MFT remaining at the end of mining (78 million m^3) will be deposited in an EPL.

5.3.4 Syncrude

5.3.4.1 Aurora North

The Aurora mine site will include mining operations and bitumen froth production (Syncrude 2010a). Approximately 117 million tonnes of ore are planned to be mined annually. Bitumen Froth will be sent to the Syncrude Mildred Lake mining and extraction operations for further processing. Mineral solids contained in the froth stream will be managed at Mildred Lake. Aurora North is integrated with Mildred Lake operations but some tailings will be deposited on site. Tailings technologies at Aurora will include CT and MFT end pit lakes. Currently, coarse tailings are deposited into the ETF or an in-pit cell with conventional beaching operations. Containment dykes are being constructed with mine waste overburden, tailings sand or a combination of both. MFT is currently being accumulated in the ETF from conventional beaching deposition techniques. Starting 2013, CT will be comprised of MFT and cyclone UF tailings and deposited in-pit. Cyclone OF fines will be deposited into the ETF to form MFT. Eventually a thickener will be added to manage the additional fines and produce feedstock for the CT plant. When the CT plant is not operating, coarse tailings will be deposited onto previous CT deposits and generate run off (fines and water). The fines and water will be collected and transferred to the ETF. CT performance is estimated at 95% and TT facilities performance is at 75%, however the “performance” metric is not clearly defined. A supplemental fine tailings technology is under investigation (TT, centrifuge, accelerated dewatering, etc.). Approximately 40 million m^3 of process water is stored on the site. At the end of mine life there will be approximately 175 million m^3 of MFT to be transferred to an EPL.

5.3.4.2 Aurora South

Aurora South is a new mine site currently in the development and design stage (Syncrude 2010b). The tailings management plan for Aurora South will include placement of tailings into and ETF and then in-pit once space is available. Dykes will be constructed of overburden and/or coarse sand cells. Conventional beaching and dyke construction will generate fluid fine tailings that will create MFT. MFT will be subsequently dewatered with centrifuges and placed in dumps out of pit as well as in-pit. The centrifuge cake will be placed at 55% solids content. Coarse sand will be deposited in pit. The tailings plan currently is conceptual, therefore limited data was provided. At the end of mine life there will be approximately 48 million m^3 of MFT and will be stored in an EPL.

5.3.5 AERI Screening Study

The AERI published a study in 2009 on oil sands tailings technologies and practices referred to as the "AERI model" (Devenny 2010). The study included background information on mineable oil sand projects including tailings and reclamation components. This information was then used to screen alternative tailings technologies. A database of relevant mining and tailings planning parameters was compiled from public sources and review of current operating mines. A "base case" mine and tailings plan was developed as part of the screening study and it represents the early Syncrude Mildred Lake operations.

The AERI model mine plan includes an ETF initially constructed with overburden switching to conventional beaching of whole tailings. Beaching of whole tailings into the ETF results in the production of MFT. When sufficient space is available, CT technology is used to deposit tailings in-pit (after approximately 6 years). Residual MFT will be managed with an EPL.

Site specific assumptions used in the model include:

- 100,000 bbl/day production and extracts 90% of bitumen in place.
- 30 year life span of project.
- Overburden is 50 m thick and is composed of surface muskeg (0 to 2 m thick); Weak surface soil within the top 5 m; and the remainder is glacial till (25% of which contains material derived from Clearwater clay shale).
- Mine pit is 4 km square and 100 m deep. Mine advances in 1 km^2 blocks.
- Each million m^3 of ore results in 2 ha of ground disturbance.
- Overburden dumps are max 25 m high.
- Extraction efficiency is based on ERCB ID 2001-7

The screening study claims that each cubic metre of ore could form 0.91 m^3 of MFT if all fines are managed separately from the coarse stream (no beaching or cell construction). However, historically only 0.266 m^3 is formed based on Syncrude data. This implies only 30 % of fines in the ore actually make MFT. The study speculates that the other fines settle as silt or are captured in beach deposits.

5.3.6 Summary of Mine Plan Data

5.3.6.1 Ore Quality

The ore body geology and grade are major driving mechanisms for tailings plans as they dictate the water consumption and mineral solids production rate. Based on the data from the years 2011 to 2029, the average ore body composition for each mine site is compiled in Figure 5.4. The average bitumen grade is nearly constant at each site and compares well with the AERI model. The fines content, however, varies significantly from among sites. The AERI model assumed an average higher fines content than most sites (except Aurora).

The yearly variation in ore composition will depend highly on the actual mine site geology and mine plan. The yearly bitumen and fines content are presented in Figure 5.5 and Figure 5.6 respectively. The average ore body components for the oil sands data sets reviewed was (Bitumen = 11.2%, Water = 4.7%, Coarse = 71.4%, Fine = 12.7%). The fines content at each site varies by up to 6 % over the time frame evaluated. Bitumen content also varied nearly 2 % at the mine sites. The average fines content from all the sites is 12.7% which is lower than the AERI model value of 16%. The average bitumen and water contents are nearly identical to the AERI model. The ratio of bitumen to fines content of the ore bodies for the years 2011-2029 is shown on Figure 5.7 to determine the relationship between ore grade and fines content. There is considerable scatter to the data, but it is evident the Shell JPM site has higher grade ore with lower fines. Shell's MRM and the Imperial Oil Kearl operation have a similar distribution with CNRL having on average a lower grade/higher fines content ore. Aurora North has the highest fines content and a wide range in ore grade.

5.3.6.2 Production Schedule

The ore production schedule is also a major driver in a mine plan. A summary of the five mine site production schedules is provided in Figure 5.8. CNRL and Kearl will not realize full production until after approximately 2017. The AERI model utilized a production schedule that was at the low end of the current mine plans (only similar to Shell's JPM). Most mine sites are operating at double the AERI production rate. Also included on the figures is the full scale average daily bitumen production (barrels per day) for each site. The production life of the mines range from 30 to 56 years with an average of 45 years.

5.3.6.3 Overburden and Waste

In addition to the ore body, the amount and timing of overburden and waste materials is important. These materials are typically utilized as construction materials in containment structures. Overburden not used for construction must be strategically stockpiled as waste dumps so as not to sterilize minable ore. The ratio of mined ore to mined waste (overburden, interburden and plant rejects) as reported in the Directive 074 plans from the years 2011 to 2029 is presented in Figure 5.9. Aurora North mine has the highest average ore to overburden ratio. The other mines sites have ratios typically between 0.5 and 1.25. An ore to waste ratio of 1 was used in the AERI model.

Related to the waste schedule is the amount of material that is deemed suitable for use as construction material in the containment structures. The fraction of the total waste that is suitable for construction at each site is shown in Figure 5.10. The Kearl mine plan does not include suitable construction mine waste until after the 2029 period and thus relies on coarse tailings for impoundment dyke structures. CNRL on the other hand, has the highest percentage of suitable construction waste materials and this is reflected in their dyke construction plans. The AERI model assumed that all mine waste is suitable for construction.

5.3.6.4 Water Usage

An estimation of the water consumption at each site was calculated by compiling data from the material balance tables in the ERCB Directive 074 reports. Using the mass balance data from these reports, it was possible to determine the volume of water used in the extraction process (except in the Syncrude data tables). The water use intensity at each site is defined as tonnes of water per tonne of ore excavated (Figure 5.11). There is some scatter, but a general trend of a slight increase in water intensity with increasing fines content is evident. There is however, a significant difference in the typical water intensity among sites. It could not be determined as to what contributes to the differences, but may be related to parent ore body type (alluvial vs marine ore) or the extraction process. The water intensity was also compared to the ratio of bitumen to fines in Figure 5.12 to assess the impact of the bitumen/fines ratio on water use. Again, as the fines content increased (decrease in Bitumen/fines ratio) the water intensity also increased. In both cases the AERI model water intensity appears to be approximately equal to the average water intensity among the sites.

5.3.6.5 Tailings and Material Properties

A key component to each of the submitted tailings plans and the AERI model were the planning assumptions associated with the tailings and material properties. These assumptions include Sg, dry densities of tailings and waste deposits, fines capture (SFR) and beach slopes. The information is summarized in the

Table 5.4 and Table 5.5. Upon review of both the CNRL Directive 074 2009/2010 and 2011 plans and the Kearl 2010/2011 plans, it appears that Kearl may have inadvertently switched their Sg values in their recent submission. The Sg of fines should be lower due to the presence of bitumen (according to CNRL's 2010 submission).

5.3.6.6 Fluid Tailings Production

Using information provided in the ERCB Directive 074 tailings plans, pond status reports, and publically available data (Oil Sands Information Portal), it was possible to compile the historical and planned MFT production rates for CNRL, Shell and Syncrude. The production values are presented in Table 5.6 and were calculated based on the following approach. At the Shell MRM (Shell 2010c) and Syncrude Aurora (Syncrude 2010c) mine sites, ERCB Pond status reports were available for 2010 which included the total volume of fluid fine tailings stored within the ETF structures and in-pit. Bitumen production data was assembled from the Oil Sands Information Portal (AESRD 2013). A long term average MFT production rate was calculated by using the total volume of fluid tailings (as of June 30, 2010) divided by the total bitumen production (to June 30, 2010). Shell MRM has been operating since 2002 and has produced 0.186 m^3 of MFT per barrel (bbl) of bitumen produced. Syncrude Aurora has been operating since 2000 and the long term average is 0.155 m^3 of MFT/bbl of bitumen. The Aurora MFT production is about 17% lower than at Shell MRM. However, at Aurora, bitumen froth, which contains some fines, is shipped to the Syncrude Mildred Lake operations, therefore the total fines available to make MFT is lower. Over the period of 2011 to 2029, the average amount of fines shipped to Mildred lake accounts for 7.2% of the total mass of fines. Given the differences in ore body and extraction processes between the sites, accounting for the transfer of fines to Mildred Lake, and the accuracy and inherent measurement errors in the pond status measurements techniques and production reports, the MFT production values are comparable between Shell and Syncrude.

The MFT production method adopted by CNRL is based on a conservative estimate using historical operational data from other oil sand operators. To compare the MFT production estimate, CNRL's data must first be converted from tonne of ore to barrel of bitumen. The average bitumen production from CNRL's Directive 074 report for the period of 2011 to 2029 is 0.599 bbl/tonne ore. The resulting MFT production rate is 0.3 m^3 MFT/bbl of bitumen. CNRL states this value is conservative based on the historical range provided (0.183 to 0.267 m^3 MFT/bbl bitumen). The AERI base case model was based on early Syncrude historical data and was reported as 0.266 m^3 of MFT / m^3 of ore or 0.21 m^3 of MFT / bbl bitumen.

5.4 TAILINGS DEPOSITION FACILITIES

The water based extraction process currently employed by the oil sands industry produces a slurried tailings stream that must be contained within constructed storage facilities. During the start-up phase of a new mine site, these tailings must be contained above grade in an ETF. The ETF also serves to store recycle water used in the extraction process, and non-releasable water such as precipitation that contacts oil sands, dyke drainage water and run-off from waste dumps (McRoberts 2008). Due to the abundance of quality ore deposits, mine sites are constrained in available surface area. Tailings facilities, waste dumps and the extraction plant compete for limited space (Sobkowicz and Morgenstern 2009). The sizing of the ETFs are further complicated by limits on the rate of construction for structural components and weak foundations that necessitate flat downstream slopes (Morgenstern et al. 1988). An ETF must have sufficient storage to contain the necessary process water and fluid tailings (MFT). Once sufficient space is available, deposition of whole tailings or engineered products (ie NST) typically moves to constructed cells within the mined out pit.

5.4.1 External Tailings Facility

Containment dykes forming the ETF are usually constructed with overburden as a starter dyke, switching to hydraulic construction techniques (using tailings) to the final design height (McRoberts 2008). Individual mine and tailings plan will dictate the amount of suitable overburden, waste material and tailings available for dyke construction. At least one main dyke is typically designed as an all overburden structure (McRoberts 2008) and CNRL plans to construct their entire ETF with overburden (CNRL 2010). The starter dykes are used to retain water for the start-up of extraction at a new mine. As tailings are deposited into the ETF, beaches below water (BBW) form from loose segregated sand. As tailings deposition continues, the beaches raise from below the water to form beaches above water (BAW). An upstream or modified centerline construction technique is employed to raise the dykes using compacted segregated coarse tailings in a cell construction method (McRoberts 2008). The configuration of an external tailings pond depends on several site specific factors and topography is a major component. The ETFs tend to be ring-dyke structures since the topography does not allow for classical valley fill dykes (McRoberts 2008). However, CNRL does utilize topography for containment along one side of their ETF (CNRL 2010). Devenny (2010) reports typical ETFs can be 50 to 100 m in height and at least 50 m wide at the crest. Constructed side slopes range from 4:1 for in areas with competent foundations and 15:1 over areas with weak foundations. The footprints of an ETF can be 15 km^2 or greater. The active design deposition for the ETFs before in-pit space is available is about 6 years. Table 5.7 provides a summary of the available information for the ETFs from Shell, CNRL, Imperial and Syncrude Aurora mine sites from the respective Directive 074 reports unless otherwise stated.

5.4.2 In-Pit Dykes

In-pit deposition is an integral component of all the oil sand mine waste plans. Once sufficient space is available in the mined out pit, tailings deposition will move from the external tailings facility to constructed in-pit impoundments. The in-pit dykes may be constructed of overburden and other suitable mine waste or they utilize the coarse fraction (sand) of the tailings. The location, configuration, and construction material varies among the sites and depends on the individual mine plans (McRoberts 2008). In-pit dyke designs for existing operations only (Shell MRM, CNRL, Syncrude Aurora) and the AERI model are summarized in Table 5.8.

Shell JPM and MRM have same designs as do Aurora North and South. The design for Shell's in pit Cell 1 will utilize lean oil sand and overburden for construction material at a rate of 16-25 million bulk m^3 per year. Detailed design information was not included in the CNRL Directive 074 report, however details could be inferred from the available information. All in pit dykes will be constructed with overburden and waste materials. In the south pit, three disposal areas (DDA 2, 3, and 4) are planned. CNRL plans to construct dykes at a rate of up to 70 million m^3 per year. At the Aurora North mine, dyke designs will include the use of overburden, borrow material and tailings sand constructed with either mobile equipment or slurry cell construction. Seven in-pit dykes are planned. The top 10 m of the dykes are constructed with slurry sand cell methods. Syncrude plans to construct overburden dykes at a rate of up to 32.2 million bcm/year and coarse sand dykes up to 10.6 Mm^3/year.

The AERI model utilized a base case scenario which is a clone of the early Syncrude operations. In pit dykes will be constructed with overburden or compacted sand with the annual dyke length built of 0.74 km (8.25 million m^3/year). The AERI model assumed narrow dyke crests, smaller storage volumes and lower construction rates than the operating mine sites. This reflects the fact that the AERI model assumes a mine production rate that is up to 2.5 times lower than the mine sites.

5.5 WASTE DUMPS

Waste from surface oil sand mines include the overburden deposits above the oil sands ore, interbedded soils (i.e. shales) and low grade "lean" oil sands. Morgenstern et al, (1988) report it may be possible to encounter up to 14 or more different types of waste overburden during mining that must be managed and stockpiled. Those materials not meeting stringent construction quality design standards (for dyke structures) must be disposed of in external waste dumps or used a reclamation cover for dewatered tailings deposits. Site constraints such as location of the minable ore body, external impoundments, site infrastructure and lease boundaries will dictate the dump configuration and ultimately the height of the dumps. Additionally, fill and placement methods change depending on availability of source materials, design and operational strategies, and weather during placement (Mckenna, 2002). The design slope of a waste dump ultimately depends on the geotechnical quality of the material used in the dump. Details on the waste dump designs were not included in the Directive 074 tailings plans. In the AERI model, waste dumps were designed with a height of 25 m. The model did not assume a slope and chose to calculate the dump footprint based on a ratio of the mine pit disturbance. To calculate the dump footprint, the AERI model used the following ratio: depth of overburden in-situ (50 m) / the height of pile (25 m) * by the incremental pit area disturbance. In comparison, Mckenna (2002) suggests the waste dumps at Syncrude's Mildred Lake site reach average heights of 40 m and even up to 90 m. The dumps can have overall slopes of 5:1 or flatter. At the Imperial Kearl site, waste dumps are planned to reach 40 to 80 m in height with 4:1 slopes or flatter as mandated by foundation conditions (EUB 2007). Morgenstern et al (1988) report early waste dumps were constructed to heights of 55 m. Waste dump structures are ultimately designed based on the waste material properties, foundation conditions, construction methods employed and geographical constraints.

5.6 CONCLUSIONS

An overview of the historical and planned tailings management plans for several oil sands mine sites was presented. The tailings management plans were influenced by constraints such as lease geometry, geology, quality of the ore body, and extraction process. These constraints were evident in the data provided in the ERCB's Directive 074 annual tailings reports provided by each company. It is apparent from each plan that more than one tailings technology was required to manage the large volumes of tailings arising from the oil sands mining operations. The operators plan to move forward with combinations of NST or CT technologies, coarse sand deposition, fines dewatering techniques including centrifuges, TT, or chemical dewatering with strategic deposition, and final storage of residual MFT in EPLs. Therefore, the final reclaimed landscape for each of the mine sites reviewed will be a combination of terrestrial and aquatic landforms based on the proposed tailings management plans.

5.7 TABLES

Table 5.1 Conventional Tailings Management at Metal and Oil Sands Mines (Modified from CTMC 2012).

Element	Typical metal mines	Typical oil sands mines
Dyke Configuration and Height	Cross-valley dykes, 20 to 200 m high, significant water diversions	Ring dykes, 30 to 100 m high In-pit backfill (geologic containment)
Dyke Construction	Often rock fill, sometimes with tailings sand dykes above a starter dyke	Clay or lean oil sands starter dykes, upstream tailings sand construction, waste over/intraburden
Footprint Area	20 to 250 ha; 125 ha common, often only one deposit	500 to 2500 ha, typically 4 to 10 deposits both in and out of pit
Tailings Slurry Properties	Crushed rock, little to no clay minerals.	Natural sands and clays with residual bitumen. Typically 50% solids slurry with 80% sand, 20% fines, 1% bitumen.
Beaching	Partially segregating: coarse grained near the discharge, and gradually fine down the beach and into the pond.	High degree of segregation: coarse-grained beaches with consistent grain size, clay rich fines in the pond.
Settling / Consolidation Behaviour	Fine tailings generally settle and consolidate within a few years, forming a surface suitable for reclamation.	Self-weight consolidation (no chemical or mechanical assistance) of fine tailings takes decades to centuries
Trafficability	Upper beaches are fully trafficable, lower beaches trafficable with small equipment, fines areas require special soft tailings reclamation techniques (mechanical capping with small equipment)	Beaches above water table trafficable with large equipment, beaches with high water table trafficable with good frost, fine tailings not trafficable.
Tailings Porewater	Often high metal contents, metal leaching, acid mine drainage	Elevated salts and naphthenic acids
Ease of Reclamation	Beaches typically easy to reclaim, but often require engineered covers. Final tailings area expensive but routine to reclaim and limited in aerial extent.	Beaches typically easy to reclaim, soft areas difficult to reclaim, but possible at high cost.

Table 5.2 Summary of Site Specific Conditions and subsequent Implications to Tailings Management.

Factor	Issue	Implications to Tailings Management
Weak Foundation	• Location and configuration of dykes, • Rise rate of dykes	• Footprint of impoundment • Amount of construction material available and rate of availability (Construction rate) • Ultimately will impact capacity (stage curve)
Highly permeable foundation	• Location and configuration of dykes, • Need for engineered containment (liners or interception technology) •	• Footprint of impoundment, • Construction material available for liner • Ultimately will impact capacity (stage curve) • Cost
Construction material	• Availability (volume and timing) • Competition for coarse grained materials	• Amount of construction material available and rate of availability • Construction rate • Tailings technology • Stage curve
Geometry of ore body	• Location of ex-situ dykes, • Timing of in-pit facilities, • Storage efficiency	• Footprint/stage curve • Timing of construction • Stage curve of in-pit structures

Table 5.3 CNRL Mature Tine Tailings Production Assumption.

Time (months)	Solids Content (% by mass)
0	12
30	30
98	35

Table 5.4 Oil Sand Tailings Planning Properties.

Parameter	Shell MRM	Shell JPM	CNRL	Imperial	Syncrude	AERI
Specific Gravity						
Bitumen	-	-	-	1	-	1.01
Water	-	-	1	1	-	1
Coarse	-	-	2.62	2.57	-	2.65
Fines	-	-	2.57	2.62	-	2.65
Dry Density (t/m^3)						
Ore	-	-	-	-	-	2.1
Coarse sand cell	1.69	1.69	-	1.53	-	1.67
Sand beach	1.51	1.51	-	-	1.45	1.57
Whole/primary extraction tailings	1.51	1.51	1.58	1.49	-	1.57
BBW	-	-	-	-	-	1.52
TT	0.85	0.85	-	-	-	-
TSRU	0.93	0.93	-	1.65	-	-
MFT	0.37	0.37	0.37	-	-	0.37
CT on-spec	1.62	-	-	-	1.6	-
CT off-spec	1.55	-	-	-	-	-
NST on-spec	1.62	1.62	1.56	-	-	-
NST off-spec	1.55	1.55	1.58	-	-	-
OB	2.0	2.08	-	-	-	2.0
Fines Capture						
SFR CT on-spec	4	-	-	-	-	-
SFR CT off-spec	(1)	-	-	-	-	-
SFR NST on-spec	6.5	7.5	4 to 5	-	-	-
SFR NST off-spec						
NST on-spec fines capture(2)	80%	80%	95%	-	-	-
NST off-spec fines capture(2)	50%	50%	45%	-	-	-
Whole tailings beach capture(2)	75%	75%	45 to 60%	-	-	19%

(1) Off-spec at Shell means tailings segregate to sand and fines components.

(2) Fines capture means percent of fines from the feed slurry that are captured in the NST deposit.

Table 5.5 Oil Sand Tailings Beaching Parameters.

Parameter	Shell MRM	Shell JPM	CNRL	Imperial	Syncrude	AERI
Beach Below Water Slope %						
Sand beach	7	6.6	-	-	-	4
Whole tailings	7	6.6	2	3 to 6	4	-
TT	3	3	-	-	-	-
TSRU	10	10	-	-	-	-
MFT	0	0	-	-	-	-
CT on-spec	4	0	-	-	-	-
CT off-spec	5	0	-	-	-	-
NST on-spec	4	4	0.7	-	-	-
NST off-spec	5	5	2	-	-	-
Beach Above Water Slope %						
Sand beach	3	2.5	-	-	-	2
Whole tailings	2	1.5	-	1.5	1	-
TT	1	0.5	-	-	-	-
TSRU	0	0.1	-	-	-	-
MFT	0	0	-	-	-	-
CT on-spec	1	0	-	-	0.5	-
CT off-spec	2	0	-	-	-	-
NST on-spec	1	1	-	-	-	-
NST off-spec	2	1.5	-	-	-	-

Table 5.6 Comparison of MFT production rates.

Site	MFT Production		
	m^3 MFT/bbl bitumen	m^3 MFT /tonne ore	m^3 MFT /m^3 ore
CNRL	0.3[(1)]	0.18	-
Shell MRM	0.186	-	-
Syncrude Aurora	0.155	-	-
AERI base case	0.21[(1)]	0.127[(2)]	0.266
Historical Data[(3)]	0.183 – 0.266[(1)]	0.11 – 0.16	-

(1) Assume CNRL conversion of 0.599 barrels bitumen/tonne ore.

(2) Assumed dry density of 2.1 tonne/ m^3

(3) Historical data as reported by CNRL

Table 5.7 Summary of ETF designs.

Design Parameter	Design Footprint (km^2)	Dyke length (km)	Dyke height (m)	Design Fluid Storage (Mm^3)	Design Outer Slopes	Time until in-pit space available (years)
Shell MRM	13.16(1)	12(2)	60(2)	367	4:1 to 12.5:1(1)	6(1)
Shell JPM	4.5	8.6	67	150.8	4:1 to 10:1	6(3)
CNRL	18.83(4)	-	-	775	-	10
Imperial	-	-	35 to 95(5)	950	6:1(5)	6
Syncrude Aurora N.	10(6)	-	55(6) to 75(7)	186(7)	3:1 to 6:1(7)	6-7(6)

(1) Shell 2005

(2) Martens et al. 2008.

(3) Devenney 2010.

(4) Surface area as of 2010 (AESRD, 2013)

(5) EUB (2007) Decision 2007-013.

(6) Purcell et al. 2000.

(7) Reeves 1996.

Table 5.8 Summary of In-pit dykes and disposal areas.

Site	Design Footprint (km^2)	Dyke length (km)	Dyke height (m)	Design Fluid Storage (Mm^3)	Design Slopes	Crest Width (m)
Shell Cell 1 Design	1.8	3.5	60	158	4:1	60 to 100
CNRL	4 to 6	2 to 2.5	-	300 to 630	-	-
Syncrude Aurora	-	2 to 4	40 to 70	220 to 785	4:1 to 7:1	-
AERI Model	2	3	50	73	4:1	20

5.8 FIGURES

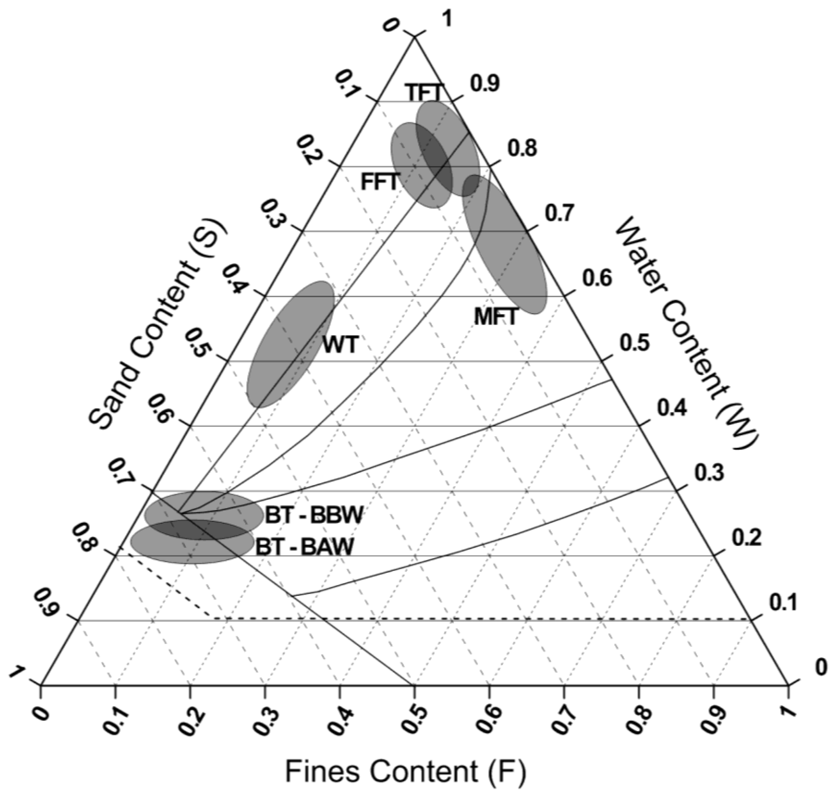


Figure 5.1 Ternary diagram of oil sands tailings streams (modified from Beier and Sego 2013). [WT-whole tailings; FFT-fluid fine tailings; TFT-thin fine tailings; MFT-mature fine tailings; BT-BBW – beach below water tailings; BT-BAW – beach above water tailings]

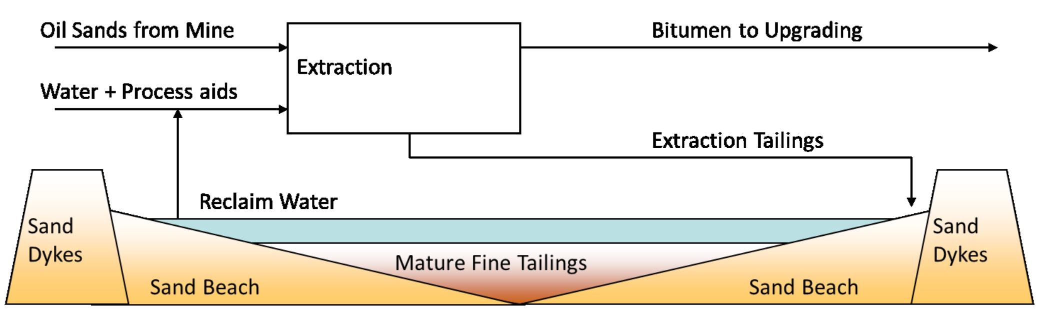


Figure 5.2 Schematic of tailings deposition (modified from Beier and Sego 2007).

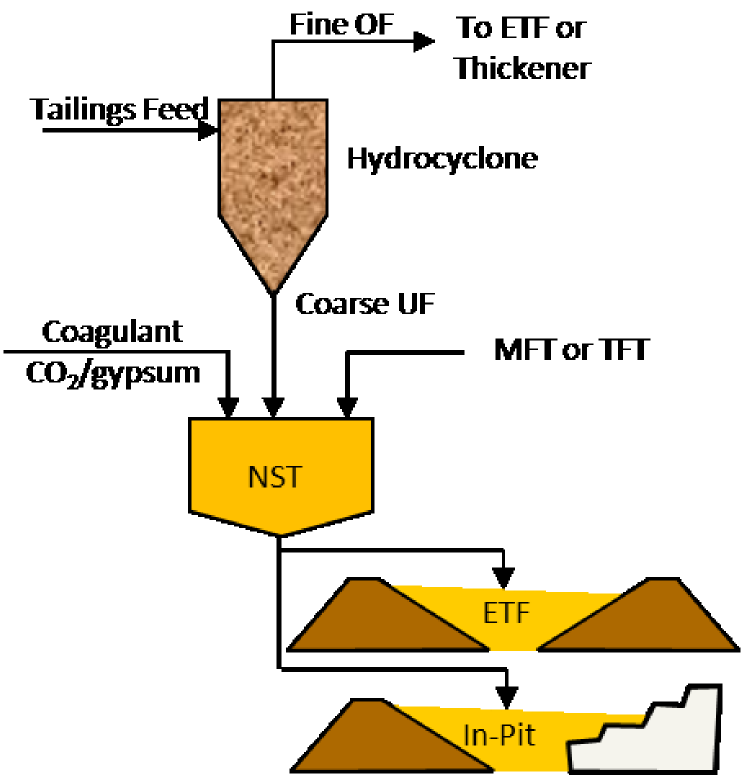


Figure 5.3 Schematic flow diagram of a typical CT/NST process (modified from Matthews et al 2002 and Beier and Sego 2009).

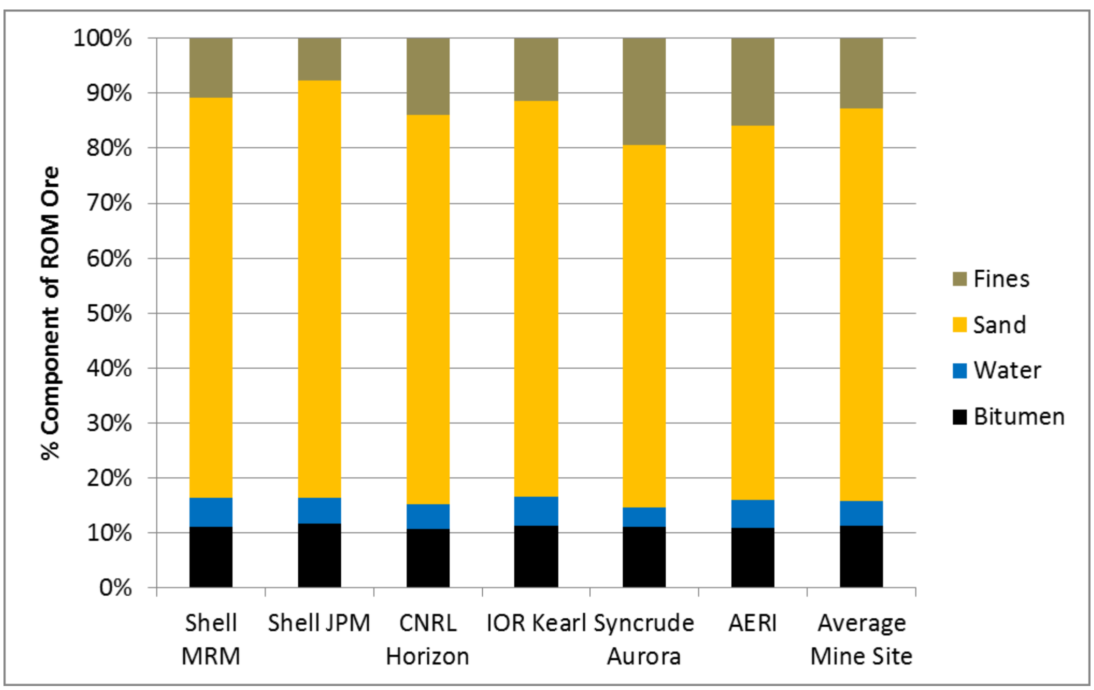


Figure 5.4 Average ore composition by site for years 2011-2029.

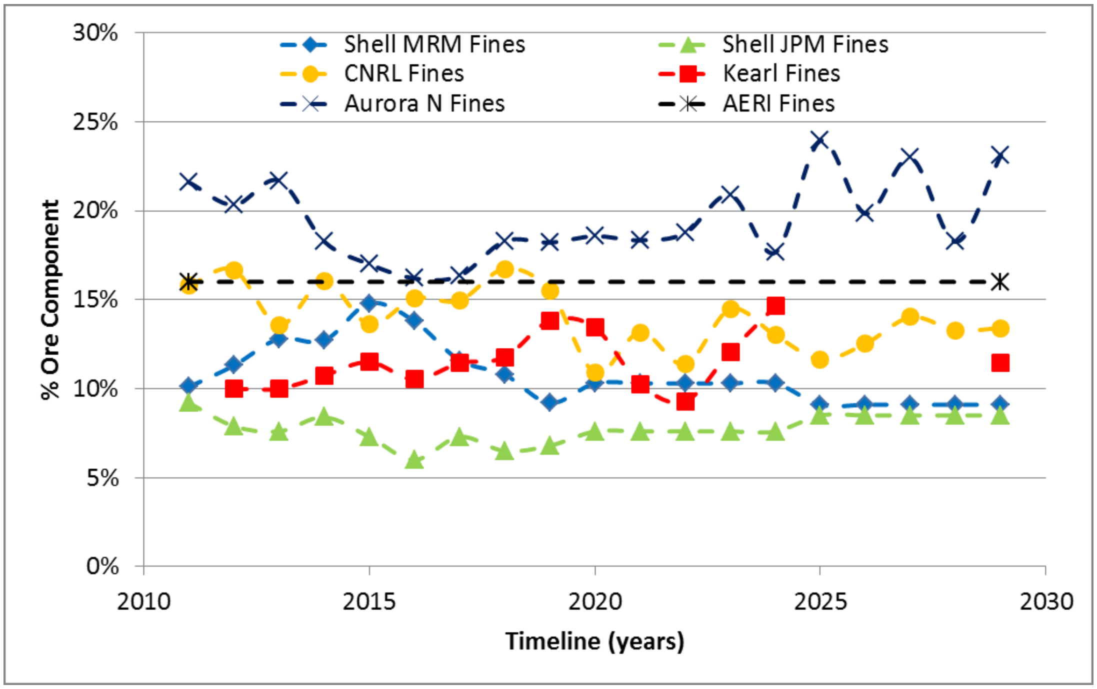


Figure 5.5 Yearly fines content variation at mine sites.

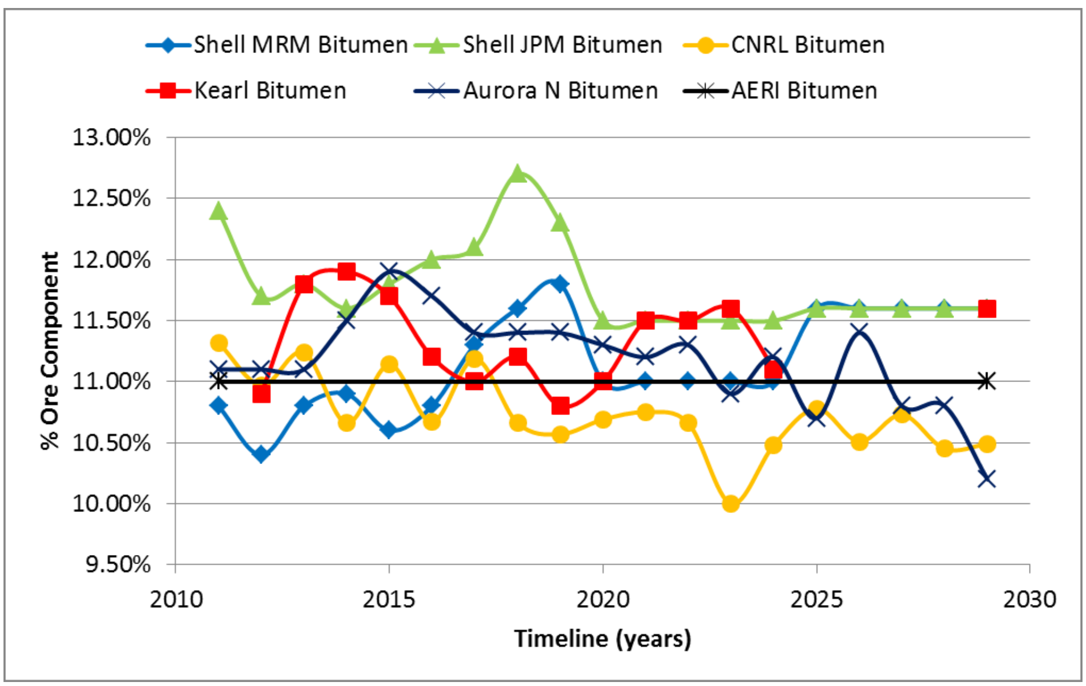


Figure 5.6 Yearly bitumen content variation at mine sites.

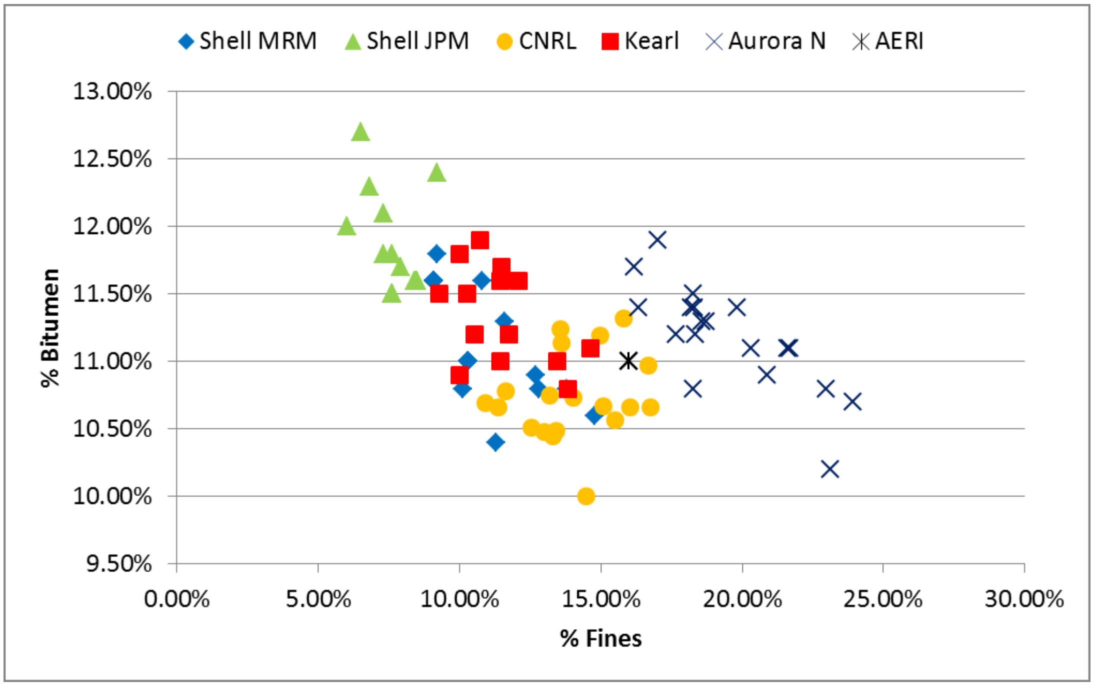


Figure 5.7 Ore bitumen versus ore fines for years 2011-2029.

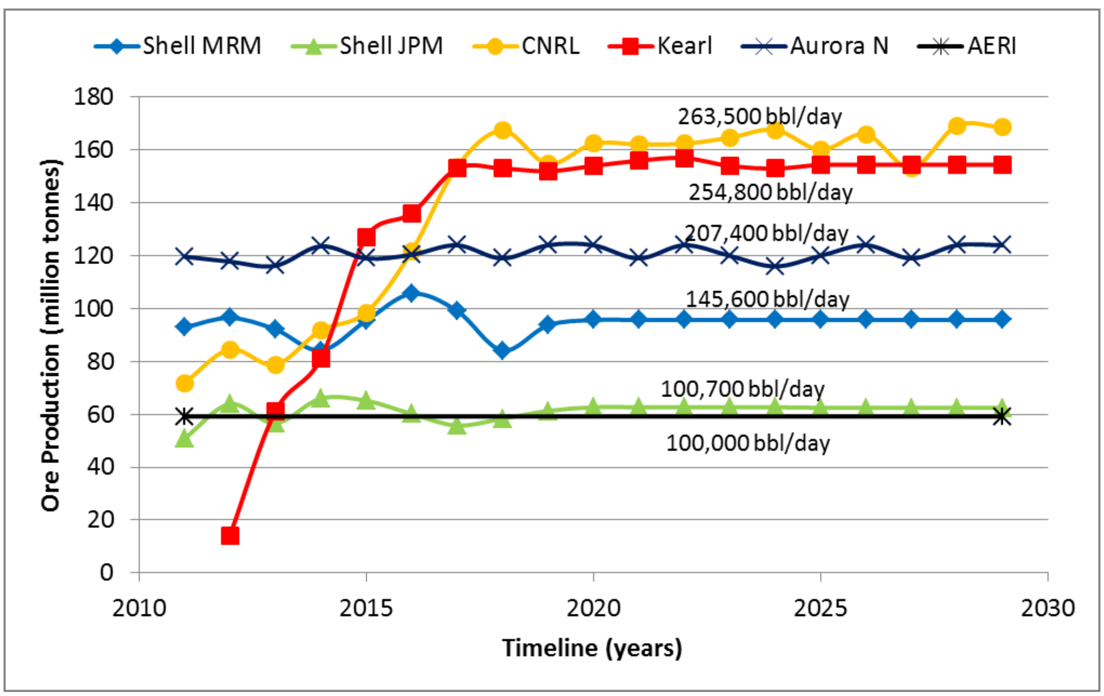


Figure 5.8 Ore Production Schedules for years 2011-2029.

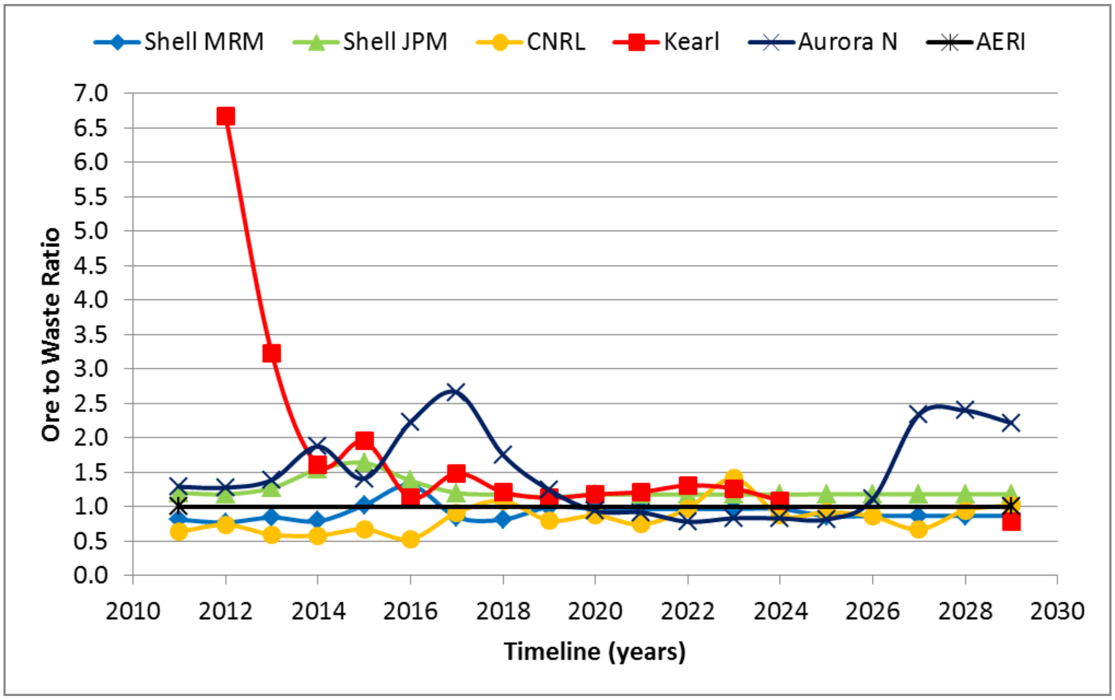


Figure 5.9 Ore to waste ratio for 2011 to 2029.

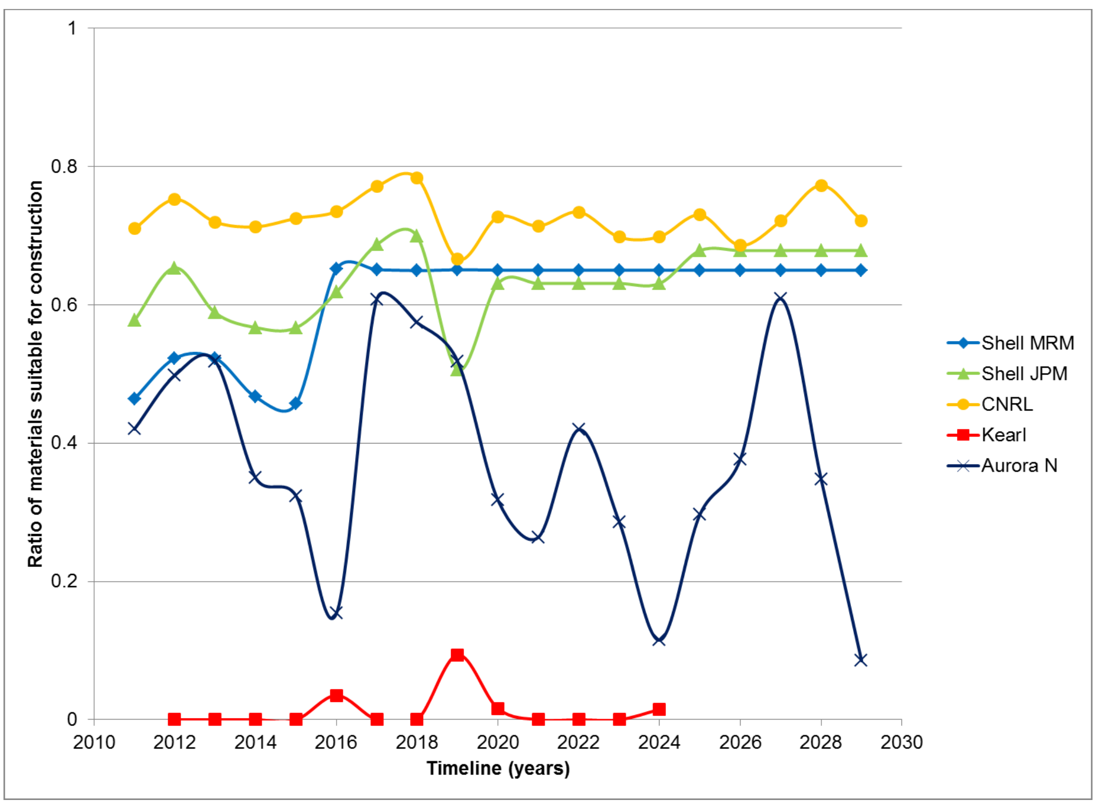


Figure 5.10 Suitability of waste for construction (2011-2029).

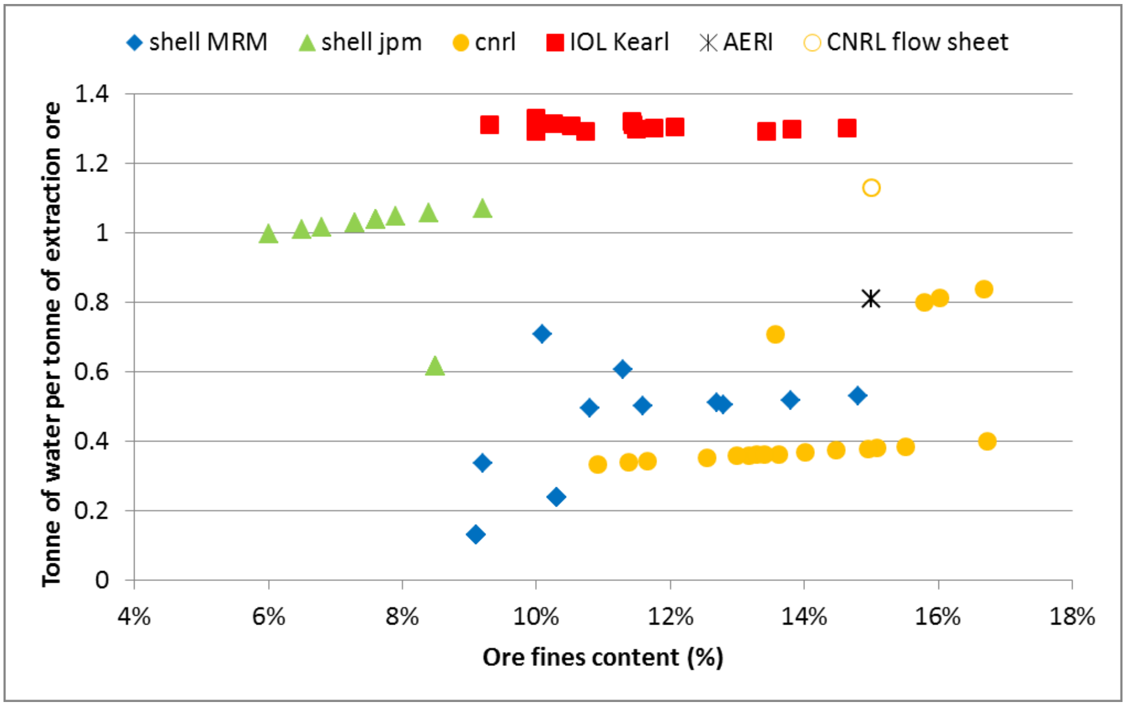


Figure 5.11 Water usage intensity versus fines ore content.

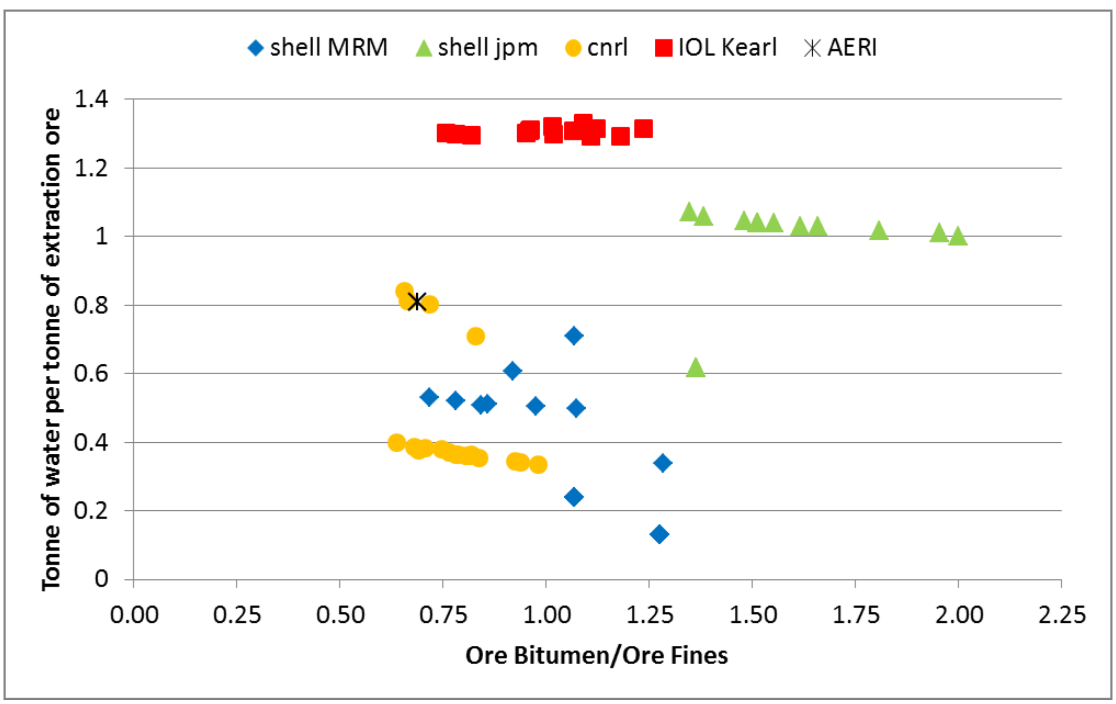


Figure 5.12 Water usage intensity versus ore bitumen/fines ratio.

5.9 REFERENCES

Alberta Energy and Utilities Board (EUB). 2007. Imperial Oil Resources Ventures Limited. Application for an Oil Sands Mine and Bitumen Processing Facility (Kearl Oil Sands Project) in the Fort McMurray Area. Decision 2007-013. 27 February 2007.

Alberta Environment and Sustainable Resource Development (AESRD). 2013. Oil Sands Information Portal. Website accessed May 1, 2013 http://environment.alberta.ca/apps/osip/ .

Azam, S. and Scott, J.D. 2005. Revisiting the Ternary Diagram for Tailings Characterization and Management. Geotechnical News, **23**: December 2005, pp. 43-46.

Beier, N. and Sego, D. 2007. The Oil Sands Tailings Research Facility. 60th Canadian Geotechnical Conference and 8th joint IAH-CNC Groundwater Specialty Conference, Ottawa, Ontario, 22-24 October 2007. 8 pp

Beier, N., Alostaz, M. and Sego, D. 2009. Natural Dewatering Strategies for Oil Sands Fine Tailings. Tailings and Mine Waste 2009 Conference, Banff, Alberta, 1-4 November 2009. Pp. 845-858.

Beier, N.A. and Sego, D.C. 2013. Tailings Management in Canada's Oil Sands. 5th International Young Geotechncial Engineering Conference, Paris, France, 3-6 September 2013, 4 pp.

Boratynec, D.J. 2003. Fundamentals of rapid dewatering of composite tailings. MSc Thesis. Department of Civil and Environmental Engineering, University of Alberta, Edmonton, Alberta. pp. 280.

Canadian Natural Resources Ltd. (CNRL). 2010. Horizon tailings management plan. 13-RPT-TA-0001 Revision 2. November 2010.

Canadian Association of Petroleum Producers (CAPP). 2012. Crude Oil, Forecast, markets and pipelines.. Website accessed Jan. 16, 2013. http://www.capp.ca/forecast/Pages/default.aspx

Chalaturnyk, R.J., Scott, D., and Ozum, B. 2002 Management of oil sands tailings. Petroleum Science and Technology, **20**(9&10):1025-1046.

Chu, A., Paradis, T., Wallwork, V., and Hurdal, J. 2008. Non segregating tailings at the Horizon oil sands project. First International Oil Sands Tailings Conference, Edmonton, Alberta, 7-10 December 2008. 10 pp

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. 5 Volumes. http://www.ai-ees.ca/reports.aspx , accessed May 1, 2013.

Devenny, D.W. 2010. A screening study of oil sand tailings technologies and practices. Report prepared for Alberta Energy Research Institute, March 2010. http://www.ai-ees.ca/media/6924/oil_sand_tailings_technologies_and_practices_a.pdf. [cited 4 April 2013].

Fair A.E. and Beier, N.A. 2012. Keynote Paper: Collaboration in Canada's Oil Sands: Fluid Fine Tailings Management. 3rd International Oil Sands Tailings Conference, Edmonton, Alberta, 3-5 December 2012, pp. 3-12.

FTFC (Fine Tailings Fundamentals Consortium), 1995. Advances in oil sands tailings research, Alberta Department of Energy, Oil Sands and Research Division, Edmonton, Alberta, Canada.

Houlihan, R. and Mian, H. 2008. Past, Present, Future Tailings – Regulatory Perspectives. First International Oil Sands Tailings Conference, Edmonton, Alberta, 7-10 December 2008. 8 pp.

Imperial Oil Resources Ventures Ltd. (Imperial). 2010. 2010 Annual tailings plan submission to ERCB. 30 September 2010.

Kaminsky, H., Omotoso, O., Etsell, T., and Ivey, D. 2009. Distribution of Clay Minerals in the Process Streams Produced by the Extraction of Bitumen from Athabasca Oil Sands. Canadian Journal of Chemical Engineering **87**(2): 85-93.

Martens, S., Lappin, T., and Godwaldt, R. 2008. Compaction of upstream construction tailings beaches using dozers. 61st Canadian Geotechnical Conference, Edmonton, AB. 21-24 Septmeber 2008. Pp. 7.

Masliyah, J., Zhou, Z., Xu, Z., Czarnecki, J. and Hamza, H. 2004. Understanding water-based bitumen extraction from Athabasca oil sands. The Canadian Journal of Chemical Engineering, **82**: 628-654.

Matthews, J. 2008. Past, Present, and Future Tailings – Tailings Experience at Albian Sands Energy. First International Oil Sands Tailings Conference, Edmonton, AB, 7-10 December 2008. Presentation only.

Matthews, J.G., Shaw, W.H., MacKinnon, M.D., and Cuddy, R.G. 2002. Development of composite tailings technology at Syncrude. International Journal of Surface Mining, Reclamation, and Environment, **16**(1): 24-39.

McKenna, G. 2008. Landscape design for oil sands tailings. First International Oil Sands Tailings Conference, December 7-10, 2008, Edmonton, AB, Canada, pp. 51.

McRoberts, E. 2008. Considerations in oil sands tailings containment. First International Oil Sands Tailings Conference, Edmonton, AB, 7-10 December 2008. Pp 13-22.

Morgenstern, N.R., Fair, A.E., and McRoberts, E.C. 1988. Geotechnical Engineering beyond soil mechanics – a case study. Canadian Geotechnical Journal, **25**: 637-661.

Oil Sands Developers Group (OSDG). 2013. Oil sands project list, updated March 2013. Unpublished report. Retrieved from http://www.oilsandsdevelopers.ca/wp-content/uploads/2013/03/Oil-Sands-Project-List-March-5-2013.pdf March 26, 2013.

Purcell, B.C., Barlow, J.P., Nordin, D., and Fleming, S. 2000. Tailings disposal at Syncrude Canada's Ltd Aurora Mine. 6th International Conference on Environmental Issues and Management of Waste in Energy and Mineral Production. Calgary, Alberta. 30 May – 2 June. 2000. Pp. 153-163.

Reeves, B. 1996. Environmental Impact Assessment for the Syncrude Canada Limited Aurora Mine. BOVAR Environmental, 1996.

Shaw, M. 2008. Past, Present, and Future Tailings Management – Suncor Energy. First International Oil Sands Tailings Conference, Edmonton, AB, 7-10 December 2008. Presentation only.

Shell Canada Energy (Shell). 2005. Application for approval of the Muskeg River Mine Expansion project. Volume 1: Project Description. 28 April 2005.

Shell Canada Energy (Shell). 2010a. Muskeg River Mine – Annual tailings management plan 2011. Directive 074 – Appendix E requirement. Submitted to ERCB. 29 September 2010.

Shell Canada Energy (Shell). 2010b. Jackpine Mine ERCB Directive 074 - Tailings management plan. Submitted to ERCB. 3 December 2010.

Shell Canada Energy (Shell). 2010c. Muskeg River Mine – 2010 Annual fluid pond status report and compliance report (D74-Appendix D and F). 29 September 2010.

Sobkowicz, J.C. and Morgenstern, N.R. 2009. A geotechnical perspective on oil sands tailings. 13th International Conference on Tailings and Mine Waste, Banff, Alberta, 1-4 November 2009. University of Alberta, pp. xvii-xli.

Syncrude Canada Ltd. (Syncrude). 2010a. 2010 Annual tailings plan submission Syncrude Aurora North (Leases 10, 12 and 34). 30 September 2010.

Syncrude Canada Ltd. (Syncrude). 2010b. 2010 Annual tailings plan submission Syncrude Aurora South (Lease 31). 30 September 2010.

Syncrude Canada Ltd. (Syncrude). 2010c. Directive 074 Baseline survey for fluid deposits Syncrude Mildred Lake and Aurora North. 30 September 2010.

Wells, P.S. and Riley, D.A. 2007. MFT drying – case study in the use of rheology modification and dewatering of fine tailings through thin lift deposition in the oil sands of Alberta. Paste 2007, Tenth International Seminar on Paste and Thickened Tailings, Perth, Australia, 13-15 March 2007. Pp. 271-284.

6 GEOTECHNICAL ASPECTS OF FLOCCULATION-BASED TECHNOLOGIES FOR DEWATERING MATURE FINE TAILINGS

6.1 INTRODUCTION

Mining and extraction of oil sands to produce bitumen has been underway in north eastern Alberta for the past five decades. The Canadian oil sands deposit contains an estimated 170 billion barrels of recoverable bitumen. Four operating oil sands companies (Suncor Energy, Syncrude Canada Ltd., Canadian Natural, and Albian Sands) are currently producing bitumen at approximately 890,000 bbls/day with several new mines and expansions planned that would double production by 2025 (CAPP 2012). At current and predicted production rates, oil sands exploitation will continue long into this century. A typical oil sand ore deposit is comprised of bitumen (~12 weight % by mass), sand, silts, clays (mineral content ~85 % by mass) and water (3 to 6 % by mass).

The production of bitumen from the oil sands ore body is based on open-pit mining and a water based extraction process. Warm to hot water, steam and process aids such as sodium hydroxide or calcium citrate are used to extract the bitumen from the mineral matrix (Masliyah et al. 2004). The extraction process recovers 90 to 92% of the bitumen and produces a tailings stream (known as whole tailings) consisting of water, sand, silt, clay and residual bitumen, at high rates of 12,000 to 30,000 tonnes/hr (Sobkowicz and Morgenstern 2009). Typical material compositions for whole tailings are provided in Table 6.1 (Chalaturnyk et al. 2002; Sobkowicz and Morgenstern 2009). In the oil sands industry fines are defined as material passing 45 μm and coarse is the material greater than 45 μm. It is noted that the solids content is the dry mass of solids (mineral solids and residual bitumen) divided by the total mass of the tailings (mineral solids, residual bitumen and fluid); sand content is the dry mass of sand divided by the dry mass of solids; fines content is the dry mass of fines (silts and clays plus residual bitumen) divided by total dry mass of solids; and clay content is the dry mass of clay divided by the dry mass of fines (Boratynec 2003).

Whole tailings are either discharged directly into a storage facility (i.e. external tailings facility) or classified through a cyclone separator and thickener before deposition (Sobkowicz and Morgenstern 2009). In the storage facility, whole tailings deposited onto beaches or into constructed cells undergo segregation. The segregated sand fraction forms beaches or is compacted in the cells to form the containment dyke. Sand classified from the whole tailings using a cyclone separator can also be deposited in, or used for the construction of the storage cells. Fines are trapped within the sand matrix of the beaches during deposition ($\geq$ 50 % of the total mass of fines), while the remaining fines and water flow into the settling pond at a solids content of 6 to 10% and are referred to as thin fine tailings (Chalaturnyk et al. 2002). Overflow from the cyclone classifier, consisting of mainly fines and residual bitumen, is further dewatered in thickeners and then deposited into the storage facility. The fines slowly settle in the storage facility over a few years to a solids content of 30 to 35 % (water content, w, of 186 to 233 %) and are referred to as MFT. Further settlement and consolidation of the MFT is extremely slow (in the order of decades) due to the dispersed nature of clay fraction as a result of the extraction process (Chalaturnyk et al. 2002; Jeeravipoolvarn et al. 2009; Suthaker and Scott 1997).

On average, approximately 0.266 m^3 of MFT and 0.91 m^3 of sand are produced for every 1 m^3 of mined ore (Devenny, 2010). To date, there is an estimated 850 million m^3 of MFT stored among the operating mine sites (Fair and Beier, 2012). In addition to managing large volumes of fluid fine tailings, water management is an issue for the mine sites. The industry operates under a zero-effluent discharge policy. No process affected water may be released from site and must be contained. Therefore, the oil sands region is dominated by a wet landscape with several, large above grade containment structures storing fluid tailings and process water.

The oil sands industry has developed methods aimed to reduce the inventory of MFT and create dry stable landscapes. For example, the process of mixing sand with the MFT, termed consolidated or composite tailings, CT, requires a mixture of segregated sand from a cyclone underflow, MFT and a coagulant such as gypsum. Ideally, CT is mixed at a gravimetric SFR of approximately 4:1 and is not expected to segregate during transport, discharge or deposition. It was anticipated that the resulting CT deposits would reach a geotechnically stable state in a timely manner so terrestrial reclamation could proceed (Sobkowicz and Morgenstern 2009). However, operational challenges have hindered the commercial full-scale success of CT. The CT operations had to compete for sand that is also used to provide economical containment dykes. When sand was utilized for CT, the depositional techniques employed (high shear upon deposition) and/or limited control of the slurry density during production of CT (leading to low SFR mixtures) contributed to segregation upon discharge. The resulting deposits of low SFR materials have proven to be difficult to reclaim (Hyndman and Sobkowicz 2010). For example, Suncor's Pond 5 is one of the first full scale trials of the CT technology with deposition occurring between 1995 - 2008. However, the resulting deposit was too weak to support terrestrial reclamation. Variation in the feed tailings streams and difficult process/depositional controls lead to segregation of the CT during deposition and the formation of weak deposits. In the case of Pond 5, the weak tailings deposits were capped with geotextile and petroleum coke to allow for access and installation of vertical wick drains. The wick drains are intended to promote dewatering so the underlying tailings can develop sufficient strength to support reclamation activities (Abusaid et al. 2011). The surface area of Pond 5 is a few square km, therefore the capping and wick drain installation represents a considerable unforeseen expense. Since the 1980s, technical advances have been made 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.

6.1.1 Oil Sands Tailings Regulations

The ERCB, Alberta's regulatory body responsible for the oil sands industry, has been concerned about the continual accumulation of fine tailings and the associated risks to reclamation activities. As such, the ERCB elected to regulate fluid fine tailings through performance criterion, and in early 2009 issued Directive 074: Tailings Performance Criteria and Requirements for Oil Sands Mining Schemes. The aim of the directive is to reduce fluid tailings accumulation by capturing the fines in dedicated disposal areas, DDAs and create trafficable surfaces for progressive reclamation. The ERCB indicates a trafficable surface must have a minimum S_u of 10 kPa. The Directive requires operators to submit tailings plans, tailings pond status reports, disposal area plans and compliance reports. Compliance with Directive 074 can be directly measured through specified strength performance in the tailings deposits. The Directive requires a minimum S_u of 5 kPa for tailings material deposited in the previous year. If any material fails to meet the 5 kPa requirement, it must be removed or remediated. Additionally, five years after active deposition, the deposit must be trafficable (i.e develop an S_u of at least 10 kPa) and ready for reclamation. It is evident that new technologies and processes must be developed to supplement current tailings management plans, specifically, additional fines-management techniques. While specifying a short-term strength requirement is consistent with the overall goal of accelerated reduction in fluid fines inventory by the oil sands industry, compliance with the directive presents some challenges from a geotechnical standpoint. The following discussion highlights some of the geotechnical issues and challenges associated with flocculation-based technologies for dewatering MFT.

6.2 FINE TAILINGS MANAGEMENT

Implementation of the ERCB's Directive 074 has driven industry to review current tailings management techniques and investigate the numerous alternative technologies and processes to manage and reclaim fine tailings. Essentially, there are three general methods to incorporate the problematic, clay-dominant fine tailings into a closure landscape. The fines can be sequestered into the coarse tailings matrix as CT, placed in an engineered mine pit under a water cap to form a lake or dewatered separately creating a cohesive, silty-clay deposit. The industry is trending toward managing the fine stream separately using chemical additives in addition to physical and environmental dewatering techniques in order to meet the Directive 074 performance criteria.

In addition to satisfying Directive 074 requirements, the chosen tailings management process should be consistent with both operational and reclamation/closure goals (Hyndman and Sobkowicz 2010). As the mining operations proceed, tailings should be reclaimed progressively, recognizing a portion of the fine tailings will be incorporated into water capped lakes within an engineered pit (OSTC 2012). In doing so, this will limit the accumulation of fluid fine tailings that would require out-of-pit containment and remediation at the end of mine life. Fluid containment structures should be limited to a minimum (i.e. only what is required for effective tailings management). Meeting these operational goals would allow the operator to proceed with reclamation and return the mine site back to the public, thereby achieving the existing reclamation goals. Essentially, operators would be able to avoid tailings ponds/dams in the closure landscape that would require ongoing maintenance (in the order of decades). Also, operators would be able to transform the tailings deposits into geotechnically-stable landforms or functional aquatic ecosystems (in-pit lakes) that are resistant to natural processes and are self-sustaining both physically and environmentally, ensuring that these landforms and features are successfully integrated into the future natural ecosystem.

6.2.1 Fines Dewatering Methods Technologies

Physical/mechanical methods for dewatering oil sands fine tailings include centrifugation and thickening. In centrifugation operations, MFT is dredged from a tailings pond, diluted and then mixed with a polyacrylamide flocculant. The flocculated MFT stream is then processed in the centrifuge and dewatered to a C_w of 55% (w of 82%) prior to deposition. Where thickeners are utilized, the fines stream for thickeners would come directly from the extraction process as cyclone overflow, COF, rather than from the MFT pond. These sand-depleted tailings streams or COF would be flocculated and dewatered in the thickeners prior to deposition at solids contents of up to 40% (w of 150%) and referred to as TT (Hyndman and Sobkowicz 2010), or up to 60% (w of 67%) and referred to as paste tailings, PT (Masala and Matthews 2010; Yuan and Lahaie 2009).

Alternatively, the fines may be dewatered through a combination of chemical addition and strategic deposition. Polymer solutions are injected directly into the transfer pipeline containing dredged MFT. This process has been termed "in-line flocculation" or ILTT. Two depositional techniques are available for the flocculated fine material. The flocculated mixture can be discharged onto a gently-sloped beach in thin layers at a solids content of 30 to 35% (w of 186 to 233 %, Kolstad et al. 2012). Substantial instantaneous dewatering then occurs as a result of aggregation of fines into flocs which enhances the saturated hydraulic conductivity and water release properties of MFT. Subsequently, additional dewatering is achieved via a combination of settlement, seepage and environmental dewatering (desiccation and freeze/thaw). This technology has been commercially implemented at Suncor as part of their Tailings Reduction Operations (Wells et al. 2011) and a commercial-scale demonstration (referred to as AFD) is currently underway at the Shell Muskeg River Mine Site (Kolstad et al. 2012).

The flocculated tailings may also be discharged into large depositional cells (>10 m deep) to promote self-weight consolidation and environmental dewatering via evaporation. As water is released to the surface, active water management is required by means of decant structures and mechanical channeling (perimeter ditching) to promote further dewatering and development of strength. This depositional technique is referred to as "rim-ditching" or "accelerated dewatering" and has been piloted at the Syncrude mining operation. Research and development of this technology is ongoing (OSTC 2012; Sobkowicz 2010).

Each of these fines management techniques involves some form of polymer or chemical addition to promote dewatering and strength gain. However, it is important to note that these processed fines are typically produced and deposited at water contents above their natural liquid limit, w_L, with reliance on initial water release, consolidation and environmental dewatering for densification and shear strength gain.

6.3 GEOTECHNICAL PROPERTIES OF FLOCCULATED FINE TAILINGS

6.3.1 Data Source

Available data from public literature and field data provided from the Shell Muskeg River Mine AFD operations was compiled to illustrate the impact of flocculation on the geotechnical behaviour of oil sands fine tailings. The public data sources are summarized in Table 6.2. Banas (1991), Jeeravipoolvarn (2010), Sorta and Sego (2010) and Suthaker and Scott (1997) analyzed grab samples harvested from a tailings pond or tailings process line while Miller et al. (2011*a*) and Miller et al. (2011*b*) generated tailings samples by processing 80 tonne ore samples with various extraction and deposition methods. The above authors used the vane shear test and/or cavity expansion method to measure S_u in their fine tailings samples using the equipment and procedures developed and described in detail by Suthaker and Scott (1997) and Banas (1991). Only Atterberg Limit data was available from Yuan and Lahaie's (2009) small scale pilot test on thickened oil sands tailings. Though Masala and Matthews (2010) did not specifically describe the methods used to determine the S_u values for their PT and MFT deposits, it was learnt that field vane shear test was used (A. Dunmola, personal communications, 2013).

Shell Canada Energy also provided S_u data from a multi-year commercial-scale field demonstration of an in-line flocculation process named AFD. Kolstad et al. (2012) provides an overview of the AFD process. A field vane shear test was used to measure the *in-situ* peak and residual stress from the surficial AFD deposits (< 1m thick). Grab samples were also collected to characterize the deposit.

6.3.2 Material Properties

Oil sand fine tailings and MFT are comprised of fine solids passing the 44 μm sieve including silts, clays (>50%), and residual bitumen. Clays that are commonly found in MFT include kaolinite (50 to 60%) and illite (30 to 50%) with some mixed layer smectites (Chalaturnyk et al. 2002, FTFC, 1995, and Kaminski et al., 2009). The activity of flocculated fine tailings (Shell 2011 & 2010 AFD) ranges from 0.9 to 1.2 which coincides with the major clays (kaolinite and illite) commonly found in MFT. Typical Atterberg limits for a range of MFT and flocculated fine tailings samples originating from the Syncrude and Shell mine sites are shown in Figure 6.1. The w_L of MFT and fine tailings range from 40 to 65%, but can be up to 100% when treated with chemical amendments such as flocculants. The plasticity index, I_p, ranged from 20 to 30% for MFT samples and up to 73% with the addition of flocculants. Based on the index testing, MFT and fine tailings can be classified as medium to high plasticity inorganic clays.

6.3.3 Undrained Shear Strength of Oil Sand Tailings

To meet the regulatory performance criteria for fine-grained oil sands tailings set forth by the ERCB, it is necessary to remove sufficient water from the fine tailings to increase solids content (density) and consequently increasing strength. For reference, Hyndman and Sobkowicz (2010) and Sobkowicz and Morgenstern (2009) provide a description of the geotechnical and material property transformations as oil sands tailings undergo physical and natural dewatering processes. The strength of the fine tailings may also be enhanced through chemical addition. Fine tailings management techniques discussed previously involve some form of polymer or chemical addition to promote dewatering and strength gain. To illustrate the impact of flocculant addition on the strength gain in oil sands fine tailings, S_u data from untreated MFT deposits (Shell and Syncrude MFT) and chemically amended fine tailings (Syncrude Sheared ILTT and Shell 2011 AFD) is compared in Figure 6.2. MFT exhibits very low S_u (in the range of pascals) at a solids content range of 30 to 35%. Therefore, significant amounts of water must be removed before the material would meet the performance criterion of 5 kPa (to a solids content greater than 70%). However, the addition of flocculants can enhance the shear strength at lower solids contents. The Syncrude ILTT material developed an S_u of 5 kPa at approximately 60 %

solids content while the Shell 2011 AFD material could achieve the same S_u at solids contents as low as 50%.

In addition to solids content (density), the grain size distribution, clay content and mineralogy, and water chemistry will also impact the shear strength of tailings (Mitchell and Soga 2005; Sobkowicz and Morgenstern 2009). Therefore, in order to normalize these influences, Liquidity Index, I_L, may be used instead of solids content. The relationship between undrained remolded shear strength, S_{uR}, and I_L for typical MFT from the Shell and Syncrude mining operations is provided in Figure 6.3. Additionally, Figure 6.3 contains data for natural clay deposits as reported in Locat and Demers (1988) and Mitchell and Soga (2005). As can be seen, the relationship (Equation 6.1) proposed by Locat and Demers (1988) provides a good overall fit for typical MFT. Therefore, for ease of comparison, typical MFT is represented using Equation 6.1 in Figure 6.4 and Figure 6.5.

[6.1] $$S_{uR} = \left(\frac{19.8}{I_L}\right)^{2.44}$$

It is known that MFT exhibits a thixotropic behaviour. This behaviour is illustrated in Figure 6.4. Data from Miller et al. (2011*a*) and Miller et al. (2011*b*) on MFT S_u gain after 365 days at rest (displayed as "Miller 365 days peak") and combined data sets from Banas (1991) and Suthaker and Scott (1997) on strength gain after 680 days at rest (displayed as "Banas 680 day peak") is compared to the S_{uR} of typical MFT in Figure 6.4. Upon shearing of this thixotropic material (displayed as "Banas 680 day residual"), the S_u collapses to the S_{uR} line for typical MFT (open box symbols).

Directive 074 requires S_u values in the range of 5 to 10 kPa or greater. For typical MFT (Figure 6.4), this would require dewatering from w of 233% (I_L of ~6.5) to below its w_L (I_L of ~ 0.6) following the Locat and Demers (1988) line. Summaries of S_u versus I_L for fluid fine tailings treated using various dewatering techniques (ILTT and PT) as reported in the literature (Jeeravipoolvarn 2010; Masala and Matthews 2010) are provided in Figure 6.5. The data in Figure 6.5 suggest it is possible to achieve the required S_u value of 5 to 10 kPa with polymer addition to fluid fine tailings. The S_u versus I_L data from Shell's AFD field deposits from 2010 and 2011 have also been plotted similarly in Figure 6.6 and Figure 6.7, respectively. Although there is significant scatter in Shell's AFD field data (Figure 6.6 and Figure 6.7), it is evident the AFD process in 2010 did not achieve an S_u of 5 kPa. In 2011, only a portion of the AFD material reached an S_u of greater than 5 kPa at the time of sampling (a few months after deposition). Further improvements to the AFD process and/or longer dewatering rates may be necessary.

6.3.4 Shear Strength Sensitivity

The S_u sensitivity of the chemically-amended fine tailings deposits is also of interest. Sensitivity, S_t, the ratio of the S_u to undrained residual shear strength, S_{ur}, is calculated using Equation 6.2,

$$[6.2] \qquad S_t = \frac{S_u}{S_{ur}}$$

In Figure 6.5, various values of sensitivity are represented by lines based on Equation 6.2 and assuming the S_{ur} is equivalent to the S_{uR} calculated by Locat and Demers (1988) using Equation 6.1. From Figure 6.5, it can be deduced that chemically-modified fine tailings deposits (PT and ILTT) may exhibit shear-sensitive behaviour based on their reported strengths. Deposits with S_t up to 9 were reported at I_L greater than 1. Due to the scatter in the AFD's S_u versus I_L data in Figure 6.6 and Figure 6.7, a plot of S_u versus S_{ur} was created to assess the sensitivity of the AFD material (Figure 6.8). The 2010 AFD deposit does not exhibit the same sensitivity as the PT or ILTT materials since the sensitivity was typically below 2. However, the 2011 AFD deposit did exhibit sensitivity greater than 8 for approximately 10 % of the samples, with the bulk of the samples exhibiting sensitivities from 2 to 4. The S_t data for the 2011 AFD material was also plotted against solids content in Figure 6.9 and I_L in Figure 6.10. As the 2011 AFD material is dewatered (increase in solids content and decrease in I_L), S_t is observed to generally decrease to values below 5.

6.3.5 Consolidation of Flocculated Tailings

The consolidation behavior of the fine tailings is also influenced by chemical amendments as illustrated on Figure 6.11. The compressibility behaviour of fresh fine tailings (Miller et al. 2011b), COF (Jeeravipoolvarn 2010) and remoulded Albian MFT (Zhang and Sego 2012) are representative of "young" or sheared tailings samples without any chemical treatment. With time, untreated MFT was observed to exhibit an over-consolidation stress of approximately 0.5-2 kPa due to thixotropic strength gain (Jeeravipoolvarn et al. 2009 [10 m MFT]; and Miller et al. 2011b [Sycnrude MFT]). There is also evidence that flocculation will contribute to an apparent over-consolidation. A small over-consolidation stress was observed for an ILTT sample that was created by flocculating a COF (Jeeravipoolvarn 2010). Masala et al. (2012) also found that flocculating fine TSRU tailings resulted in the development of an over-consolidation stress of about 80 kPa. After remoulding the TSRU sample and breakage of the flocculated structure, the TSRU tailings exhibited a virgin compression curve.

6.4 IMPLICATIONS OF FINES MANAGEMENT TECHNIQUES USING FLOCCULANTS

The ability to meet deposit performance targets using flocculent-based processes relies heavily on the ability to match the required flocculent dosage, mixing time and energy, and deposition with the feed tailings characteristics. According to Shell (A Dunmola, personal communication 2012), differences in shear strength behaviour between Shell's AFD deposit in 2010 and 2011 (Figure 6.6, Figure 6.7, and Figure 6.8) are attributed to improvements and optimization of the AFD flocculation process and deposition methods for the 2011 season. However, detailed operational data of the flocculation process for the AFD 2010 and 2011 programs were not made available. The variation in feed tailings may also have contributed to the different performances because the tailings used in the 2010 AFD process had greater LL and PI than the 2011 AFD tailings. The reader is also reminded that the AFD data presented in this paper were from sampling and testing the deposits after just a few months of drying (summer to early fall).

Fine tailings management techniques based on flocculation typically produced tailings deposits discharged at water contents above their w_L with reliance on initial water release, consolidation and environmental dewatering for densification and shear strength gain. Consequently, significant amounts of water may still be bound within the deposits even at the S_u target for regulatory compliance (5 to 10 kPa). For example, untreated MFT can achieve an S_u of 5 kPa at a solids content of 73% or I_L of 0.6 (Figure 6.2 and Figure 6.3). In contrast, for Shell's AFD deposits, an S_u of 5 kPa can be achieved at solids contents as low as 50% (I_L of 1.75, Figure 6.2 and Figure 6.7). Therefore, for every tonne of dry solids, an AFD deposit can hold up to 0.63 m^3 more water than untreated MFT at the same S_u. However, the oil sands industry has had limited success implementing MFT dewatering processes that achieve the necessary strength targets without flocculent amendments. Freeze - thaw dewatering is a potential "flocculent-free" technology that has been shown to meet strength and dewatering objectives at both the laboratory and pilot scale, but has not been rigorously tested or implemented at the commercial scale (Proskin et al. 2010; Proskin 2012 and Zhang and Sego 2012). For mining operations that have limited lease space, implementing flocculent based dewatering technologies may lead to operational challenges associated with managing larger active deposits.

Available public data and field data provided by Shell Canada Energy also show that flocculated fines tailings deposits may exhibit high S_u sensitivity (Figure 6.5 and Figure 6.8). Mitchell and Soga (2005) classified deposits with S_t of 4 to 8 as "very sensitive" and S_t of 8 to 16 as "slightly quick clays". Based on this classification, a portion of the 2011 Shell AFD deposit with solids contents below 50% (Figure 6.9) could be classified as "slightly quick clay". However, as the solids content of the 2011 AFD material increased, the S_t decreased to below 5. It is recognized that the products from these flocculation-based dewatering technologies may create potentially metastable and liquefiable deposits if left unmitigated. Such unmitigated deposits could mean significant future containment is required even though the deposits meet the regulatory performance criteria.

Due to the bound water and potentially metastable state, oil sands operators adopting flocculation-based dewatering technologies typically target processing such deposits to significantly higher S_u (in excess of the regulatory requirements) or provide sufficient containment so they can transform the tailings deposits into geotechnically-stable landforms. The final disposal options for these materials can include re-handling and disposal of the partially-dewatered material within containment cells, co-mixing with overburden material, load surcharging within engineered overburden structures or leaving the material in place as part of a multi-layer deposit (OSTC 2012). Materials left in place must rely on further consolidation and environmental dewatering such as desiccation and freeze-thaw dewatering for densification and shear strength gain. The target end point for these deposits may require the material to dewater to near the I_P (S_u of 100 to 200 kPa, Hyndman and Sobkowicz 2010). Such requirement for additional time for dewatering may necessitate larger depositional footprints or reduced throughput (OSTC 2012).

Shell expects co-mixing with overburden material and/or load surcharging within engineered overburden structures will addresses the issue of potential instability of the flocculated MFT deposits in addition to satisfying the requirements of the ERCB's Directive 074 (A. Dunmola personal communication 2012). According to Shell's tailings plan for MRM (Shell, 2010), between 5.7 and 6.8 million m^3/year of MFT will be consumed in the AFD process over the next 7 years. Assuming a S_u of 5 kPa is achieved (I_L of 1.75), the corresponding volumes of AFD material ranges from 3.0 to 3.6 million m^3/year. If all of this material were mixed with or stored within overburden (material not suitable for construction) the yearly ratios of AFD to overburden would be 1:5 to 1:10 (Table 6.3). Elkateb (2003) evaluated the engineering behavior of co-disposed embankments of thickened tailings (flocculated fine tailings) and dense sand at mix ratios of 1:10. The stability and displacements were found to be sensitive to the S_u of the fine tailings and the embankment shape. Therefore, DDA designs needs to consider the implications of incorporating high water content flocculated fines on the overall stability.

An alternative disposal scenario for flocculated fines tailings involves continuous deposition into deep containment cells or DDAs. Dewatering and corresponding gain in strength thus rely on self-weight consolidation and drainage. Flocculent addition to the fine tailings stream may actually hinder the self-weight consolidation process. Mesri (1975) states the S_u of sedimentary clays is proportional to a pre-consolidation pressure (σ'_p). Flocculent addition was shown to increase the S_u of fine tailings (Figure 6.1 and Figure 6.5) which could lead to the development of σ'_p. Available compressibility data on flocculated oil sands fines tailings does in fact indicate that flocculation will contribute to an apparent over-consolidation (Jeeravipoolvarn 2010; and Masala et al. 2012). Several relationships between S_u and σ'_p have been proposed based on laboratory data and field data. Schiffman et al. (1988) propose the ratio as a function of I_P (Equation 6.3).

[6.3] $$\frac{S_u}{\sigma'_p} = 0.11 + 0.0037 \times I_p$$

Mesri (1989) contests the ratio is equivalent to 0.22 for all soft clays. However, Masala and Matthews (2010) present a range of 0.17 to 0.33 for Shell TT (flocculated and thickened fine tailings) based on laboratory measurements and 0.35 based on field measurements. Using equation 6.3, the Shell AFD 2010 and 2011 materials would have an average S_u/σ'_p = 0.32 and 0.27, respectively. For the ILTT materials tested by Jeeravipoolvarn (2010), the S_u/σ'_p ratio was found to increase linearly with void ratio (above void ratios of 1.5). Miller et al. (2011a) used a S_u/σ'_p ratio of 0.22 to calculate the σ'_p of 2.3 kPa for Syncrude MFT, based on the measured S_u. The calculated σ'_p corresponds well with the compressibility data for Syncrude MFT on Figure 6.11, where a σ'_p can be seen at approximately 2 kPa.

The implication of developing a σ'_p due to flocculation can be illustrated through the following theoretical scenario (Figure 6.12). Flocculated fine tailings (I_P = 50, average from Figure 6.1) are intermittently poured into a large, deep containment cell. Development of strength is quite rapid due to the flocculation process and optimal deposition therefore a S_u of 2 kPa can be achieved within two days following deposition (Kolstad, 2012). Using equation 6.3, the corresponding σ'_p for the flocculated fine tailings is calculated as 6.7 kPa. Therefore, the fine tailings deposits must have a surcharge loading of greater than 6.7 kPa for significant consolidation to occur. An equivalent surcharge loading can be calculated by assuming a submerged unit weight (γ') for the fine tailings. From Figure 6.6 and Figure 6.7, at S_u between 2 and 5 kPa, the I_L for flocculated fine tailings can range from 1 to at least 2. At these water contents and assuming a specific gravity of 2.5 (Jeeravipoolvarn 2010), the γ' can range from 4.2 to 8.2 kN/m^3. Therefore, the equivalent surcharge will range from 0.8 to 1.6 m, depending on the density of deposited tailings. If the time between layer placements were such that the deposit reached an S_u of 5 kPa before the next layer was applied, a surcharge load of 2 to 4 m would be required to induce consolidation.

6.5 CONCLUSIONS

Bitumen has been extracted from the oil sands deposits in northern Alberta for several decades. Although technological advances have improved mining and extraction efficiencies, the industry still faces challenges in finding practical methods to control and reduce the formation of fluid fine tailings or MFT. It was shown that these deposits of "MFT" behave like natural clay slurries and can be represented by Locat and Demers' (1988) I_L versus remolded S_u relationship. In response to the ERCB's Directive 074 in 2009, the oil sands industry has undertaken considerable research and development with polymer-based flocculation to augment dewatering and strength gain of the fine tailings stream. However, the use of chemical amendments may present some challenges that extend beyond compliance with Directive 074.

The data presented in Figure 6.5 and Figure 6.7 demonstrate that it is possible to meet regulatory requirements (S_u of 5 kPa) with polymer addition to fluid fine tailings. However, chemically-amended fine tailings can have lower storage efficiencies compared to untreated tailings if the latter can be dewatered to the same target S_u. For mining operations that have limited lease space, implementing flocculent based dewatering technologies may lead to operational challenges associated with managing larger active deposits. Additionally, available data suggests that these deposits may exhibit sensitive, metastable behavior upon deposition. To address this challenge, oil sands operators implement mitigative measures to ensure the stability of such flocculated MFT deposits, such as mixing or containing the flocculated fines within overburden structures. These mitigation measures require double handling of material, increasing the overall cost of the process. Based on Shell's tailings plan for MRM (Shell, 2010), mix ratios of flocculated fines from AFD to overburden would be 1:5 to 1:10. Final deposit designs needs to consider the implications of incorporating high water content flocculated fines on the overall stability. Based on available data and published literature, it was found that flocculent addition to the fine tailings stream may actually hinder the self-weight consolidation process through the development of a pre-consolidation pressure. For disposal scenarios that rely on drainage and self-weight consolidation (multi-layer deposits), several layers of tailings may be required before a sufficient surcharge load will induce compression and dewatering of the underlying tailings deposit. Therefore, improvement in current understanding of the storage and DDA design implications, sensitivity and long-term geotechnical behaviour of flocculated dewatered fine tailings deposits is required.

6.6 TABLES

Table 6.1 Oil Sands Tailings Properties.

Parameter	Whole Tailings (Typical values)	Mature fine tailings (Typical values)
Solids Content (%)	55	30-35
Sand content (% by dry mass)	82	<5
Fines content (% by total dry mass including bitumen)	17	>95
Clay content (% by dry mass of fines)		30-50

Sand (>45 μm)

Fines (<45 μm)

Clay (<2 μm)

Table 6.2 Public Sources of Oil Sands Tailings Data.

Data	Source
10 m MFT	Jeeravipoolvarn et al. (2009)
Albian MFT	Zhang and Sego (2012)
Fresh Fine Tailings	Miller et al. (2011a,b)
Shell MFT	Masala and Matthews (2010) and Sorta and Sego (2010)
Shell PT	Masala and Matthews (2010)
Syncrude COF	Jeeravipoolvarn (2010)
Syncrude ILTT	Jeeravipoolvarn (2010)
Syncrude MFT	Banas (1991), Miller et al. (2011a,b), Sorta and Sego (2010) and Suthaker and Scott (1997)
Syncrude TT	Yuan and Lahaie (2009)
TSRU and TSRU Remoulded	Masala et al. (2012)

Table 6.3 AFD deposit volumes and overburden mix ratios.

Year	MFT Consumed in AFD (Mm^3)	Volume of AFD at 5 kPa (Mm^3)	Overburden to dump (Mm^3)	Ratio (AFD fines to OB)
2013	5.9	3.1	25.6	1:8.2
2014	5.9	3.1	25.5	1:8.2
2015	6.8	3.6	22.7	1:6.3
2016	6.8	3.6	17.7	1:4.9
2017	6.8	3.6	36.2	1:10.1
2018	6.8	3.6	23.7	1:6.6
2019	5.7	3.0	20.3	1:6.7

6.7 FIGURES

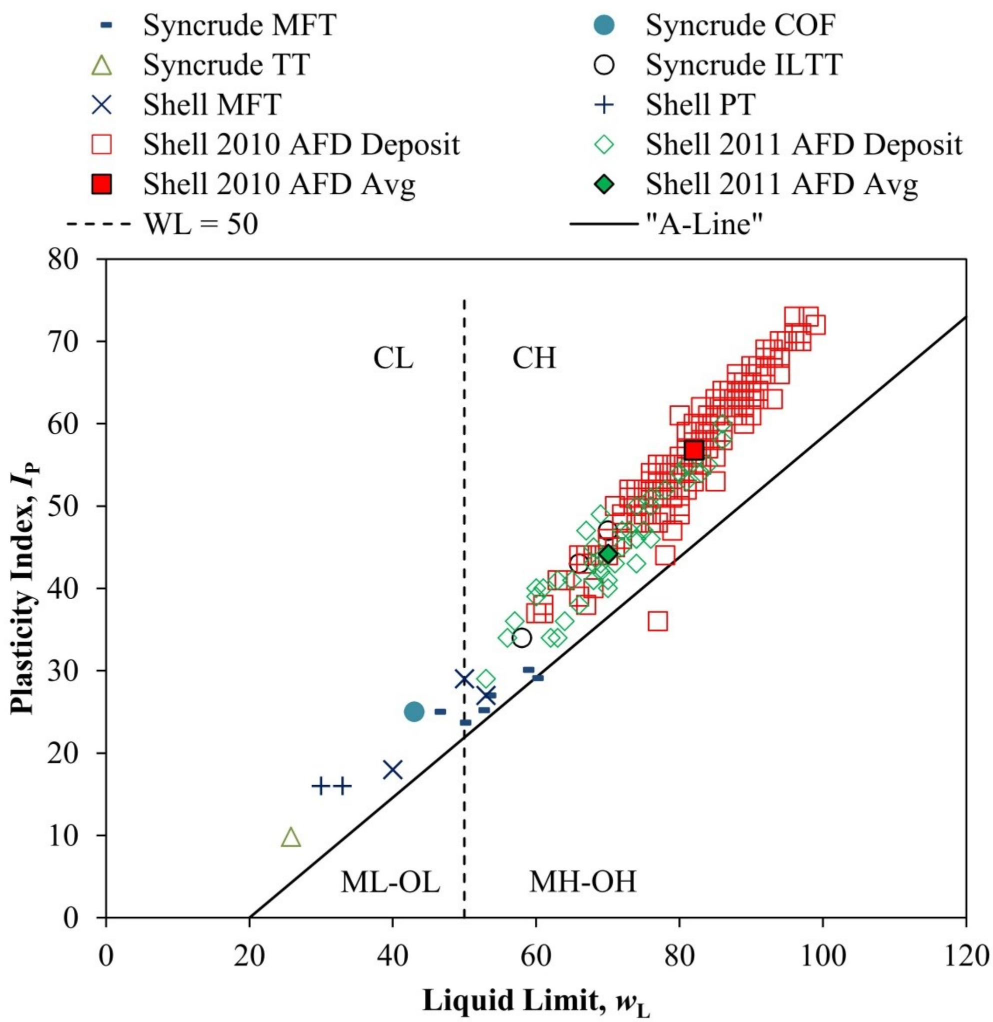


Figure 6.1 Plasticity chart of oil sands fine tailings. [non-flocculated tailings include Syncrude MFT, Syncrude COF and Shell MFT; all other tailings are flocculated]

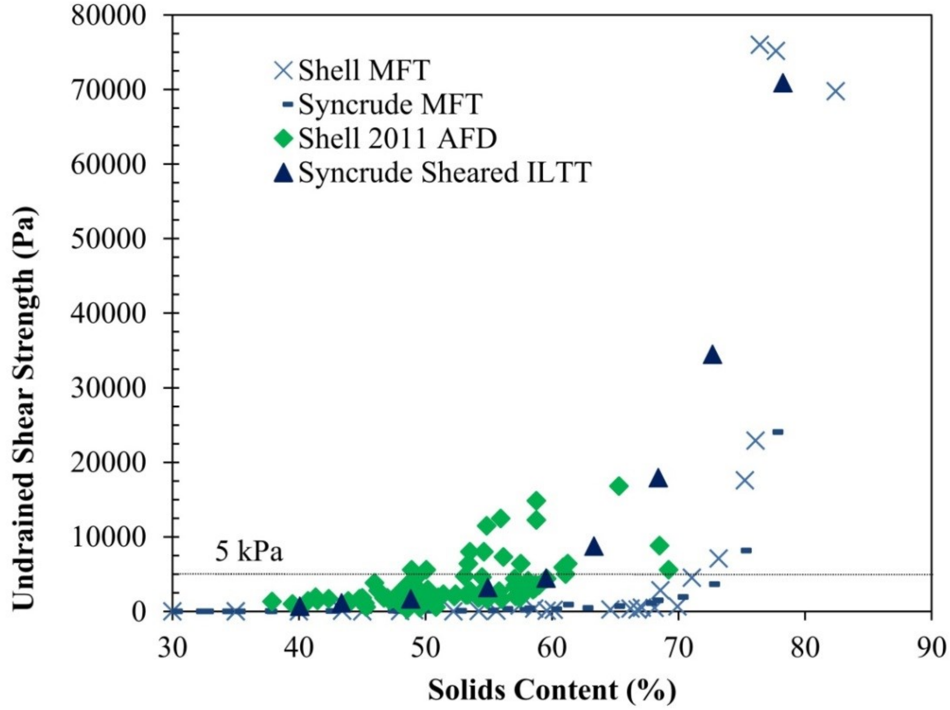


Figure 6.2 S_u of oil sands tailings as a function of solids content.

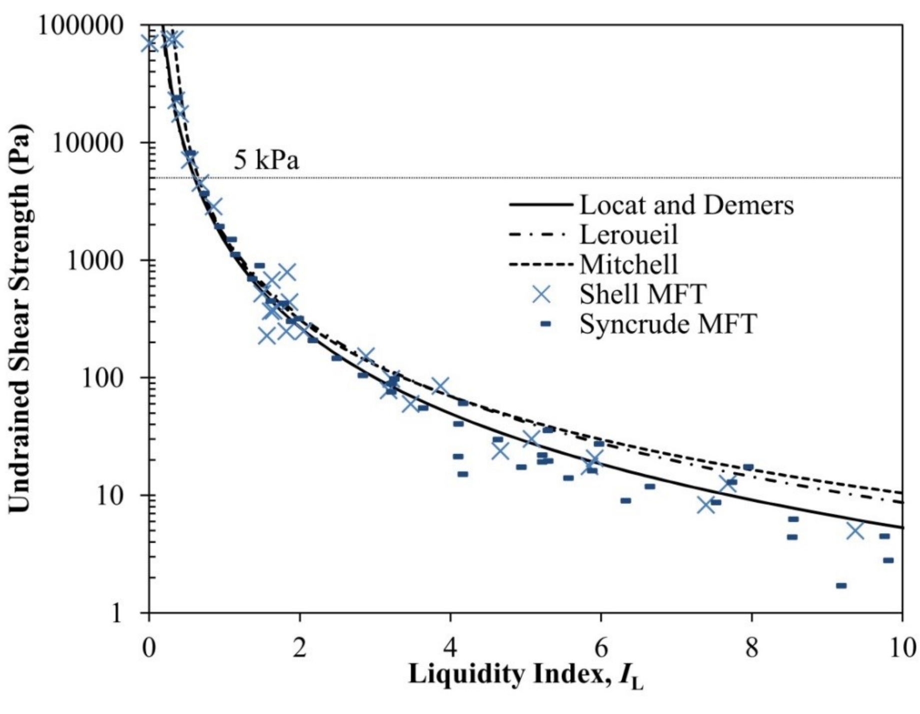


Figure 6.3 Remolded S_u of oil sands mature fine tailings.

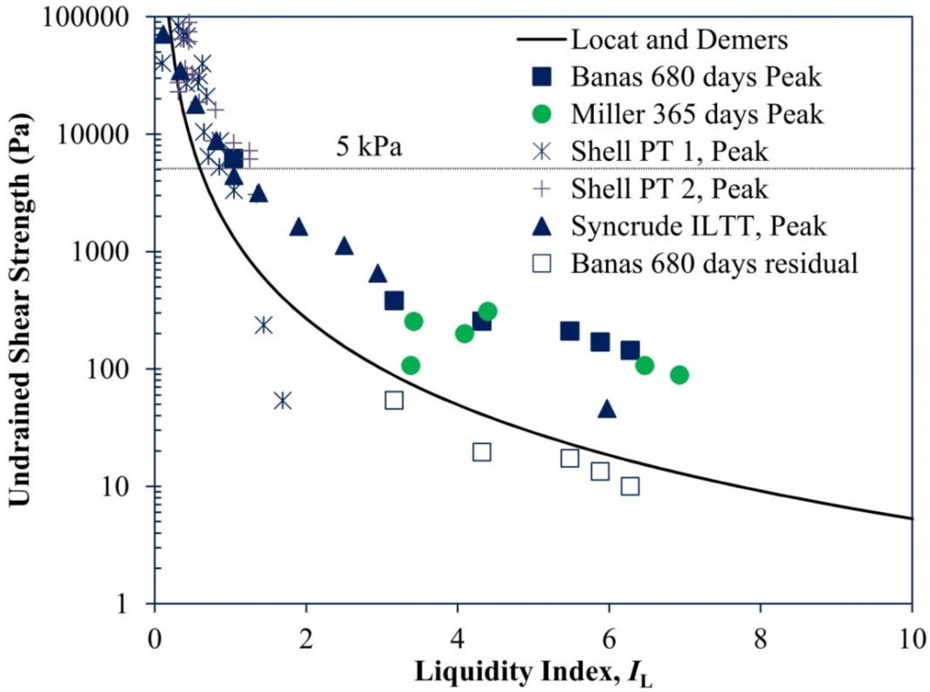


Figure 6.4 S_u of oil sands fine tailings.

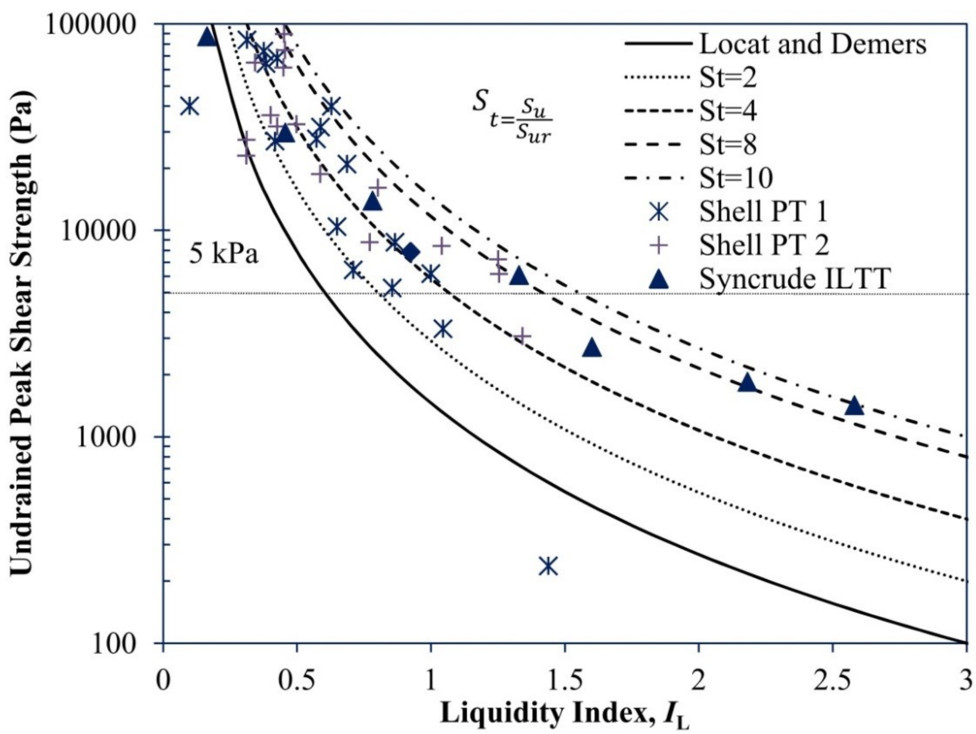


Figure 6.5 S_u sensitivity of oil sands fine tailings.

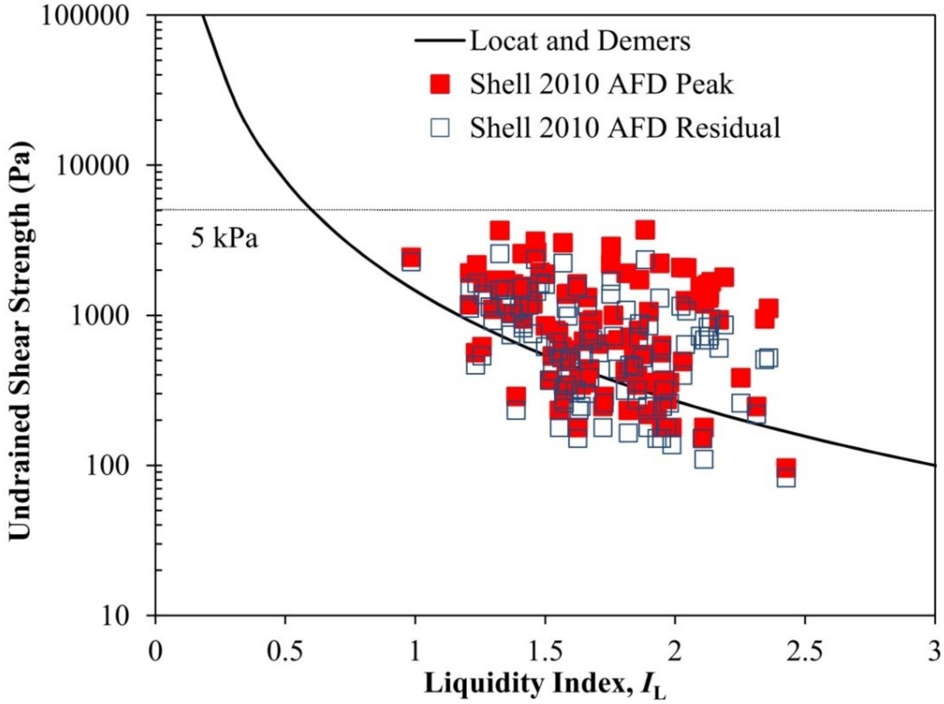


Figure 6.6 S_u of Shell's 2010 AFD deposit.

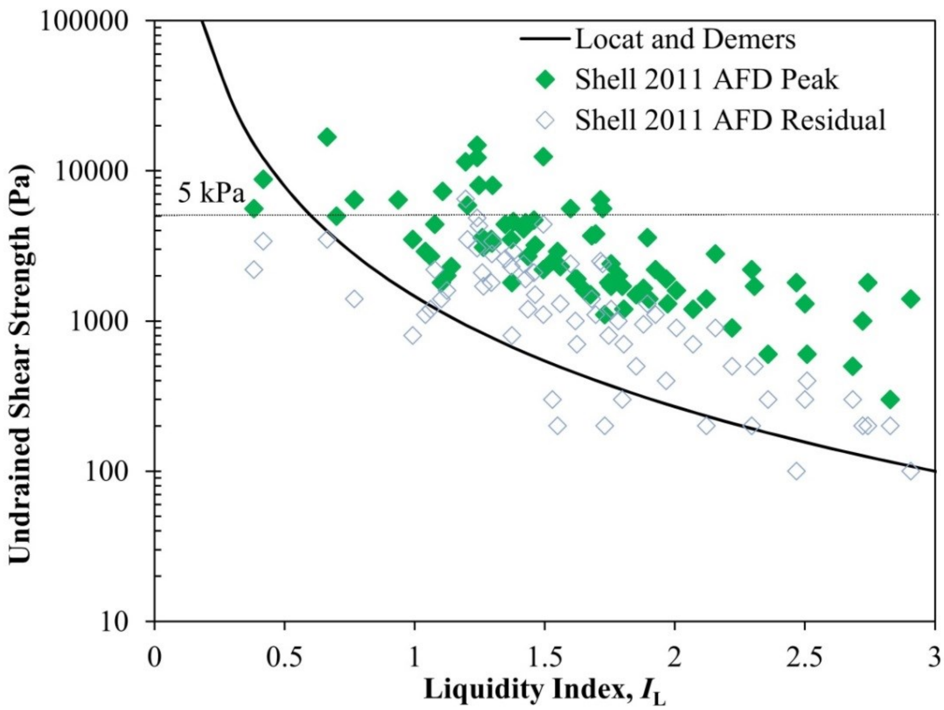


Figure 6.7 S_u of Shell's 2011 AFD deposit.

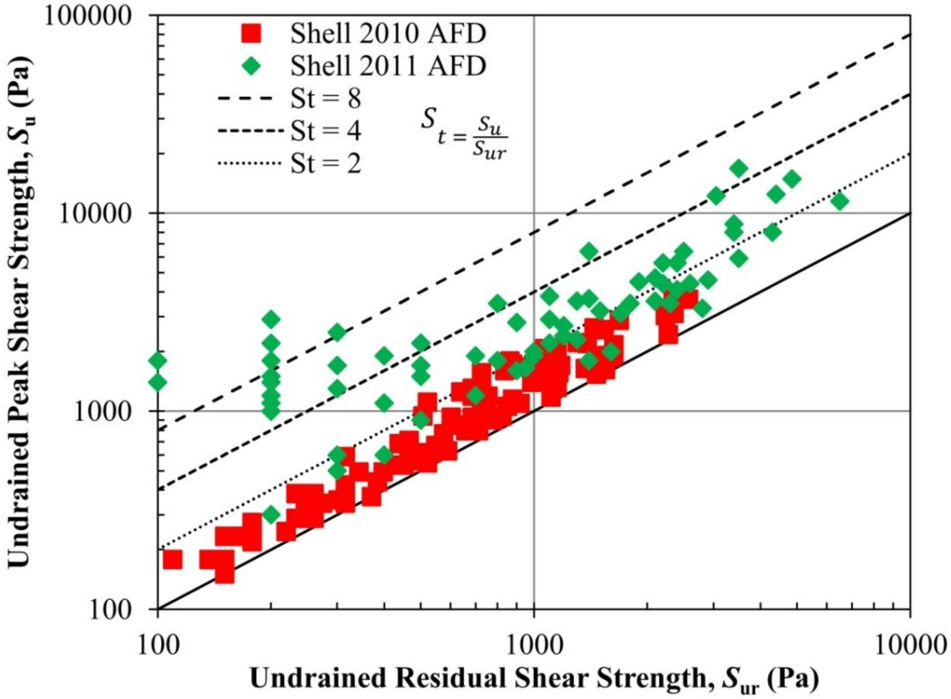


Figure 6.8 S_u sensitivity of Shell's AFD deposits.

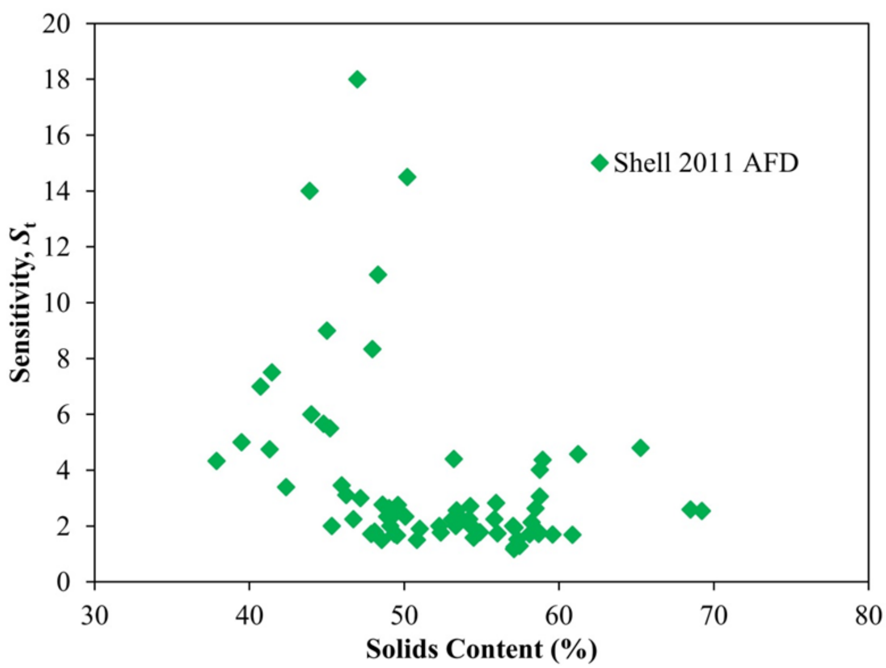


Figure 6.9 S_u sensitivity of Shell's AFD deposits as a function of solids content.

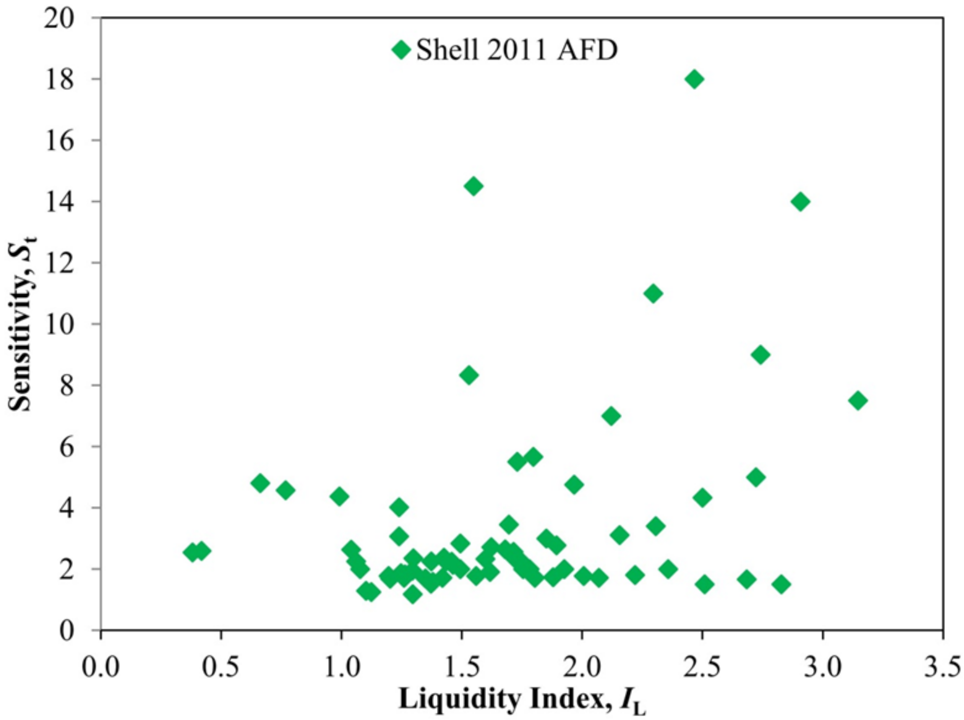


Figure 6.10 S_u sensitivity of Shell's AFD deposits as a function of Liquidity Index.

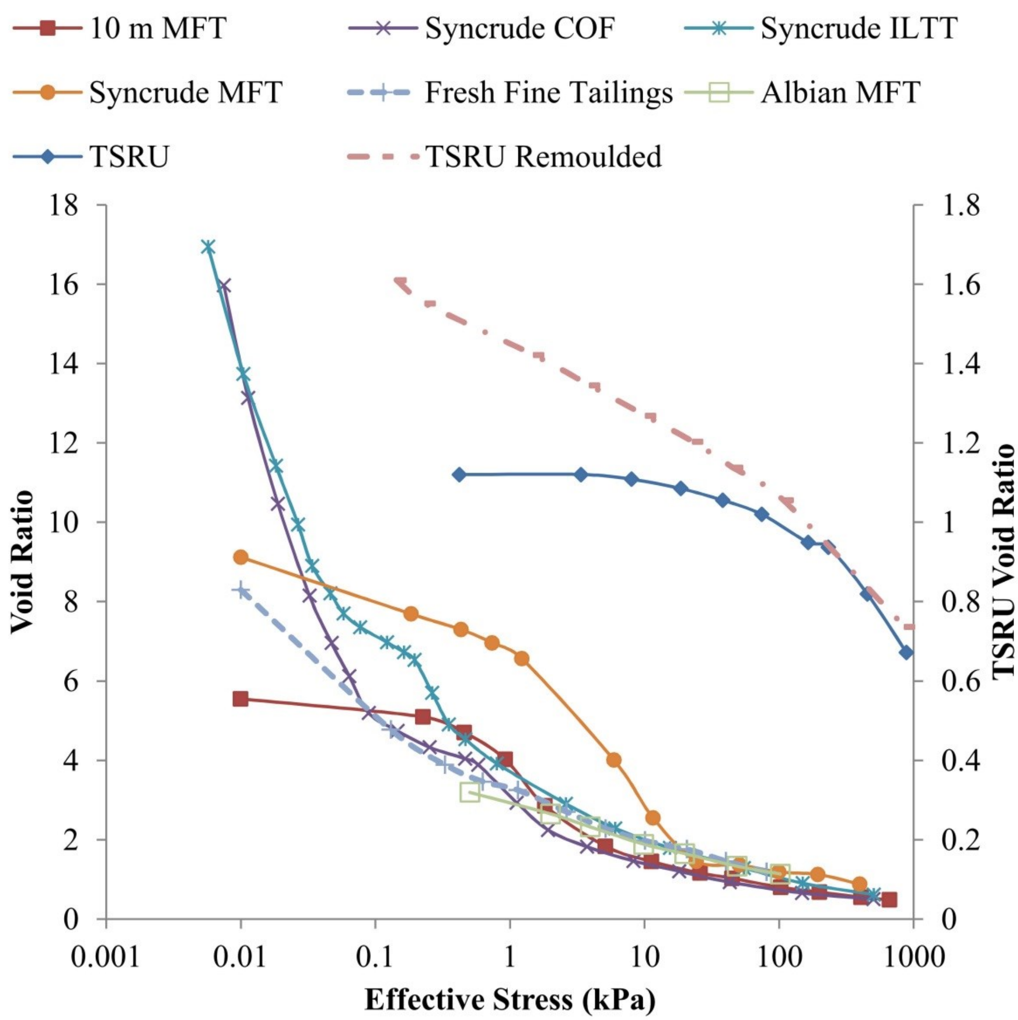


Figure 6.11 Compressibility of various fine tailings.

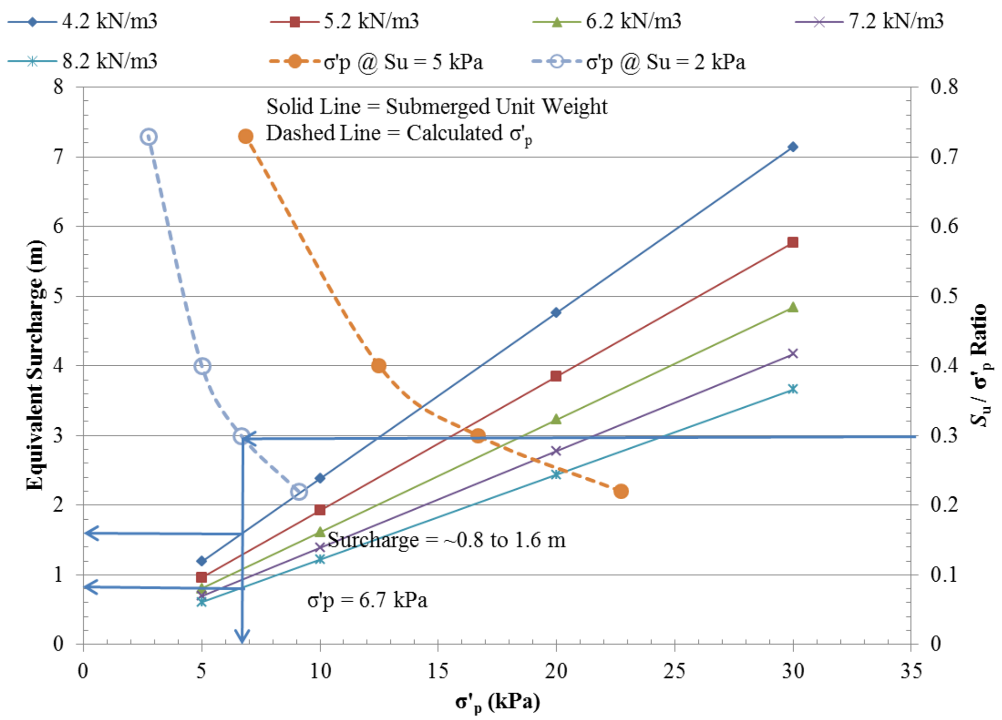


Figure 6.12. Equivalent surcharge loading for fine tailings with a pre-consolidation pressure.

6.8 REFERENCES

Abusaid, A., Fear, C., McRoberts, E., and Wells, S. 2011. An update to the construction of the Suncor oil sands tailings Pond 5 cover. *In* Proceedings of the 15th Tailings and Mine Waste Conference, Vancouver, British Columbia, 6-9 November 2011. pp. 363-375.

Banas, L. 1991. Thixotropic behavior of oil sands tailings sludge. M.Sc. thesis, Department of Civil Engineering, University of Alberta, Edmonton, Alberta. pp. 394.

Boratynec, D.J. 2003. Fundamentals of rapid dewatering of composite tailings. MSc Thesis. Department of Civil & Environmental Engineering, University of Alberta, Edmonton, Alberta. pp. 280.

Canadian Association of Petroleum Producers (CAPP). 2012. Crude Oil, Forecast, markets and pipelines. http://www.capp.ca/forecast/Pages/default.aspx . Accessed Jan. 16, 2013.

Chalaturnyk, R.J., Scott, D., and Ozum, B. 2002. Management of oil sands tailings. Petroleum Science and Technology, **20**(9&10): 1025-1046.

Devenny, D.W. 2010. A screening study of oil sand tailings technologies and practices. Report prepared for Alberta Energy Research Institute, March 2010. http://www.ai-ees.ca/media/6924/oil_sand_tailings_technologies_and_practices_a.pdf. [cited 4 April 2013].

Dunmola, A. 2012. Personal Communication.

Elkateb, T. 2003. Quantification of soil heterogeneity. PhD Thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, Alberta.

Fair A.E. and Beier, N.A. 2012. Keynote Paper: Collaboration in Canada's Oil Sands: Fluid Fine Tailings Management. *In* Proceedings of the 3rd

International Oil Sands Tailings Conference, Edmonton, Alberta, 3-5 December 2012, pp. 3-12.

FTFC. 1995. Clark hot water extraction for fine tailings. *In* Advances in oil sands tailings research. Fine Tailings Fundamentals Consortium (FTFC), Oil Sands and Research Division, Alberta Department of Energy, Edmonton, Canada. **Vol 1**. pp. 79-84.

Hyndman, A. and Sobkowicz, J. 2010. Oil sands tailings: Reclamation goals and the state of technology. *In* Proceedings of the 63rd Canadian Geotechnical Conference, Calgary, Alberta, 12-16 September 2010. Canadian Geotechnical Society, pp. 642 – 655.

Jeeravipoolvarn, S. 2010. Geotechnical behavior of in-line thickened oil sands tailings. PhD Thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, Alberta. pp. 431.

Jeeravipoolvarn, S., Scott, J.D., and Chalaturnyk, R.J. 2009. 10 m standpipe tests on oil sands tailings: long-term experimental results and prediction. Canadian Geotechnical Journal, **46**(8): 875-888.

Kaminsky, H., Omotoso, O., Etsell, T., and Ivey, D. 2009. Distribution of Clay Minerals in the Process Streams Produced by the Extraction of Bitumen From Athabasca Oil Sands. Canadian Journal of Chemical Engineering **87**(2): 85-93.

Kolstad, D., Dunmola, A., Dhadli, N., O'Kane, M., Song, J. and Masala, S. 2012. Towards the improvement in geotechnical performance of atmospheric fines drying (AFD) deposits at Shell's Muskeg River Mine. *In* Proceedings of the 3rd International Oil Sands Tailings Conference, Edmonton, Alberta, 3-5 December 2012, pp. 411-420.

Locat, J. and Demers, D. 1988. Viscosity, yield stress, remolded strength, and liquidity index relationships for sensitive clays. Canadian Geotechnical Journal, **25**: 799-806. doi: 10.1139/t88-088.

Masala S. and Matthews, J. 2010. Predicting development of undrained shear strength in soft oil sand tailings. *In* Proceedings of the 2nd International Oil Sands Tailings Conference, Edmonton, Alberta, 5-8 December 2010, University of Alberta, pp. 31-40.

Masala, S., Langseth, J., Smyth, K, Dunmola, A., Mckay, D., and Clark, J. 2012. TSRU tailings pilot-scale thickening and deposition trials. *In* Proceedings of the 3rd International Oil Sands Tailings Conference, Edmonton, Alberta, Canada, 3-5 December 2012, pp. 333-342.

Masliyah, J., Zhou, Z., Xu, Z. Czarkecki, J. and Hamza, H. 2004. Understanding water-based bitumen extraction from Athabasca oil sands. Canadian Journal of Chemical Engineering **82**(8): 628-654.

Mesri, G. 1975. New design procedure for stability of soft clays. (Discussion). Journal of Geotechnical Engineering, **101**(4): 1090-1093.

Mesri, G. 1989. A reevaluation of $S_{u(mob)} = 0.22\sigma'_p$ using laboratory shear tests. Canadian Geotechnical Journal **26**: 162-164.

Miller, W.G., Scott, J.D. and Sego, D.C., 2011a. Effect of extraction water chemistry on the consolidation of oil sands fine tailings. CIM Journal **1**(2): 113-129.

Miller, W.G., Scott, J.D. and Sego, D.C., 2011b. Influence of extraction process and coagulant addition on thixotropic strength of oil sands fine tailings. CIM Journal **1**(3): 197-205.

Mitchell, J.K. and Soga, K. 2005. Fundamentals of Soil Behaviour, 3rd Edition, John Wiley and Sons, Hoboken, New Jersey, USA.

OSTC. 2012. Technical Guide for Fluid Fine Tailings Management. Oil Sands Tailings Consortium (OSTC), Calgary, Alberta, 30 August 2012. Available from http://www.cosia.ca/uploads/files/Projects/Tailings/Tech_Guide_Fluid_Tailings_Mgmt_30%20Aug%202012_COSIA.pdf [cited 21 March 2013].

Proskin, S. 2012. Revisiting 1992-94 field tests of freeze-thaw dewatering of oil sands MFT. *In* Proceedings of the David C. Sego Symposium, Edmonton, Alberta, 26-27 April 2012, pp. 101-112.

Proskin, S., Sego, D. and Alostaz, M. 2010. Freeze-Thaw and Consolidation Tests on Suncor Mature Fine Tailings (MFT). Cold Regions Science and Technology, **63**(3): 110 – 120.

Reid, D. and Fourie, A., 2012. The effects of "in-line" flocculation on the geotechnical properties of clay tailings. *In* Proceedings of the 16th Tailings and Mine Waste 2012, Keystone, Colorado, Unites States of America, 14-17 October 2012.

Schiffman, R.L., Vick, S.G., and Gibson, R.E. 1988. Behaviour and properties of hydraulic fill structures. Geotechnical Special Publication, No 21. ASCE, New York, N.Y. pp. 166-202.

Shell Canada Energy (Shell). 2010. Muskeg River Mine – Annual tailings management plan 2011 – D074 Appendix E Requirement. Report submitted to ERCB, September 29, 2010. http://ercb.ca/regulations-and-directives/directives/directive074/tailings-2010 [cited May 5, 2012].

Sobkowicz, J.C. and Morgenstern, N.R. 2009. A geotechnical perspective on oil sands tailings. *In* Proceedings of the 13th International Conference on Tailings and Mine Waste, Banff, Alberta, 1-4 November 2009. University of Alberta, pp. xvii-xli.

Sobkowicz, J.C. 2010. History and development in the treatment of oil sands fine tailings. *In* Proceedings of the 14th International Conference on Tailings and Mine Waste, Vail, Colorado, USA, 17-20 October 2010. Taylor Francis Group, pp. 11-30.

Sorta, A.R. and Sego, D.C. 2010. Segregation related to use of Centrifuge in Oil Sands Tailings. *In* Proceedings of the 63rd Canadian Geotechnical Conference, Calgary, Alberta, 12-16 September 2010. Canadian Geotechnical Society, pp. 656-664.

Suthaker, N.N and Scott, J.D. 1997. Thixotropic strength measurement of oil sand fine tailings. Canadian Geotechnical Journal **34**(6): 974-984.

Wells, P.S., Revington, A., and Omotoso, O. 2011. Mature fine tailings drying – technology update. *In* Proceedings of Paste 2011, R.J. Jewel and A.B. Fourie (eds), Perth, Australia, 5-7 April 2011, Australia Centre for Geomechanics, pp.155 - 166.

Yuan, S. and Lahaie, R. 2009. Thickened tailings (paste) technology and its applicability in oil sand tailings management. *In* Proceedings of the 13th International Conference on Tailings and Mine Waste, Banff, Alberta, 1-4 November 2009. University of Alberta, pp 829-845.

Zhang, Y. and Sego, D. 2012. Freeze-thaw enhancement of Albian Mature fine Tailings (MFT). *In* Proceedings of the 3rd International Oil Sands Tailings Conference, Edmonton, Alberta, Canada, 3-5 December 2012, pp. 267-274.

7 APPLICATION OF A TAILINGS MANAGEMENT SIMULATION MODEL TO AN OIL SANDS MINE

7.1 INTRODUCTION

Tailings management is an inherent component of any water based mining process. In the oil sands mining industry, tailings management has evolved from simple fluid storage in single external impoundments to multistage mechanical and chemical dewatering processes and storage in several in-pit and external impoundments. The industry is currently focusing on transforming their fluid tailings and waste materials into deposits that can be incorporated into closure landforms and subsequently reclaimed (Sobkowicz and Morgenstern 2009). As discussed in Chapter 2, there are numerous technologies that may potentially transform the fluid tailings streams into geotechnically stable deposits. A joint industry-government study (Tailings Roadmap) was undertaken in 2012 to screen and evaluate the hundreds of tailings dewatering and reclamation technologies (Sobkowicz 2012). The Tailings Roadmap project ranked and sorted the technologies based on professional qualitative opinions and developed nine "technology roadmaps" with potential to improve oil sands tailings management practices. Since the evaluations did not include site-specific considerations, each technology requires assessment of applicability to the individual mine sites. Additionally, new technologies, processes and applications are constantly brought forward to the industry. Majority of the technologies and vendors lack detailed understanding of oil sands operations (i.e. technology exploited from another industry) or the technologies are conceptual or bench scale and require further research and development.

To address the ongoing need to evaluate tailings management technologies and processes, a dynamic systems model, TMSim was developed (Chapter 3). The TMSim model will provide industry a quantitative tool to aide in the evaluation of technologies and provide guidance to the developers on strengths and limits of the technology. TMSim was developed to incorporate mine plan data, various stages of dewatering including classification, pre- and post- deposition dewatering, and an impoundment material balance including tailings, process water, construction material and capping materials.

The objective of this chapter is to demonstrate the application of the TMSim simulation tool to an existing oil sands mine tailings plan. All data utilized in the following simulations were collected from publically available sources of information. The Syncrude Canada Ltd. (Syncrude) Aurora North (Aurora) mine was chosen as the model site. The TMSim model will simulate Syncrude's current tailings technology, CT, a method of blending fine grained fluid tailings with coarse grained cyclone underflow as presented in their recent Directive 074 tailings management plan (Synrcude 2012).

7.2 OIL SANDS MODEL DATA

7.2.1 Aurora Model Mine Plan

The following model oil sands mine plan is based on information from the Syncrude Aurora mine and tailings management data obtained from the Aurora North Environmental Impact Assessment report (Reeves 1996) the 2012 Annual Tailings Plan report (Syncrude 2012) and the 2010 Baseline Survey for Fluid Deposits (Syncrude 2010a). To develop the model mine plan described below, several adaptations and assumptions were required which are detailed in Appendix 3.

The Syncrude Aurora North oil sands mine (started in 2000) uses truck and shovel surface mining technology. A warm water extraction process is utilized to separate bitumen from the ore. Bitumen froth produced at Aurora is pipelined to another Syncrude mine site, Mildred Lake, for further processing and upgrading. The Aurora mine excavates approximately 95.5 Mm^3 of ore and mine waste overburden per year resulting in approximately 200,000 bbl/day of bitumen production. The average Aurora ore contains 11.1 % bitumen, 3.5 % water, 15.1 % fine mineral solids (<45 μm) and 70.3 % coarse mineral solids (Figure 7.1). Stripped overburden (mine waste) consists of high, medium and low spec materials (based on clay content), classified for applicability as construction material and plant rejects. The overburden classification is referenced in Syncrude (2012) but no specific ranges of clay contents are provided. The average ratio of excavated ore to overburden and mine waste is 1.59. A summary of the annual ore and waste production schedule is included as Figure 7.2.

The ultimate Aurora mine pit limits (Figure 7.3), were digitized to determine the pit dimensions and area (Figure 7.4). The Aurora pit covers approximately 48.8 km^2. Based on the total mined volume of ore and overburden from 2000 to 2012 estimated at 800 Mm^3 (Appendix 3) and 2578 Mm^3 calculated for 2013 to 2039, an average pit depth was determined to be 69.2 m (Appendix 3). Based on the average ore to overburden ratio of 1.59, the overburden thickness is approximately 25.7 m and the ore is 43.5 m thick. These estimated values agree with Reeves (1996) report, with an overburden thickness (including intra-burden) of 23 to 28 m and ore depths from 41.7 to 53.7 m on average. The mine pit slopes in overburden zones are 3:1 and 2:1 in the ore body formation. Overburden disposal dumps will have side slopes of 3 to 4:1 and as low as 8:1 where poor foundation conditions are expected to a maximum height of 60 m.

The primary extraction process at Aurora produces a tailings stream consisting of sand, silts, clay, residual bitumen and water referred to by Syncrude as straight coarse tails (SCT) otherwise known as whole tailings. A small stream of floatation tailings is also produced during the extraction process. The floatation tailings will be incorporated and managed with the whole tailings. Historically, the whole tailings stream was deposited into an ETF known as the Aurora settling basin (ASB). The coarse fraction of the tailings stream settled to form beaches and structural components of the ASB while the fine-grained fraction settled in the pond to form FFT otherwise known as MFT. The ASB was used to store tailings (SCT and FFT) until sufficient in pit space was available in mid-2010. The ASB was constructed with overburden materials and SCT using beaching and cell construction techniques. The ASB currently contains 79.9 Mm^3 of FFT and 46.5 Mm^3 of process water (Syncrude 2012) and is at its maximum construction elevation. The digitized pond surface area is 6.19 km^2. The maximum planned storage volume of the ASB is 129.8 Mm^3 of FFT and process water (based on maximum volume calculated from Syncrude 2012) plus a coarse cyclone overflow deposit of 51.3 Mm^3.

The Aurora mine pit will eventually be used as in-pit storage areas or DDAs for the tailings. Under the current tailings plan, a total of six in-pit dykes will be used to segregate each of the DDAs and allow safe deposition of tailings and fluid storage while mining progresses within the pit. The Aurora mine storage areas and dyke locations (centerlines) are included in Figure 7.3 and Figure 7.4. The Aurora east pit north (AEPN) DDA currently contains 18.6 Mm^3 of FFT, 0.7 Mm^3 of water and 47.3 Mm^3 of SCT.

The tailings planning parameters utilized by Syncrude for the Aurora mine are included in Table 7.1. The SCT stream is used for beaching and cell construction. The Stage 1 beaching model for Syncrude Aurora is summarized in Appendix 1. By assuming a fines capture fraction (typically 50%) the amount of sand that is captured in the beach can be calculated from the fines content of the SCT (or ore). The final average beach/cell density was calculated as 1.53 tonne/m^3.

Syncrude assumes the consolidated average (by depth) solids content by weight (C_w) for the FFT will be 43-45%. From the tailings pond baseline report, in ASB, the FFT Cw ranges from 20% near the surface to an average of 70% at depth (27 m deep; Syncrude 2010a). CNRL utilized historic Syncrude Mildred Lake settling basin (MLSB) FFT data to develop a model settlement data set (Table 7.2). The CNRL model projects a lower FFT solids content than the Aurora estimate. However, the SFR of either data set is not provided. The increased C_w for Aurora is likely due to an increased sand content. For modeling purposes, the CNRL data set will be used.

Properties of several samples of fine tailings originating from Syncrude are presented in Table 7.3. COF is cyclone overflow from a 2008 sampling program (Jeeravipoolvarn 2010), MFT is from Scott (2014) and represents an aged FFT sample, Ore A (marine) and Ore B (non-marine) represent young fine tailings derived from a caustic extraction process (Miller et al. 2011), and the 10 m sample represents the MFT stored in a 10 m column for 30 years at the University of Alberta (Jeeravipoolvarn et al. 2010). The FFT from the ASB can be expected to have fines contents (F,%) greater than 90%, and clay contents from 30 to 50%.

Aurora's water requirements will be satisfied by recycling process water on site, importing water from MLSB, site run off (precipitation), and mine depressurization wells. The ASB will be used as the primary source of recycle water (Reeves 1996). Site runoff consisting of precipitation on the mine site was estimated at 0.8 m^3/m^3 of bitumen produced. Mine depressurization was estimated at 0.176 m^3/m^3 of bitumen produced and surficial muskeg and aquifer dewatering at 0.32 m^3/m^3 of bitumen produced. Site runoff, mine depressurization and surficial waters will be directed to ASB for use as recycle water. Process losses were estimated at 0.2 m^3/m^3 of bitumen produced. The target water storage volume at Aurora is 40 Mm^3 in the ASB and in-pit cells. Water import from MLSB will be used to make up any deficiencies in the target storage volume. Seepage loss from the ASB during active mining operations was estimated as negligible due to the seepage collection measures in place (interception ditches and cut off walls) (Reeves 1996). Any seepage collected is returned to the ASB pond. Vertical seepage was also assumed to be negligible due to the low saturated hydraulic conductivity of the underlying soils (Reeves 1996).

Select inorganic ion concentrations for various pore and process water streams at the Aurora site are included in Table 7.4. Two different ore connate water and fine tailings are provided by Miller et al. (2011). The ion concentrations for Ore B tailings are similar to the ASB MFT pore fluid, therefore, the Ore B connate water will be used as connate water for the Aurora mine. The pond water from the ASB is also provided along with the Athabasca river water and for comparison, Syncrude's MLSB pond water quality. The ASB pond water quality will be used as the recycle water quality in the extraction process. Mine runoff water (tailings seepage and depressurization water) quality was provided by

Syncrude (2010b) and will represent the runoff quality. The Basal aquifer water quality was provided by Reeves (1996) and is representative of the deep depressurization water quality.

The Aurora site, situated in northern Alberta, endures cold winters and warm summers. Based on Environment Canada's (2014) 30 year climate normals for Fort McMurray (1981 to 2011), the average daily temp in January and July are -17.4^{o}C and 17.1^{o}C. The mean annual precipitation is 418 mm of which 316.5 mm is rain (the remainder is snow). Majority of the rain (67 %) falls as intense short storms from June to August. The mean monthly temperature and precipitation (rain, snow, and total) are included on Figure 7.5. Also included in Figure 7.5 is the mean monthly precipitation and potential evaporation reported by Song et al. (2011). Their data is based on 100 years of records from mining sites near the Aurora mine and Fort McMurray area. Song et al (2011) report a mean annual precipitation of 470 mm with snowfall accumulating from November through February. The mean annual potential evaporation is 640 mm with evaporation occurring from April to September. Melting occurs in March and April providing significant melt run off. For comparison, Reeves (1996) estimated a mean annual evaporation of 667 mm and mean annual precipitation of 426 mm. Climate data for the Aurora Site will be based on the mean annual temperature data from Environment Canada and potential evaporation/precipitation data from Song et al (2011).

7.2.2 CT Technology

The tailings management strategy at Aurora includes CT made from CUF and FFT dredged from ASB, at an SFR of 4, with gypsum added as a coagulant at a dosage of 1200 to 1400 g/m^3. Composite tailings will be deposited into one of five in-pit storage DDAs. At the end of mining, FFT not incorporated into a CT deposit will be transferred and managed in a water capped EPL. The annual CT, SCT and FFT production schedule is included in Figure 7.2. The CT tailings planning assumptions for the Aurora mine are included in Table 7.1.

An empirical hydrocyclone Stage 1 model (Appendix 1) was developed based on the reported Syncrude tailings plans (Syncrude 2012). Based on the F_{ORE}, the fraction of coarse sand and fines captured in the cyclone underflow can be calculated. The volume of the cyclone overflow is also a function of the ore fines content. The Syncrude CT technology assumes a cyclone underflow at a target C_w of 72% will be mixed with FFT at a C_w of ~ 44% with a target in pipe CT C_w of 65% at a SFR of 4. The target final deposit C_w is expected to be ~80% (density of 1.587 tonne/m^3). The TMSim Stage 2 dewatering model will utilize a user defined SFR of 4 and the CUF properties to calculate the demand of FFT required to meet the required SFR.

7.2.2.1 Impoundment DDAs

The digitized Aurora mine pit limits, DDAs and dyke locations (centerlines) are included as Figure 7.4. Since detailed elevation data and stage curves were not available for the in pit dykes and DDAs, model dykes and seven DDAs were developed and used as a surrogate during the simulations (Appendix 3). Each of the seven model DDAs are equal in size. They will be square in shape with dimensions of 2.74 km by 2.74 km. Figure 7.6 depicts the layout of the seven model DDAs over the actual pit limits. Cross sections of the model DDAs are presented in Figure 7.7 and Figure 7.8. Based on the total mined ore (800 Mm^3) to year 14 (2013), zones occupying DDA 1 and 67% of DDA 2 have been completely mined (Figure 7.9). At current mining rates, subsequent DDAs will be fully mined approximately every 6 years.

A total of 6 in-pit dykes will be used to segregate the seven DDAs and allow safe deposition of tailings and fluid storage while mining progresses within the pit. DDAs 1 – 5 will be used for CT deposition while DDAs 6 and 7 will be an EPL. In-pit impoundment dykes are to be constructed of overburden with the final 10 m elevation using SCT. The in pit dykes will be constructed at 5:1 side slopes. The crest width for overburden structures will be 180 m based on the design included in Syncrude (2012). The final 10 m of the dyke will utilize cell construction techniques using coarse tailings. The overall footprint of the dykes will be approximately 900 m wide and 2.74 km long. Mining will need to progress at least 450 m beyond the dyke centerline before the dyke can be completed. The first in pit dyke 1 has been constructed by start of year 14 (2013). Dyke 2 will be constructed from a combination of overburden and coarse tailings. Dykes 1 and 2 will be constructed to an elevation of 80 m, and dykes 3 to 6 to a final elevation of 74 m. The construction material demand for the DDAs will be based on downstream construction techniques so early storage is available for runoff from the starter beaches. Assuming a crest width proportional to the height, the elevation/material demand (overburden) for each dyke is provided in Figure 7.10.

Tailings will be deposited into DDA 1 and 2 to an elevation of 80 m plus 4 m of a coarse sand cap to account for an estimated 15 m of settlement. This will ensure the final deposit surface elevation is at the ground surface. Tailings deposited into DDAs 3, 4, and 5 will continue to a final elevation of 69 m (existing ground surface) plus a 4 m coarse sand cap. The final tailings deposit surface in these DDAs will be below the existing ground surface, but will promote drainage from the site to the planned EPL in DDAs 6 and 7 (Reeves 1996).

The demand for coarse tailings beaching and dyke construction will take precedent over CT deposition. Capping of previous CT DDAs will also be completed prior to or during active CT deposition. Each DDA will require a minimum of 4 m of SCT cap. Based on the area of each DDA (7.508 km^2) a total of 30 Mm^3 of sand for capping. Therefore, before CT can be deposited into a DDA, sufficient mining excavation must have been completed to construct the overburden dyke, the overburden availability must meet the dyke construction demand, starter beaches have been placed and the previous DDA has been capped. Runoff from the starter beaches and capping activities will be collected in the active DDA and transferred to new DDAs as they are constructed. A summary of the overburden, coarse tailings beaching and capping required for each DDA is summarized in Table 7.5. No beaching is required in DDAs 6 and 7 as they will serve as the EPL for residual FFT and process water.

7.2.2.2 Deposition and Beaching

Detailed deposition plans were not available for the CT technology therefore the following deposition plan was developed based on available information. CT tailings will be deposited on opposite sides of each DDA so they can be efficiently in filled and keep fluids away from the constructed dykes. A typical cross section of the CT deposit is provided as Figure 7.11 and a plan view in Figure 7.12. Due to the symmetric nature of the depositional plan, only one observation point is need per DDA to monitor the deposit profiles.

The depositional and beaching behavior of CT was extracted from the Syncrude 1995 NST Pilot report. During the pilot trial, NST (non-segregating tailings) or CT was deposited at an average flow rate (Q_{tails}) of 370 m^3/hr from a 0.2 m diameter pipe onto a beach. The average pipe C_w of the CT was 62% with an average D_{50} particle size of 160 µm. During the initial deposition, sheet and wave flow dominated the CT deposition with a slope of approximately 0.4%. With time the flow transformed to channel and lobe flow with accumulation at the base of the cell. An average beach slope formed at the end of each test was approximately 0.6-0.7% (iBAW).

Using the Fitton empirical beach slope estimation (Equation 7.1; Fitton 2007), the estimated slope was 1%.

[7.1] $$i = \frac{26.6 * {C_w}^2}{\sqrt{Q}}$$

Using Kupper's (1990) beach slope equation, the estimated slope was 0.5% (G = 2.65).

[7.2] $$i = 5\left(\frac{A(g*(G-1)*D_{50})^{0.5}C_w}{Q}\right)^{0.5}$$

The Syncrude (2012) tailings plan assumes a CT beach of 0.5%. The Fitton method provides an upper estimate of the max slope achieved during the pilot test program. Kupper's method provided an estimate of the earlier flow profiles and provides a lower bound. For initial modeling purposes, the slope will vary linearly from the lower (0.5%) to the upper (1 %) bounds based on the range of expected CT SFRs (3.5to 5).

7.2.2.3 Deposit Behaviour

Stage 3 consolidation dewatering will be the dominant dewatering process therefore compressibility and saturated hydraulic conductivity functions for the CT are required. The large strain dewatering compressibility behaviors for Syncrude CT at various SFRs was extracted from Matthews et al. 2002 (Figure 7.13). For reference, the range of compressibility behavior for several types of CT from various companies and using different coagulants as presented by Jeeravipoolvarn (2005) is also included on Figure 7.13. The Syncrude CT compressibility falls within the lower range of those reported by Jeeravpoolvarn (2005). Compressibility can be expressed as a power law function (Equation 7.3) relating void ratio (e) to effective stress.

[7.3] $$e = A\sigma'^{B}$$

The 'A' and 'B' parameters for the Syncrude CT can be calculated based on the varying (F%) as expressed in Equations 7.4 and 7.5.

[7.4] $$A = 0.304 \ln(F\%/100) + 1.404$$

[7.5] $$B = -0.156 \ln(F\%/100) - 0.374$$

According to Miller et al (2011) compressibility is not significantly impacted by water chemistry. During consolidation dewatering of the CT, the fines content will impact the compressibility, which will impact the magnitude of settlement (Suthaker 1995). The saturated hydraulic conductivity of the tailings will impact the rate of settlement (Suthaker and Scott 1995). Therefore it is also important to understand the change in saturated hydraulic conductivity (k) with e (Equation 7.6).

[7.6] $$k = Ce^{D}$$

The saturated hydraulic conductivity of the CT tailings (Matthews et al. 2002) is included on **Error! Reference source not found.**. The saturated hydraulic conductivity can also be expressed as a function of fines void ratio (e_{fines}) where fines void ratio is calculated using Equations 7.7, 7.8, and 7.9 (Suthaker and Scott 1996),

[7.7] $$e_{fines} = (e/F)*Sg_{fines}/Sg_{coarse}$$

[7.8] $$F = 1/(SFR+1)$$

[7.9] $$e_{fines} = e*(SFR+1)*Sg_{fines}/Sg_{coarse}$$

When compared to e_{fines}, the saturated hydraulic conductivity collapses to one single curve (**Error! Reference source not found.**) with parameters C and D equal to 3×10^{-9} and 5.47. Also included on **Error! Reference source not found.** for comparison is the NST/CT saturated hydraulic conductivity curves for CNRL (CNRL 2010), Suncor (Suthaker and Scott 1996) and the upper and lower bound of several CT tailings streams (Jeeravipoolvarn 2005). The differences in saturated hydraulic conductivities are related to the different source of fine tailings and coagulant type (carbon dioxide, gypsum, lime etc.).

The strength parameters for the Syncrude CT deposit were not available. On a sample of CT, Qiu (2000) measured the Mohr-Coulomb effective stress parameters of c' (cohesion) and φ' (friction angle) as 3 and 30^{o}. To simplify the estimation of strength in oil sand tailings like CT, CNRL employs an S_u/σ'_v ratio (CNRL 2010). This negates the requirement to determine the pore pressure at failure for a particular scenario. Based on field measurements of rapidly loaded CT, CNRL assumed a ratio of S_u/σ'_v of 0.12. Sensitivity analyses can be conducted on the strength ratio to bound the predicted strength. With further detailed information or testing, improved estimates of strength can be completed.

7.3 SIMULATION RESULTS

The TMSim model was utilized to assess a base case scenario using CT technology for a model oil sand data set using the Syncrude Aurora mine as a guide. Detailed model assumptions and raw input data are included in Appendix 3. A preliminary simulation was carried out with Stage 3 dewatering to assess the CT technology model and mine plan assumptions. A final simulation was then completed which includes Stage 3 dewatering. The TMSim performance measures presented in Chapter 5 will be discussed below for each scenario.

7.3.1 Preliminary CT Technology Simulation CT-1

During the the preliminary TMSim simulation of the CT technology (CT-1), all five DDAs were filled to their maximum storage capacity by the 294th month. Therefore, the simulation was terminated at the 294th month (2.5 years before end of mining) due to insufficient storage space for subsequent tailings depostion. Although the model stopped before the total time, sufficient data was generated to afford an assessment of the TMSim model results.

The extraction model utilized in the TMSim model slightly under estimated the bitumen extraction efficiency according to the Syncrude Aurora plan (Figure 7.16). Bitumen that is not extracted from the ore is incorporated with the fine mineral component of the whole tailings stream. The average residual bitumen content by mass of the whole tailings was 1.4% (Figure 7.16).

The average extraction process water reclaim rate from the ETF is 97 Mm^3/yr (Figure 7.17). The average water make up demand from the MLSB source was less than half the extraction reclaim rate at 40 Mm^3/yr. The difference is satisfied by water liberated from settling tailings, site dewatering and depressurization water, and precipitation. Over the life of the mine, the volume of water in the ETF was maintained at approximately 25 Mm^3 (Figure 7.18).

The total volume of each waste stream deposited in DDAs from the CT-1 simulation was compared with the Syncrude Aurora tailings plan in Figure 7.19. The site wide FFT volume at the end of mining for CT-1 was within 3% of plan volume. The Syncrude plan also included a coarse cyclone overflow (COF) deposit in the ETF. For the TMSim model, this was included as coarse sediment deposit in the ETF (Figure 7.18). The CT-1 simulation was within 12% of the Syncrude plan for the COF deposit. Similarly, the CT-1 volume of beach sand was within 10% of the Syncrude plan. The CT-1 simulation ended prematurely resulting in the lower volume of COF deposit and beach sand. Slightly more process water was accumulated in the final DDA 6, therefore the total site wide water balance was approximately 25% greater than the Sycnrude plan. The Syncrude plan utilized final settled densities to calculate the volume of CT tailings deposited into the DDAs. Since the preliminary CT-1 simulation did not incorporate Stage 3 consolidation, the final volumes were nearly 80 % greater than the Syncrude plan. To ensure the volume difference can be attributed to the difference in deposit densities, a mass balance comparison for the CT deposits was calculated (Figure 7.20). The CT-1 model deposited approximately 17% more mass than the Syncrude plan. The extra mass in the CT deposit can be attributed to the lower amount of beach deposit in the model. The final dry density of the CT-1 deposit was 1/3 of the Sycrude plan, therefore the difference in total CT volume is a function of the deposit density. Summaries for the deposit volumes and heights of the dyke, beach, CT, FFT and process water for each DDA can be found in Figure 7.21 to Figure 7.30.

DDA 1 reached capacity of 407.5 Mm^3 of CT after 79 months (Figure 7.21). Transfer of ponded water and FFT to DDA 2 started at month 58. Timing of the overburden dyke and starter beach construction provided adequate freeboard and storage in DDA 1 (Figure 7.22).

Starter beach and dyke construction was initiated at time 0 for DDA 2 in coordination with storage of process water and FFT (Figure 7.23). CT deposition started at month 82 and DDA 2 reached capacity (387 Mm^3) at month 150. Timing of the overburden/sand cell dyke and starter beach construction provided adequate freeboard and storage in DDA 2 (Figure 7.24). Transfer of ponded water and FFT to DDA 3 was initiated at month 119.

CT deposition in DDA 3 was initiated in month 151 and ceased in month 194 after 264 Mm^3 were deposited (Figure 7.25). Pond water and FFT transfer to

DDA 4 was initiated in month 168. Dyke and starter beach construction in DDA 3 were completed with sufficient time to ensure adequate freeboard and storage of pond water and CT tailings (Figure 7.26).

In DDA 4, CT deposition started at month 197 and reached capacity (338 Mm^3) at month 251 (Figure 7.27). Timing of the overburden dyke and starter beach construction provided adequate freeboard and storage in DDA 4 (Figure 7.28). Transfer of ponded water and FFT to DDA 5 was initiated at month 218.

CT deposition into DDA 5 was initiated in month 253 and continued until it reached capacity (month 294) after 265 Mm^3 were deposited (Figure 7.29). Pond water and FFT transfer to DDA 6 was initiated in month 268. Dyke and starter beach construction in DDA 5 were completed with sufficient time to ensure adequate freeboard and storage of pond water and CT tailings (Figure 7.30). At the end of mining operations, all residual FFT and process water in the ETF will be transferred to DDA 6 and 7 for the end pit lake (209 Mm^3 of FFT and 50.5 Mm^3 of process water).

The preliminary simulations did not include Stage 3 dewatering (consolidation), therefore, an assessment of the effective stress profile and ultimately, deposit strength, are not possible. Similarly, since no consolidation drainage water was generated, an assessment of the ion concentration in the process water would not be relevant.

7.3.2 Stage 3 Dewatering Simulation CT-2

A simulation was then conducted based on the mine plan assumptions and CT technology model from CT-1, with the Stage 3 dewatering process included. Sufficient in-pit storage was available for the CT tailings deposit in simulation CT-2. Only DDAs 1 through 4 were required to contain the tailings deposit. DDA 5 was only used to store FFT and excess process water.

The extraction model utilized in CT-2 was the same as CT-1, therefore the bitumen recovery results are the same. The average extraction process water reclaim rate from the ETF is 97 Mm^3/yr (Figure 7.31). The average water make up demand from the MLSB source was less than 20 % of the extraction reclaim rate at 17 Mm^3/yr. The difference is satisfied by water liberated from consolidating tailings, site dewatering and depressurization water, and precipitation. Over the life of the mine, the volume of water in the ETF was maintained between 20 and 25 Mm^3 (Figure 7.32).

In Figure 7.33, the total volume of each waste stream deposited into DDAs was compared with the Syncrude Aurora tailings plan. For simulation CT-2, more FFT was consumed during the production of CT resulting in about 30 % less FFT at the end of mining. Slightly more COF deposit was produced in CT-2 than the plan volume. Since CT tailings were only deposited into DDAs 1-4, no capping and less beaching was required for DDA 5. Therefore the total beaching volume in CT-2 was about 20% less than the plan volume. There was approximately 50% more process water on site at the end of mining because DDA 4 and 5 both contained about 20 Mm^3 of process water at the end of mining. The final CT deposit at the end of mining was about 50% greater than the plan volume. Taking into account the average dry density of the CT deposit (1.49 tonne/m^3), approximately 40% more mass was deposited as CT tailings than the plan. Summaries for the deposit volumes and heights of the dyke, beach, CT, FFT and process water for each DDA can be found in Figure 7.34 to Figure 7.41.

A comparison between simulations CT-1 and CT-2 for the filling times, CT deposit rise rates, and the transfer of FFT and process water for each DDA is provided in Table 7.6. For CT-2, the tailings rise rate was approximately 32% greater and tailings deposition period was 32% longer in each DDA than for CT-1. For simulation CT-2, the timing of the overburden dykes and starter beach construction provided adequate freeboard and storage in all DDAs.

A summary of the effective stress profiles at the end of filling for each DDA is included in Figure 7.42. The vertical depth axis was normalized by the maximum depth of the deposit to allow for a comparison between the four DDAs. Due to their greater loading rates, DDA 1 and 2 developed the greatest effective stress.

DDA 3 had the lowest loading rate and therefore developed the lowest effective stress. To demonstrate the impact of the 4m sand cap (i.e. surcharge load) on the tailings deposit, a surcharge was added to the final deposit profile. Profiles of the effective stress after 5 years are also included on Figure 7.42. The effective stress profiles increased significantly due to the surcharge loading, with the greatest impact on DDA 3.

Using the assumed S_u/σ'_v ratio of 0.12, predicted undrained shear strength profiles were calculated for the tailings deposits at times right after deposition ceased and 5 years after the surcharge loading (Figure 7.43). At the end of filling, the tailings deposits had very little undrained strength. Only the bottom 20% of the deposit achieved undrained strengths greater than 5 kPa. However, following the surcharge, the undrained strength profiles improved considerably. The upper layer of the deposits was greater than 5 kPa. The lower half of the deposits were also greater than 5 kPa. Due to the low effective stresses in the center of the deposits, the undrained strengths were below 5 kPa.

Predicted species concentrations in the ETF process water pond are included in Figure 7.44. Only sodium (Na), calcium (Ca), chloride (Cl) and sulphate ($S04^{2-}$) are included on the figure for clarity. The concentrations of Na and $S04^{2-}$ are influenced by the concentrations of these species in the recycle of process water from the CT DDA process water ponds. Calcium concentrations are relatively constant, while the chloride concentration drops initially. The large fluctuations in $S04^{2-}$ (and similarly Na) are as a result of the high concentrations in the CT release water that is recycled to the ETF. Figure 7.45 compares the $S04^{2-}$ concentration in each DDA with the ETF. The average $S04^{2-}$ concentration in the DDAs is approximately 800 mg/L. Due to the mixing with fresh run off and MLSB water, the $S04^{2-}$ concentration in the ETF does not reach the same concentration and is at an average of 400 mg/L.

7.4 Discussion of Results

CT-1 simulation was completed to confirm the TMSim model assumptions and UDFs would provide acceptable results for the model oil sands mine plan. Stage 3 consolidation was not included because the Syncrude tailings plan did not include active consolidation (only final deposit volumes). Therefore, comparison of the CT-1 results and Syncrude plan could be completed.

For the preliminary simulation CT-1, all of the DDAs reached capacity by the 294th month (2.5 years before end of mining). This was expected because no stage 3 consolidation dewatering was included in this simulation. For CT-1, the tailings were deposited at an average C_w of 63% (1.02 tonne/m^3). When Stage 3 dewatering is included, the CT tailings will attain a C_w of greater than 78% (1.59 tonne/m^3), resulting in a volume reduction of nearly 55%. Therefore, when stage 3 consolidation is included, the 5 DDAs will provide adequate storage for the CT deposits.

The extraction efficiency was slightly less than the Syncrude plan for a couple periods during the simulation. Since the mine and tailings plan used for the model were based on publically available information, it is likely there are some assumptions for the calculation that were not available, resulting in the difference. The average make – up water rate (40 Mm^3) from the MLSB source was within the expected range (26-51 Mm^3) but slightly greater than the expected average (35.4 Mm^3) according to Reeves (1996). Once consolidation water is included in the simulation, it is expected the make - up water rate will be reduced.

The model mine plan assumptions and UDFs for the FFT production, Stage 1 dewatering (hydrocyclone and beaching) and the Stage 2 dewatering (CT technology) are considered acceptable because of the agreement between the model deposit volumes and the Syncrude plan. Additionally, the storage demand for process water, FFT and CT deposits was met by the timely construction of impoundment dykes within the DDAs. Therefore, the model assumptions for dyke construction were also acceptable.

When Stage 3 consolidation was included in the simulation (CT-2), the process water make up rate decreased by ~ 50% due to the availability of water liberated during consolidation. The rate of consolidation did influence the water make-up rate. With improved consolidation rates, the process water make up rate can be reduced as more water is liberated from the tailings deposits.

Due to the greater CT tailings density, less storage volume was required for CT-2 (i.e. only needed 4 DDAs). Since beaching was then not required in DDA 5, more mass of tailings could be deposited as CT resulting in an increase of 40% over the Syncrude plan. To ensure, there were no errors a mass balance for the sand mass inputs to the TMSim model and the tailings deposits for CT-2 was calculated. The total mass of sand reporting to the beach, CT and cyclone overflow model was 2220 Mtonne. Using the results of the CT-2 model, the total mass of beach sand deposited into each of the DDAs was 557 million tonnes (Mtonnes) of sand. The combined mass of sand from each CT deposit was 1625 Mtonnes and the COF deposit in the ETF was 54 Mtonnes. Approximalety 4.5 Mtonnes of sand was in the FFT slurry on the site. The total mass of sand in the beach, CT, COF and FFT deposits was calculated as 2240 Mtonnes. This is within 1% of the mass input to the beaching, CT and cyclone sub models, therefore, no errors were encountered. For comparison, the Syncrude plan had 2200 Mtonnes of sand reporting to the tailings deposits. The CT-2 model was within 2 % of the Syncrude plan. Therefore, the results from CT-2 are considered reasonable.

Based on the consolidation parameters utilized (high saturated hydraulic conductivity and low compressibility) and strength ratio used for CT-2, the tailings deposits would not meet the required performance criteria set out in Directive 074 (10 kPa after 5 years). However, if the compressibility of the CT tailings were improved, the predicted undrained strength of the deposits would increase. This could be accomplished through increasing the SFR of the CT deposit (reducing the fines content of the CT mix). If the SFR was increased, less FFT would be incorporated into the CT deposit and result in a greater volume of FFT to manage at the end of mining in the EPL. Alternatively, improved drainage from installation of drains (lower pore pressure), and/or increasing the cap surcharge would lead to an increase in effective stress, and improved undrained strength.

A mixing model was used to assess the quality of the process water in the ETF and DDAs. Since no reactions were included in the TMSim model, the concentrations are only representative of conservative species (i.e. non reactive). Therefore, when mixing waters, streams with higher concentrations (i.e. sulphate in the CT release water) will have a greater influence on the final concentrations. It is recognized that sulphate is not considered a conservative species. However, it was considered a suitable surrogate due to its influence on the final concentrations. The high concentrations of sulphate serve to magnify the expected influence a conservative species making it easier to understand the influence of various pond waters on the ETF water quality. The water quality results indicate that the concentrations in the ETF will remain considerably less than the DDA process water that is dominated by the CT release water. As long as fresh water from site runoff and import are continually added to the ETF pond, the concentrations will stay low.

7.5 CONCLUSIONS

The TMSim modelling tool was utilized to simulate a model oil sands mine and its tailings storage plan based on using CT tailings technology. All assumptions and UDFs were based on publically available sources of information. Several parameters such as deposit slope, the consolidation parameters, and S_u/σ' ratio were based on estimated values and may not reflect actual material properties at a particular mine site. A preliminary simulation was conducted and the results were compared with the Syncrude tailings plan. Model mine plan assumptions and UDFs incorporated into the TMSim were shown to be acceptable because of the agreement with the Syncrude plan. A mass balance was also completed after implementation of Stage 3 dewatering. The total mass in the various tailings deposit also agreed with the expected mass in the Syncrude plan.

The compressibility and saturated hydraulic conductivity of the CT deposits influenced the filling rate of the DDAs, the make up reclaim rate, and the predicted strength profile of the deposits. Improvements to these properties, such as increasing the target SFR of the CT mix design, would improve the deposit strength. However, decreasing the mass of fines incorporated into the CT deposits would lead to a greater overall FFT volume at the end of mining.

Based on the agreement with the Syncrude tailings plan, the TMSim model was established to be an effective quantitative tool that can be used in the evaluation of technologies for oil sands mining operations.

7.6 TABLES

Table 7.1. Aurora tailings planning assumptions (modified from Syncrude 2012).

Model Parameter	**Unit**	**Value**
CT SFR		4:1
SCT Sg	Tonne/m^3	1.45
Floatation tailings Sg	Tonne/m^3	1.21
CT Slurry Sg	Tonne/m^3	1.6
Ore or OB Sg	Tonne/m^3	2.1
Beach above water slope	%	2
Beach below water slope	%	5
CT slope	%	0.5
FFT projected consolidation	%	43-45%

Table 7.2. Model FFT settlement (based on CNRL 2010).

Time (months)	Solids Content (% by mass)
0.1	12
30	30
98	35

Table 7.3. Syncrude tailings sample properties

Parameter	COF	MFT	10 m	Ore A	Ore B
Fines content (%)	94	96	89	94	96
Clay content (%)	30*	52	45	43	49
Liquid limit (%)	43	50	46	49.5	52.1
Plasticity Index (%)	25	29	25	23.7	25.2
Bitumen content (% - total mass)	3.3	3**	3.1	0.35	0.5
SG	2.52	2.44	2.28	2.55	2.48

Notes: * = non dispersed

** = % of mineral solid

Table 7.4. Inorganic Chemistry of select water and tailings streams.

	Select Anions (mg/L)			Select Cations (mg/L)		
Source	HCO_3^-	Cl^-	SO_4^{2+}	Mg^{2+}	Ca^{2+}	Na^+
Ore A Connate[1]	75	99.8	93.6	1.89	0	0
Ore B Connate[1]	90	24.6	66.8	3.15	0	1.39
Ore A fine tailings[1]	1346	583	526	10.6	3	1.8
Ore B fine tailings[1]	860	369	380	10.8	5.9	7.4
ASB MFT[2]	729	380	91.3	21	16.2	28.8
ASB Pond Water[2]	638	300	326	19.3	18	34
MLSB Pond Water[3]	775	540	218	NR	8	17
Athabasca River[3]	115	6	22	NR	8.5	30
Mine and Surficial Water[2]	550	121	299	30	140	211
Basal Aquifer[4]	1633.8	328	0.74	15.2	19.8	39
CT Release Water[5]	859	535	1182	20	56	1120

1 – Miller et al. 2011

2 – Syncrude 2010b

3 - Allen 2008

4 – Reeves 1996

5 – MacKinnon et al 2001 and Syncrude 1995

NR – not reported

Table 7.5. DDA material demand for CT technology.

DDA	OB Dyke (Mm^3)	SCT Beach* (Mm^3)	SCT Dyke	Deposit Capping (Mm^3)
DDA 1	101.65	73.3	14.2	30
DDA 2	101.65	122.0	10.7	30
DDA 3	134.6	66.3	3.6	30
DDA 4	89	28.0	3.6	30
DDA 5	89	63.7	3.6	30
DDA 6	89	0	0	0
DDA 7	0	0	0	0
Total	604.8	353.2	35.6	150

* - includes straight beaching and beach on dyke surfaces.

Table 7.6. Summary of DDA filling and tailings rise rates

DDA	CT-1	CT-2
DDA 1		
DDA Start	0	0
DDA Full	79	104
Rise Rate (m/month)	0.99	1.31
DDA 2		
DDA Start	83	107
DDA Full	150	199
Rise Rate (m/month)	0.84	1.16
DDA 3		
DDA Start	152	200
DDA Full	194	257
Rise Rate (m/month)	0.60	0.81
DDA 4		
DDA Start	198	258
DDA Full	251	324
Rise Rate (m/month)	0.75	0.94
DDA 5		
DDA Start	253	-
DDA Full	293	-
Rise Rate (m/month)	0.57	-

7.7 FIGURES

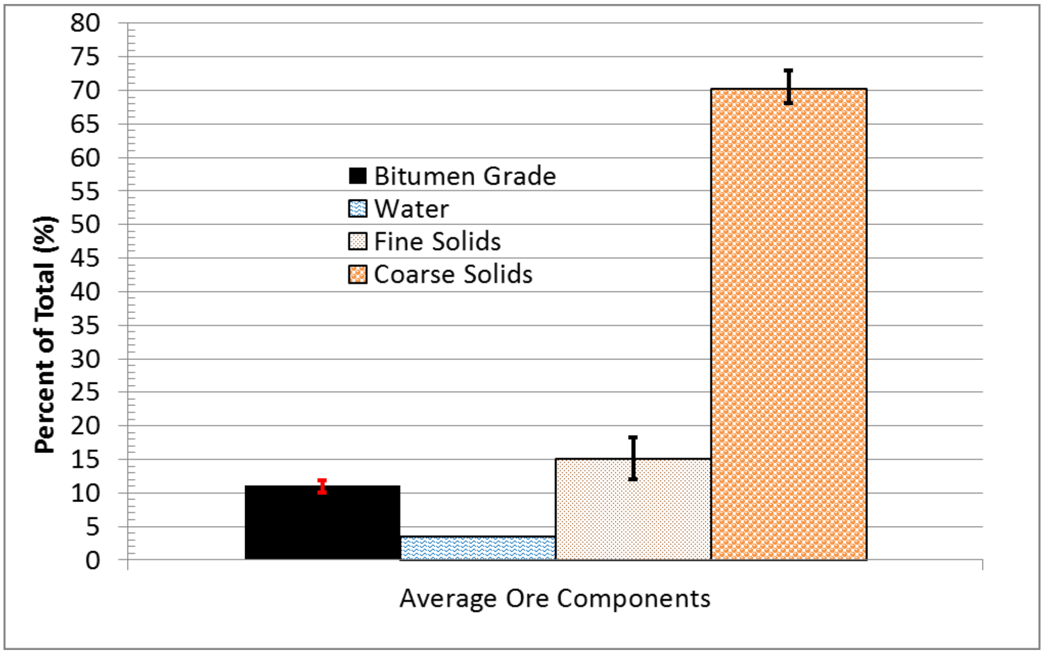


Figure 7.1. Average ore components of Syncrude Aurora North mine.

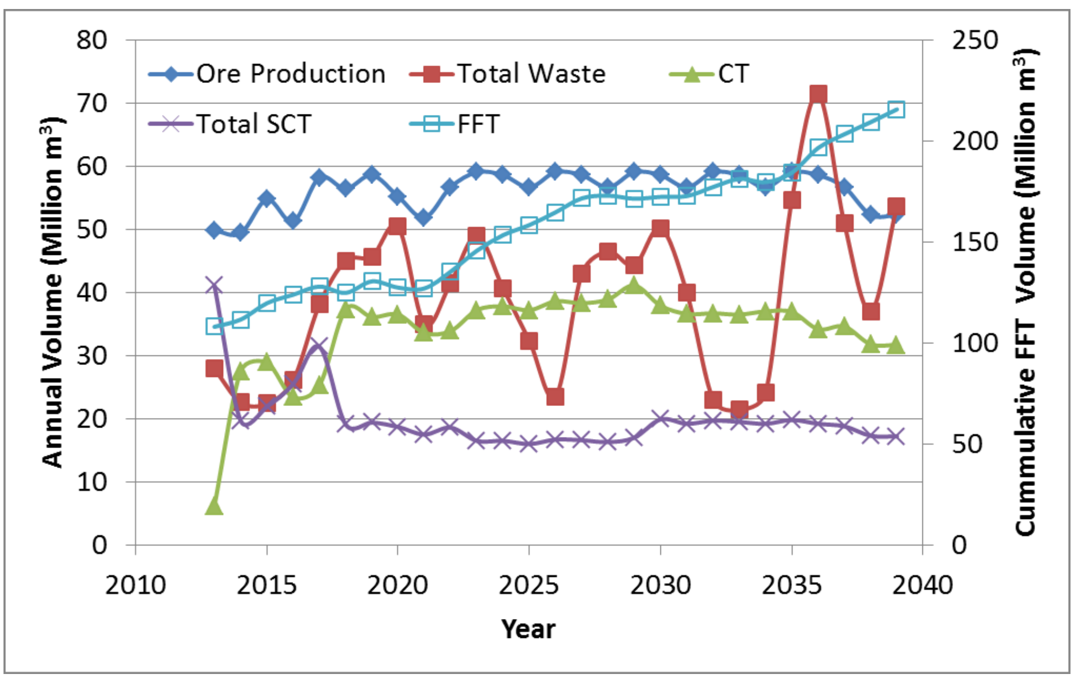


Figure 7.2. Annual ore, waste and tailings production at Syncrude Aurora North mine.

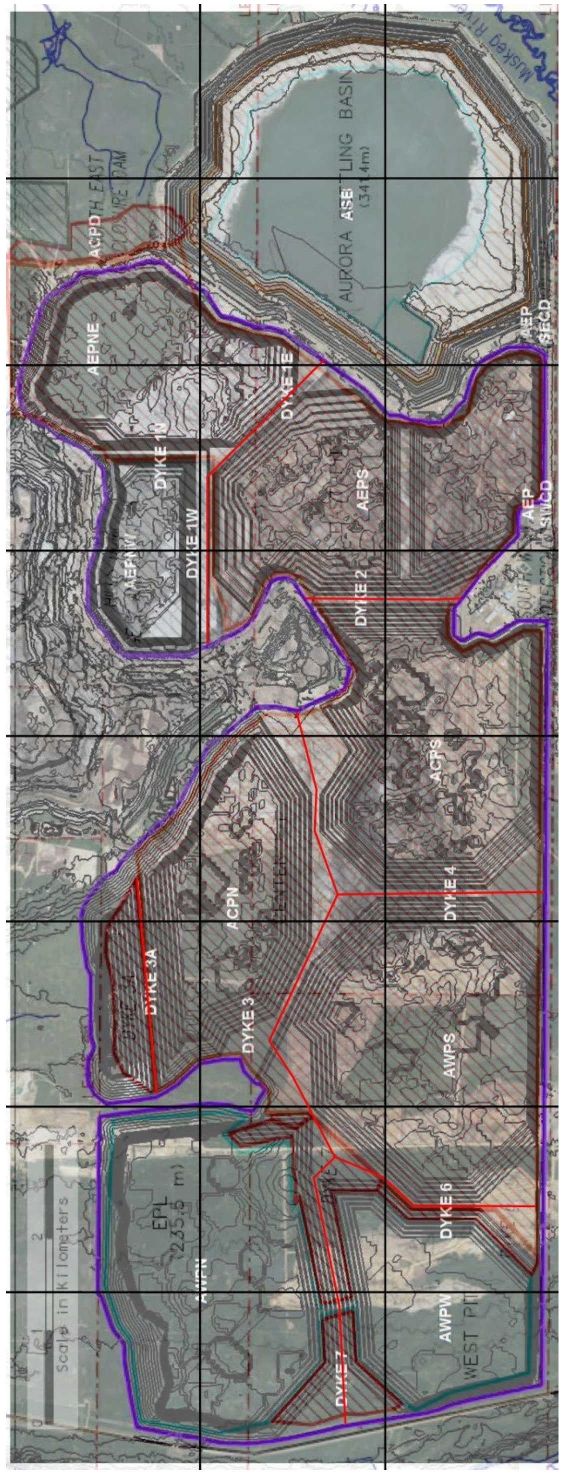


Figure 7.3. Syncrude Aurora North dykes and tailings areas (modified from Syncrude 2012). (Purple line = pit limit; red line = dyke centre line).

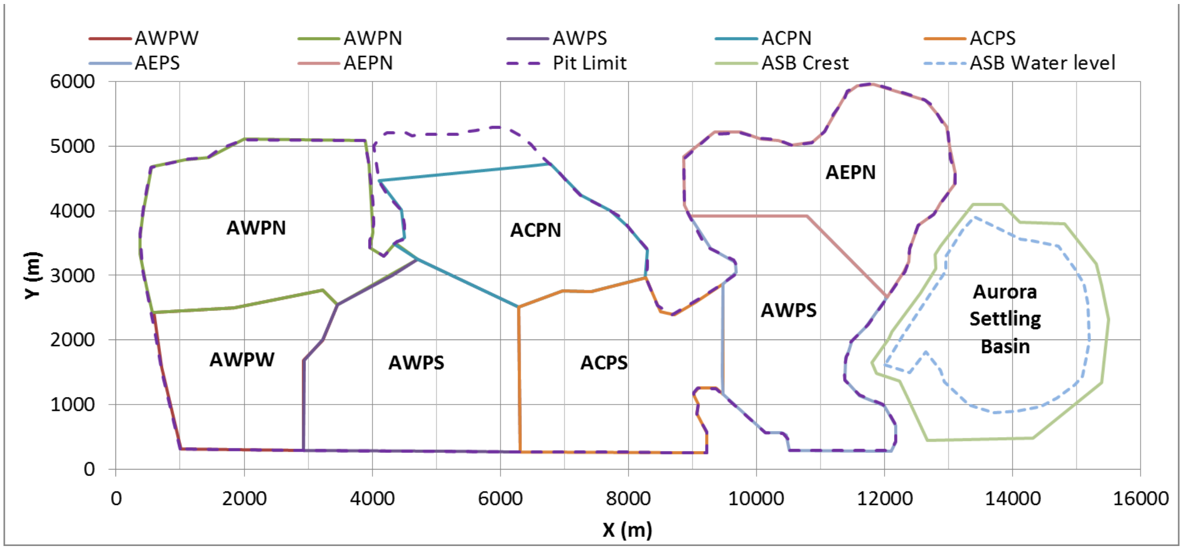


Figure 7.4. Syncrude Aurora North digitized pit and DDA limits.

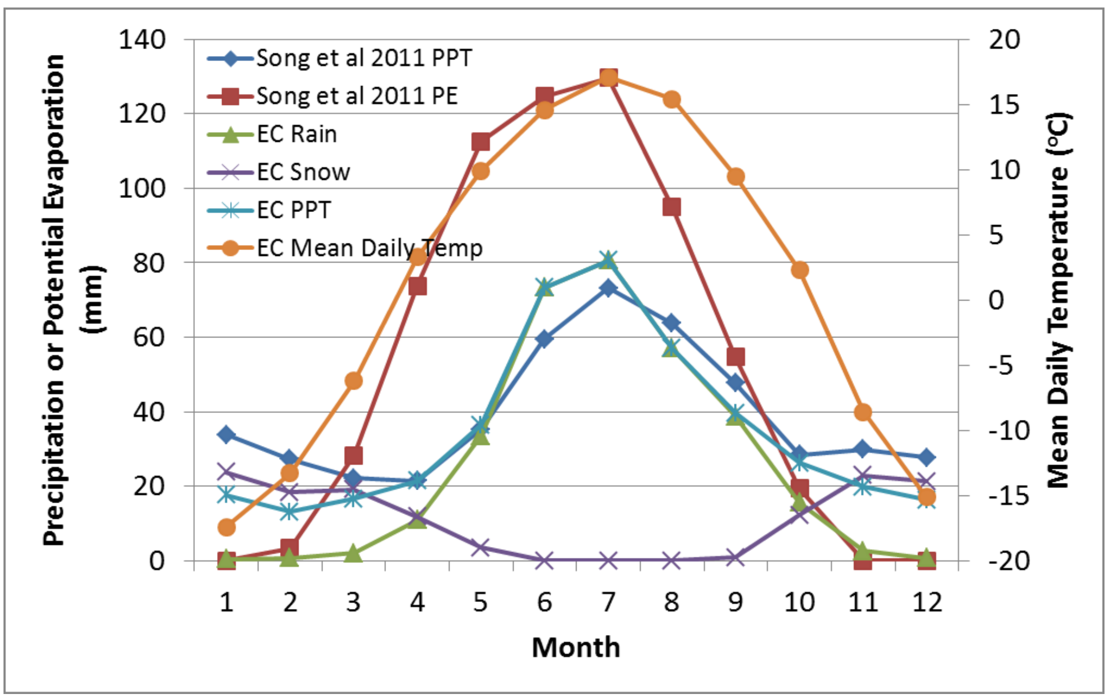


Figure 7.5. Climate normals for Fort McMurray Alberta.

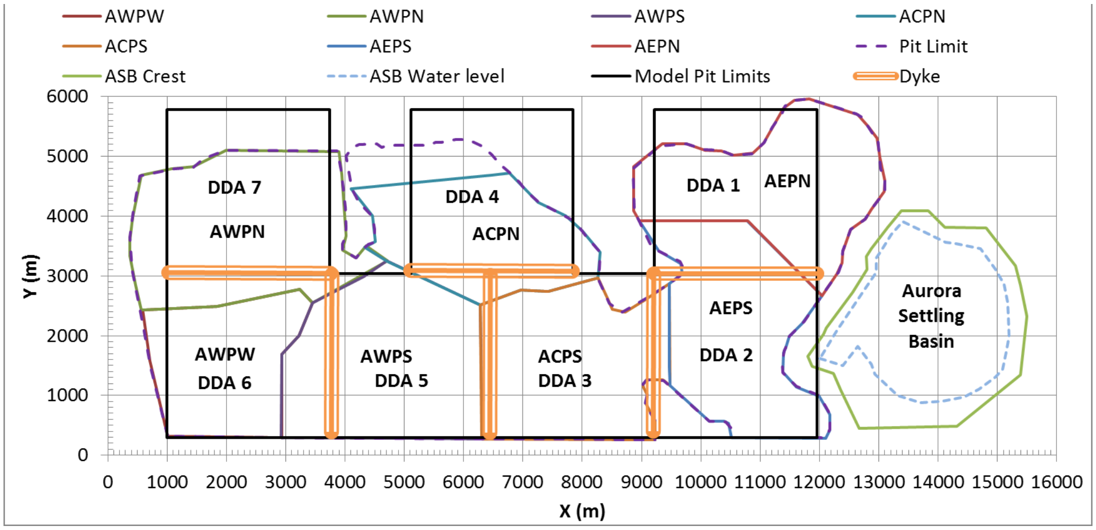


Figure 7.6. Model mine pit and in-pit DDAs for Syncrude Aurora.

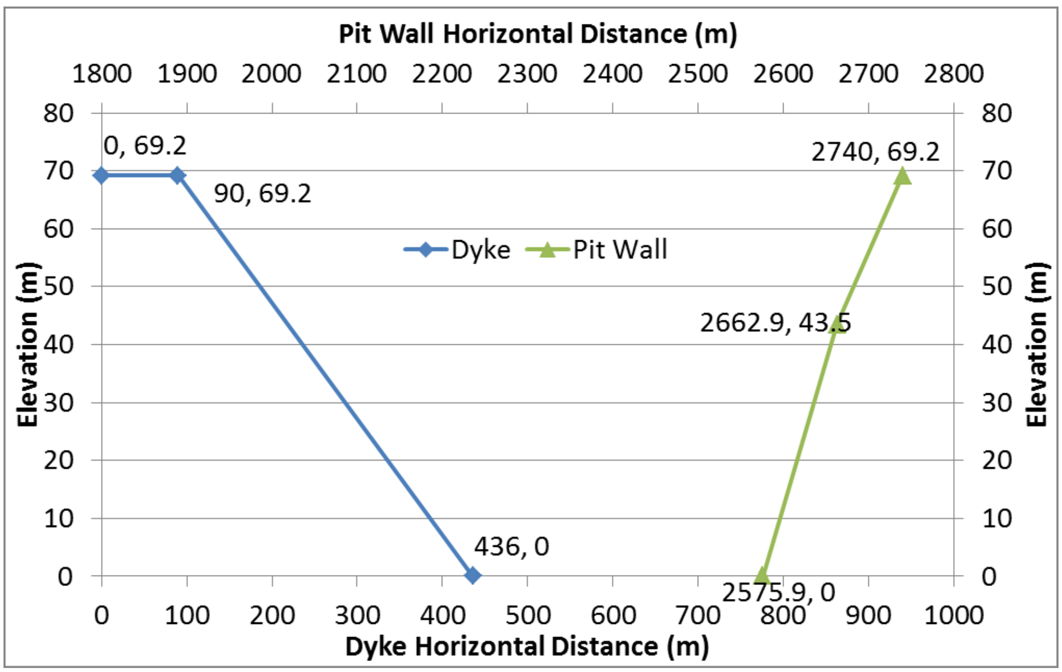


Figure 7.7. Cross section of model DDAs 1, 2, 4, 6, and 7.

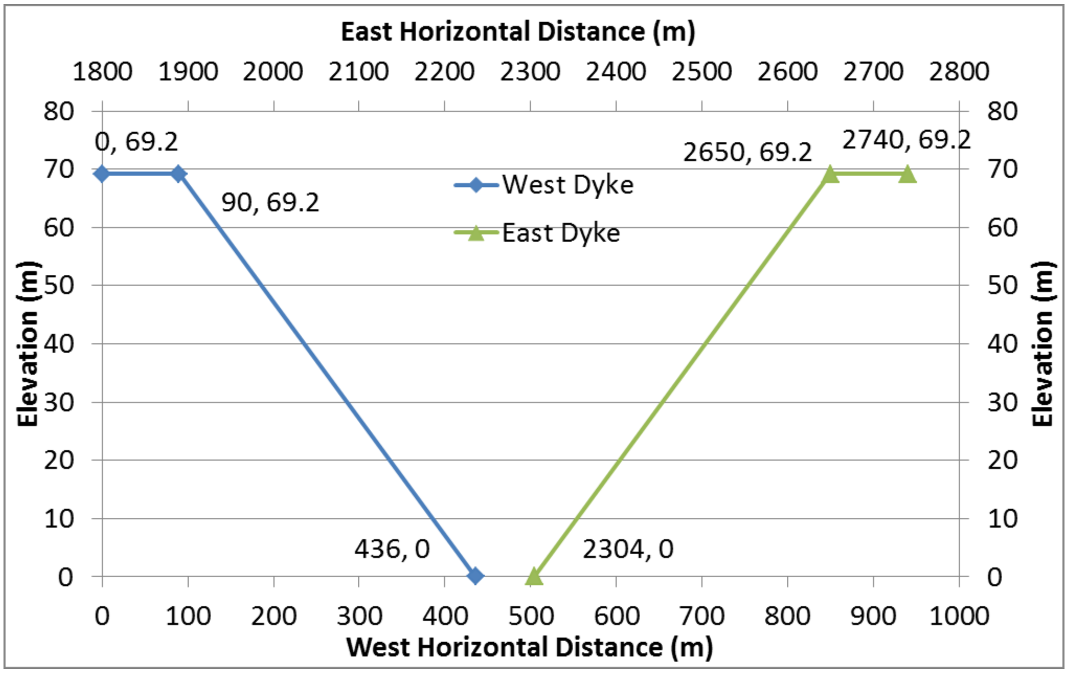


Figure 7.8. Cross section of model DDAs 3 and 5.

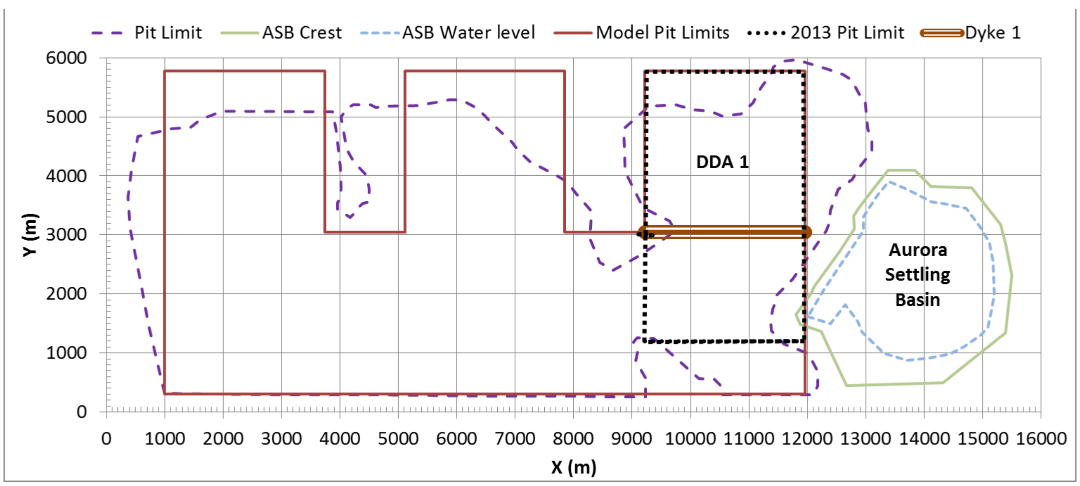


Figure 7.9. Mining extents for start of year 2013.

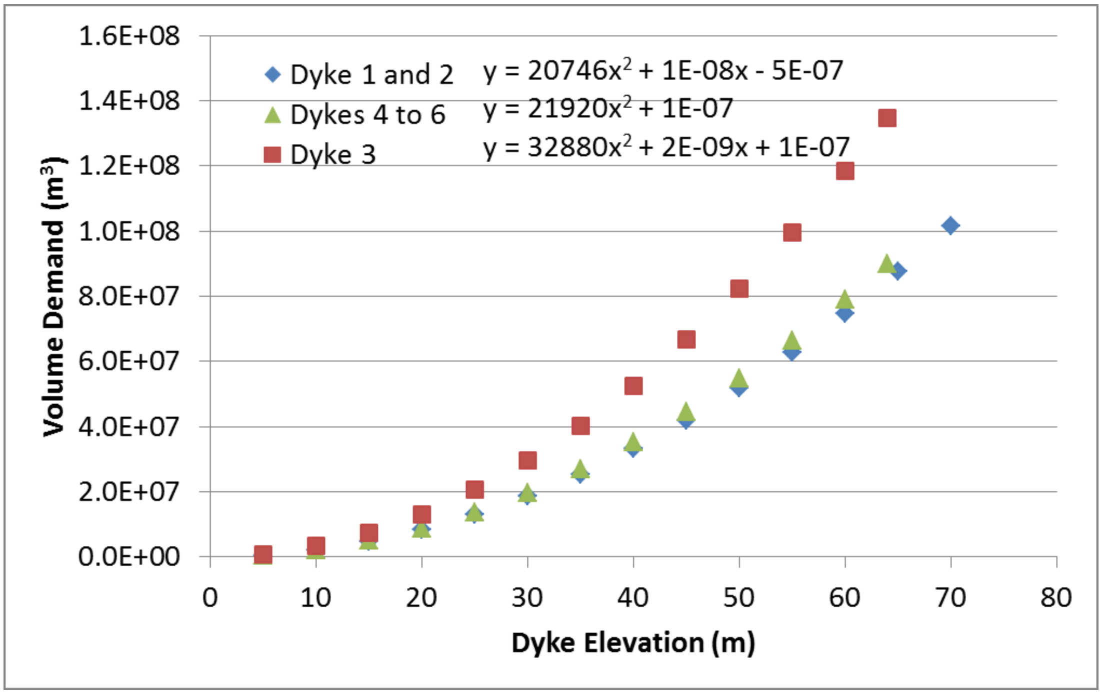


Figure 7.10. Typical CT DDA dyke construction material demand curve.

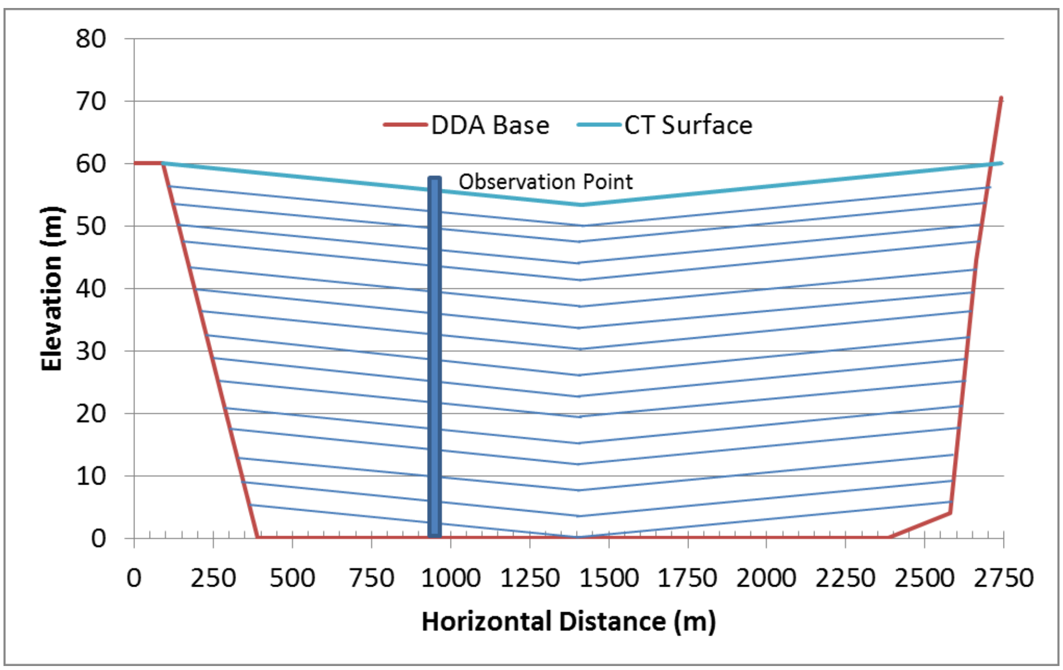


Figure 7.11. Cross section of typical CT DDA.

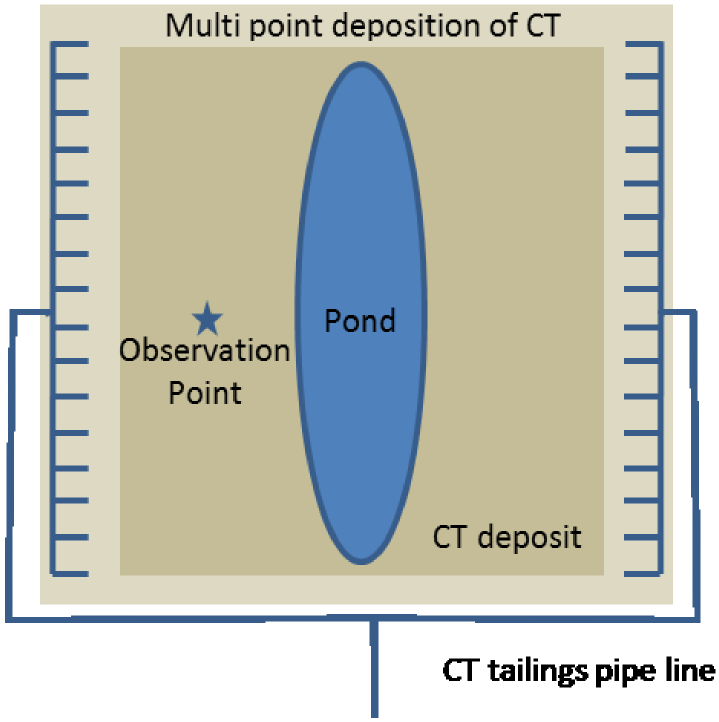


Figure 7.12. Plan view of CT DDA deposition.

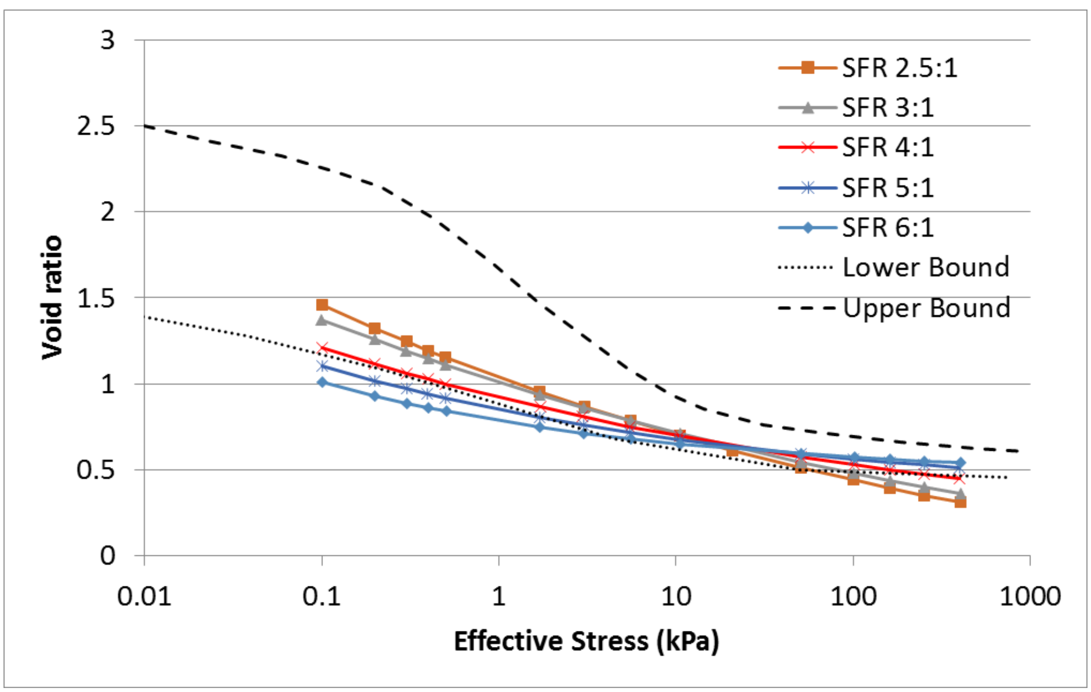


Figure 7.13. Syncrude CT compressibility at various SFRs.

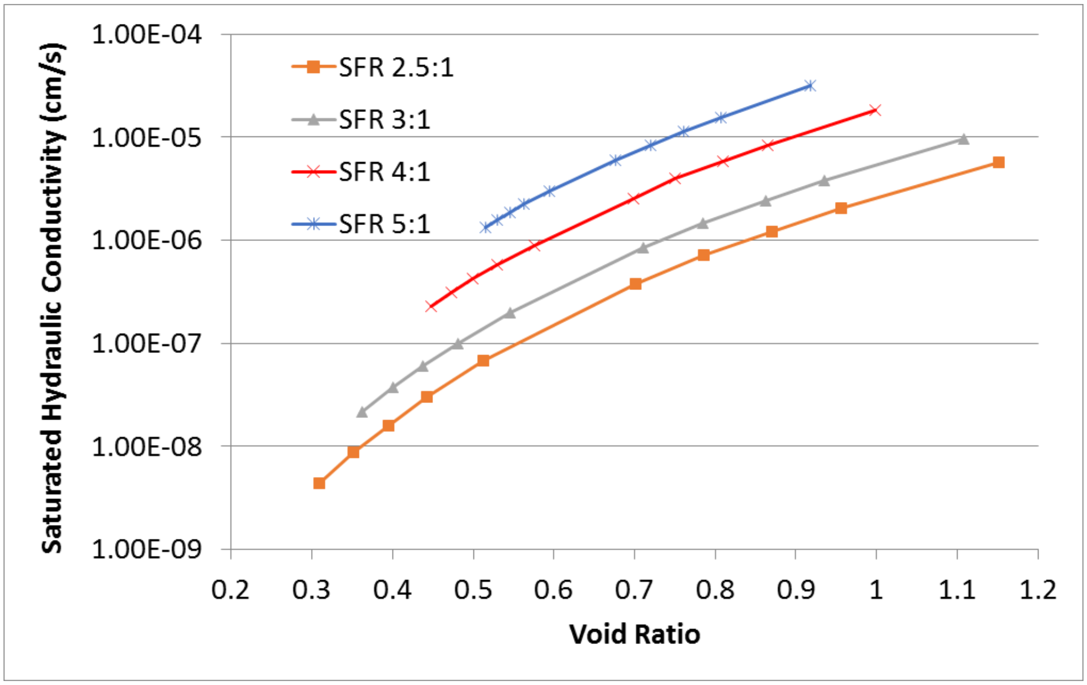


Figure 7.14. Syncrude CT saturated hydraulic conductivity at various SFRs.

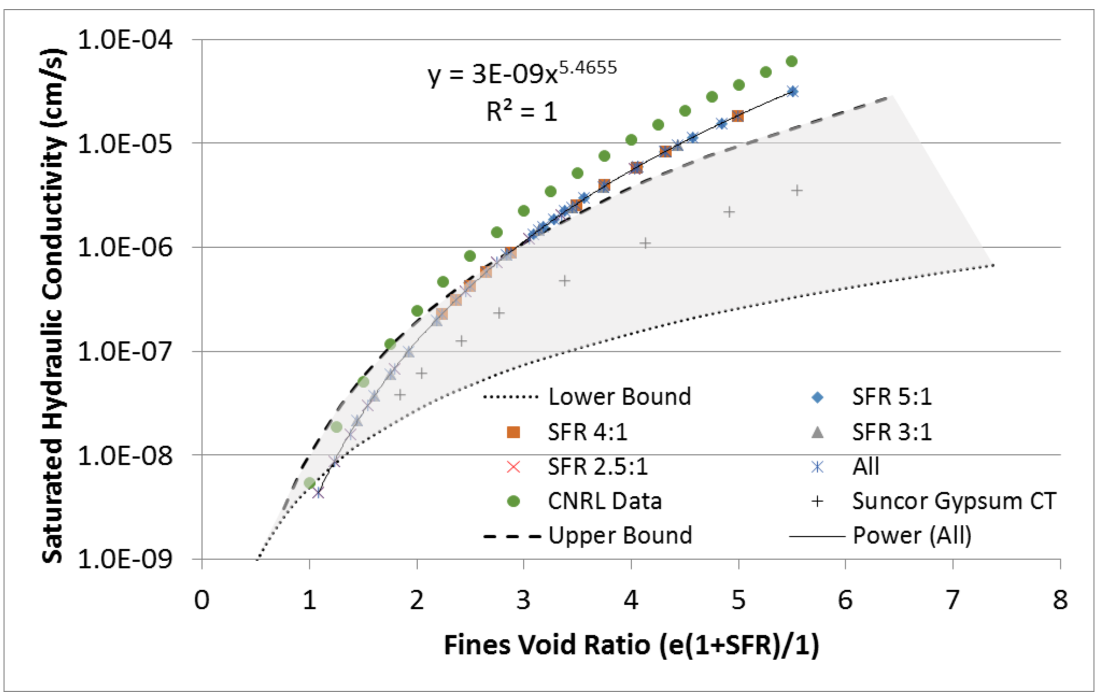


Figure 7.15. Syncrude CT saturated hydraulic conductivity as a function of fines void ratio.

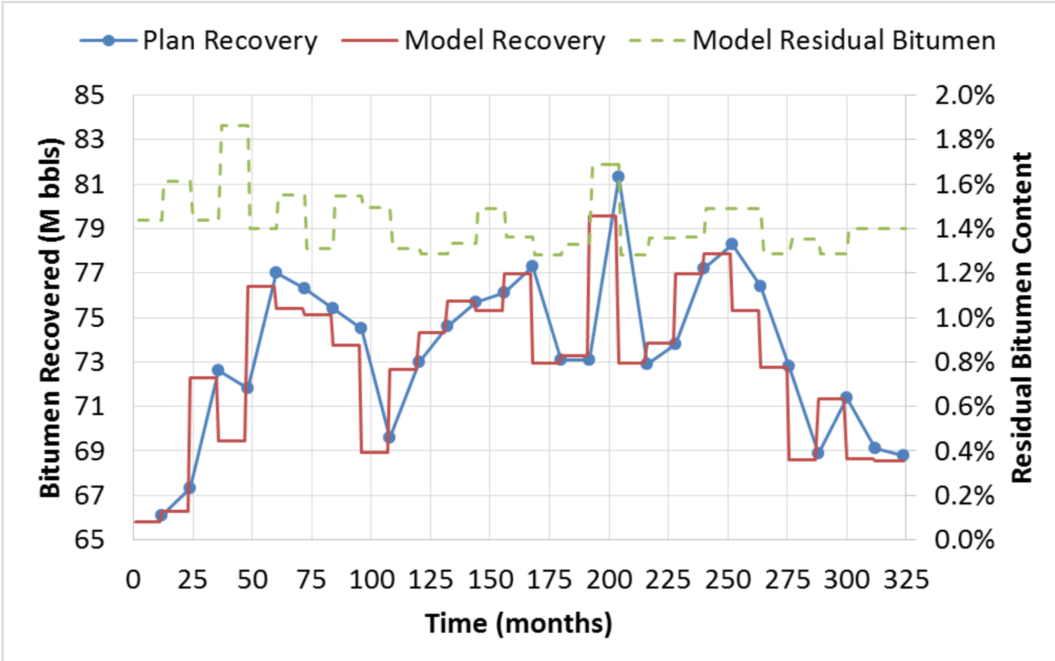


Figure 7.16. Bitumen recovery for preliminary CT simulation CT-1.

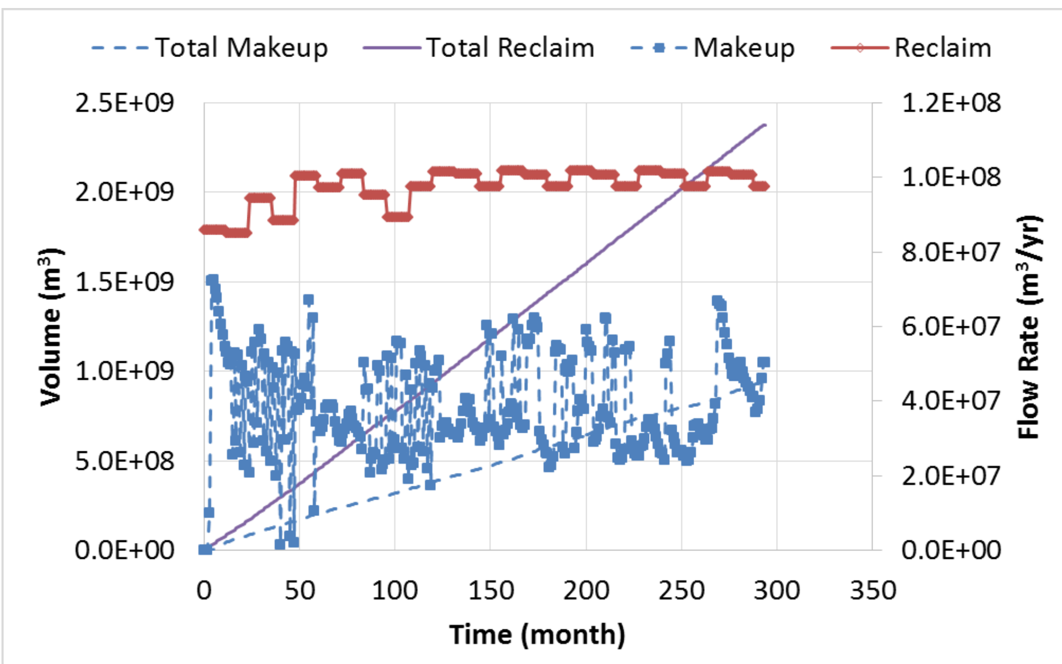


Figure 7.17. Reclaim water and makeup water requirements for preliminary CT simulation CT-1.

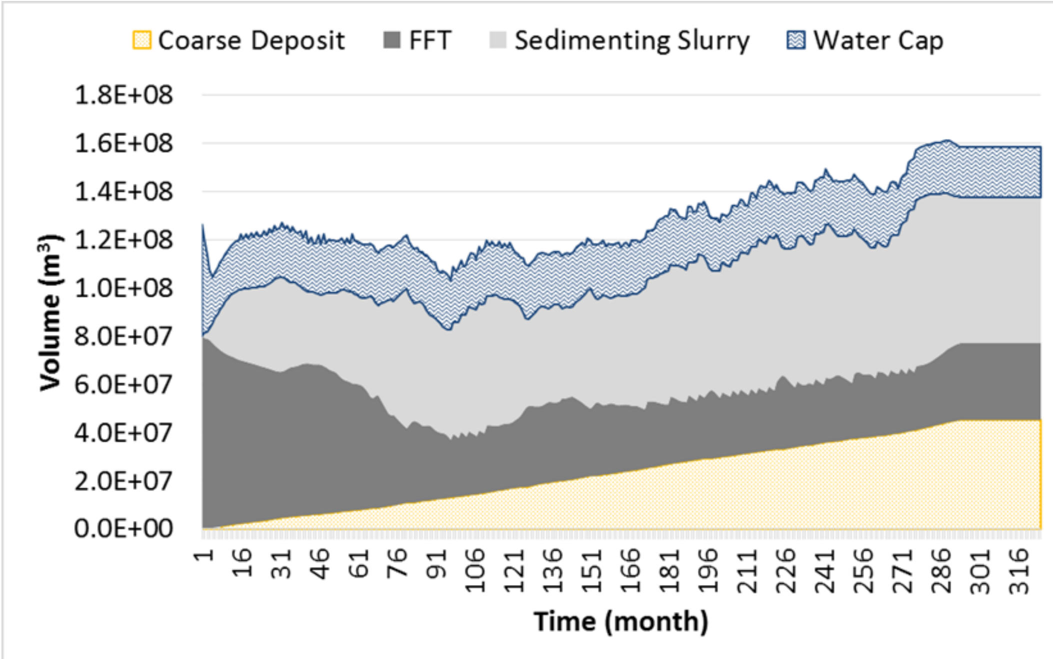


Figure 7.18. ETF volume summary for preliminary CT Simulation CT-1.

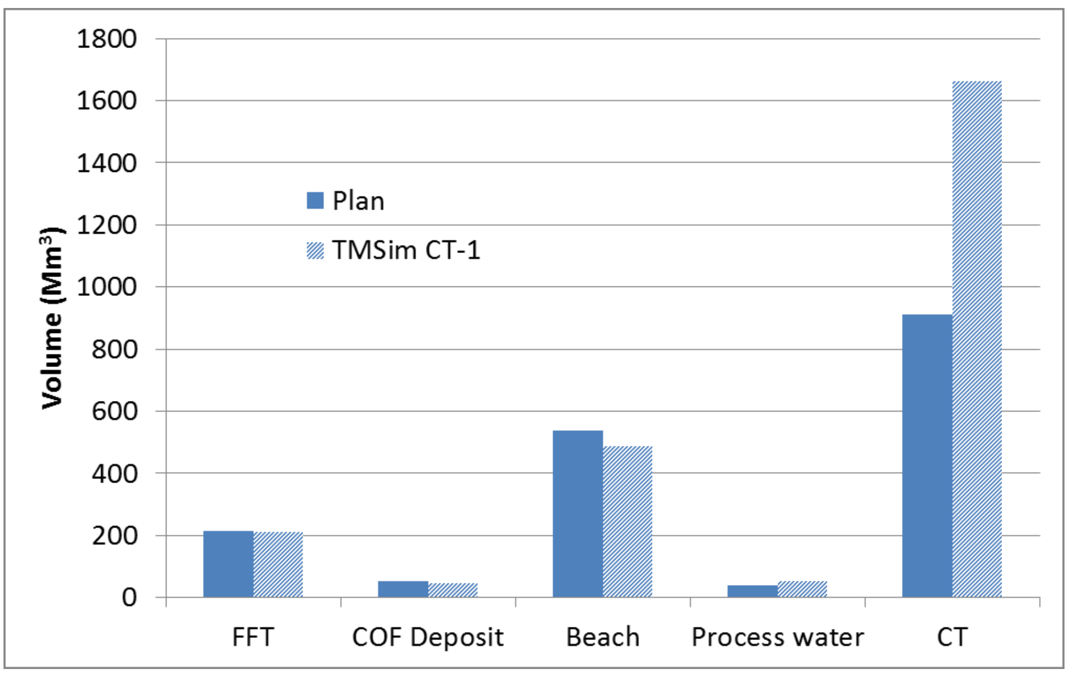


Figure 7.19. Volume comparison of CT-1 with Syncrude Aurora tailings plan.

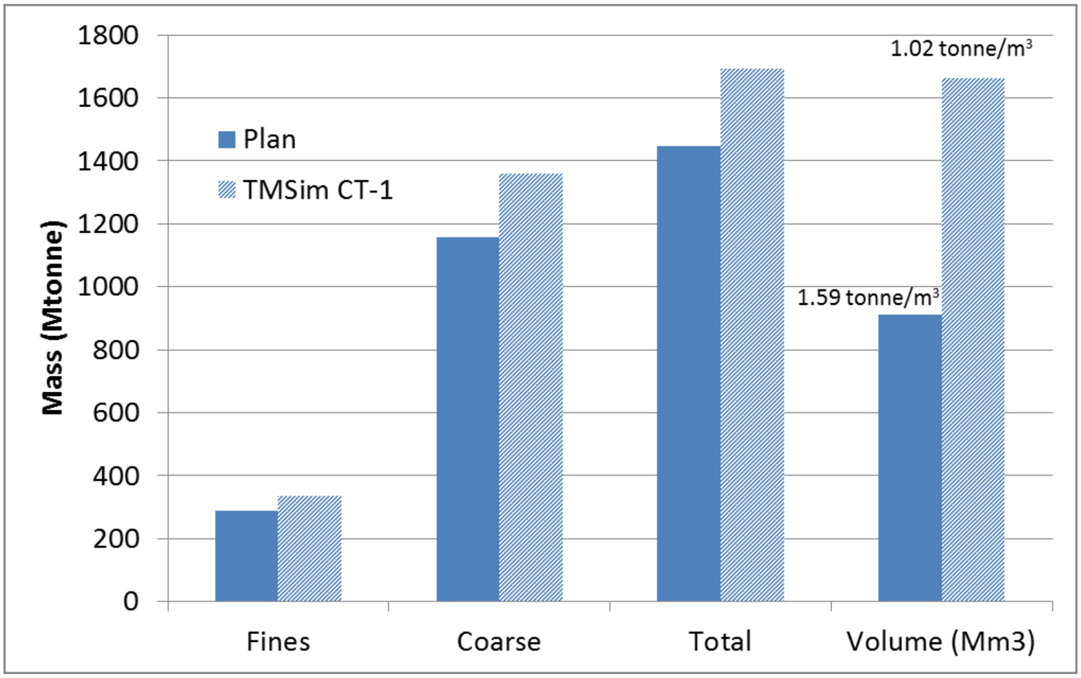


Figure 7.20. Mass comparison of CT deposit with Syncrude Aurora tailings plan.

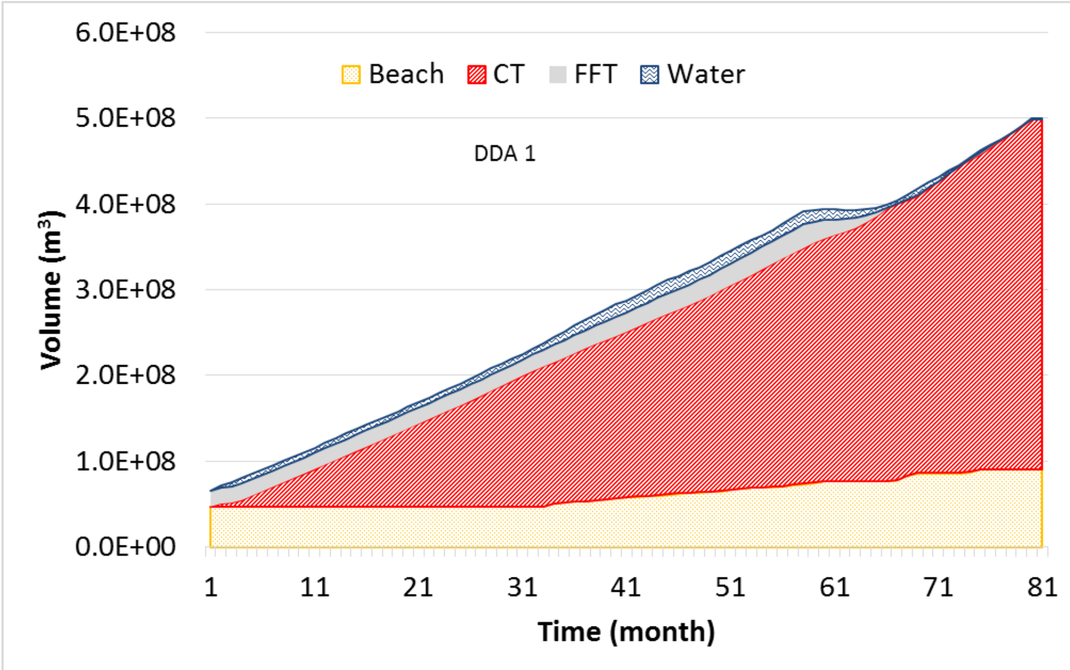


Figure 7.21. Volume summary for DDA 1 (CT-1).

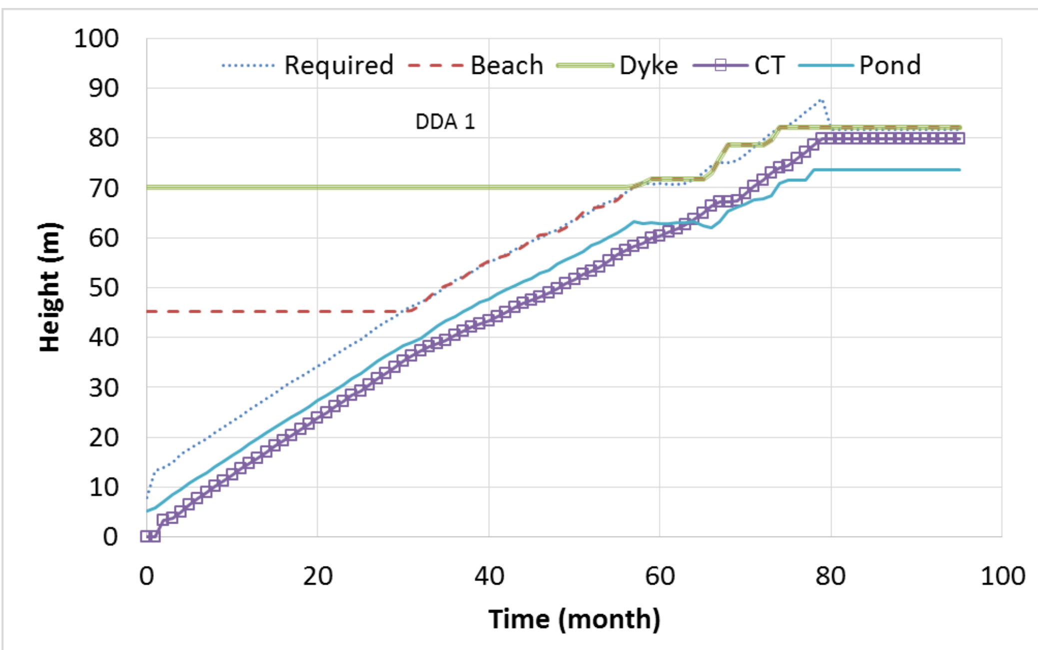


Figure 7.22. Deposit height summary for DDA 1 (CT-1).

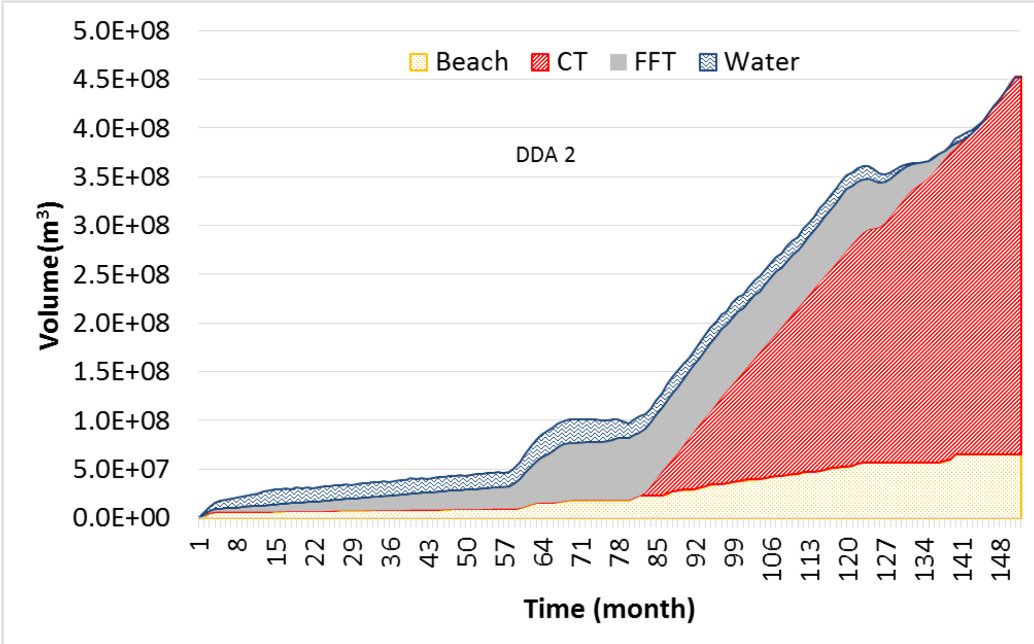


Figure 7.23. Deposit volume summary for DDA 2 (CT-1).

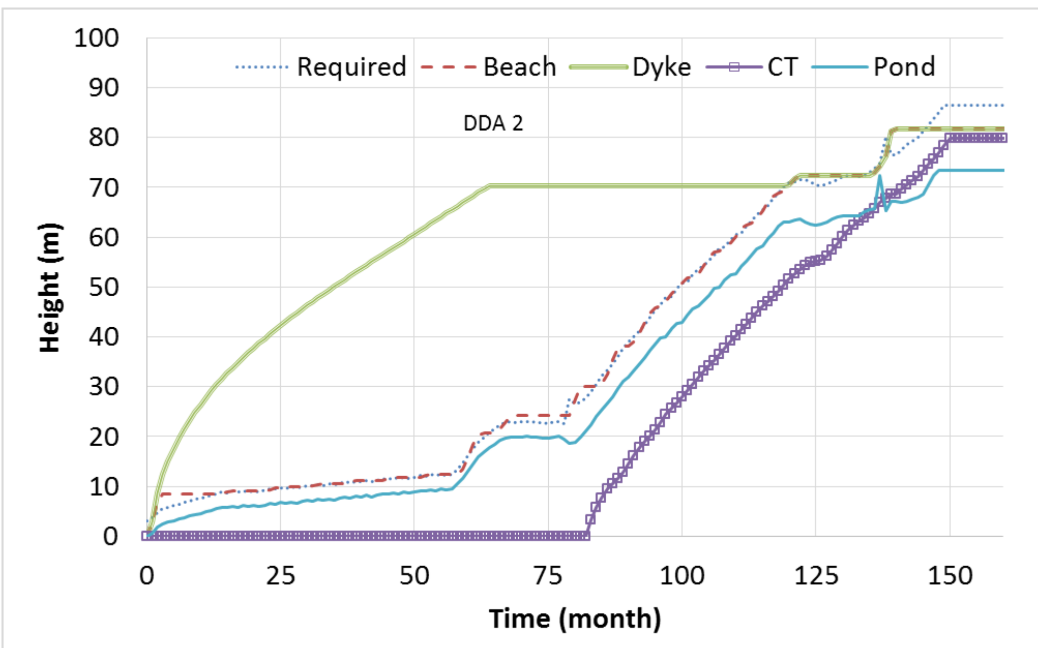


Figure 7.24. Deposit height summary for DDA 2 (CT-1).

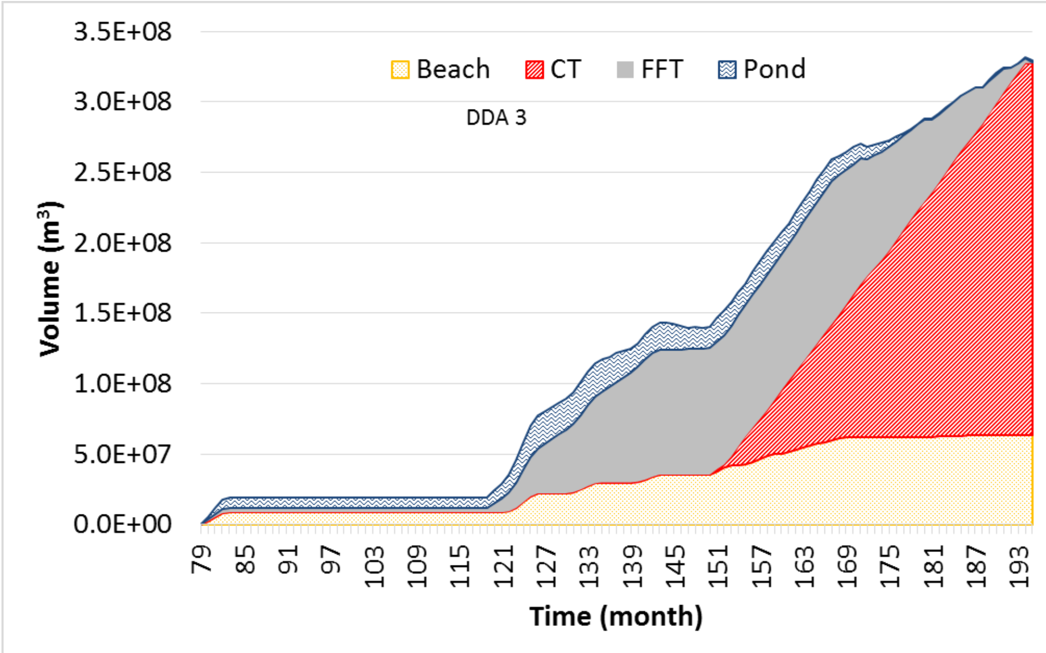


Figure 7.25. Deposit volume summary for DDA 3 (CT-1)

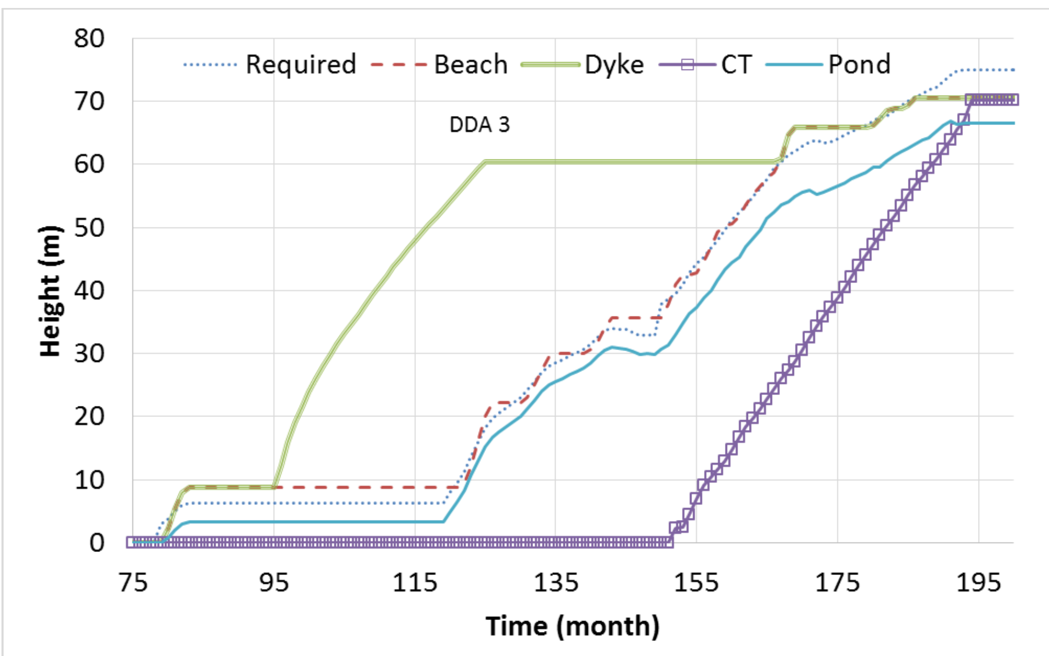


Figure 7.26. Deposit height summary for DDA 3 (CT-1)

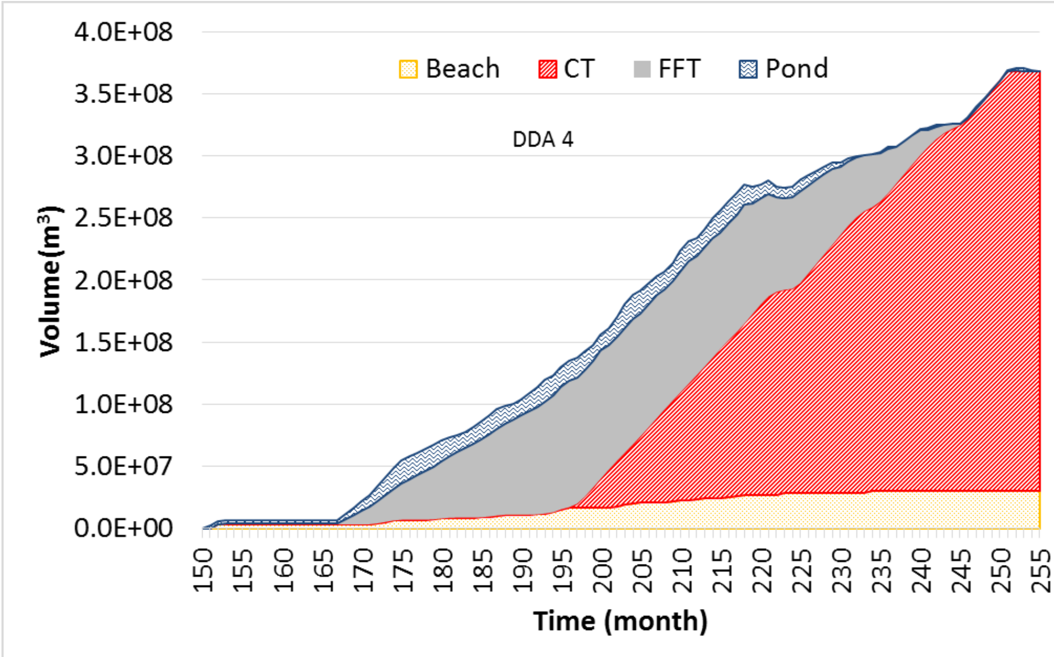


Figure 7.27. Deposit volume summary for DDA 4 (CT-1).

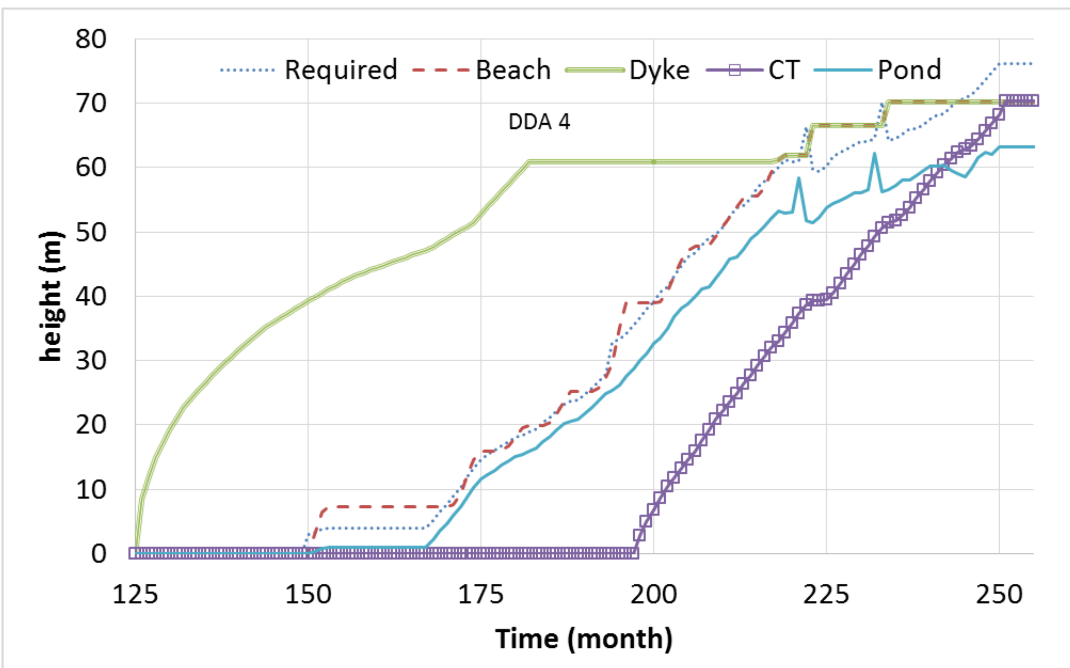


Figure 7.28. Deposit height summary for DDA 4 (CT-1).

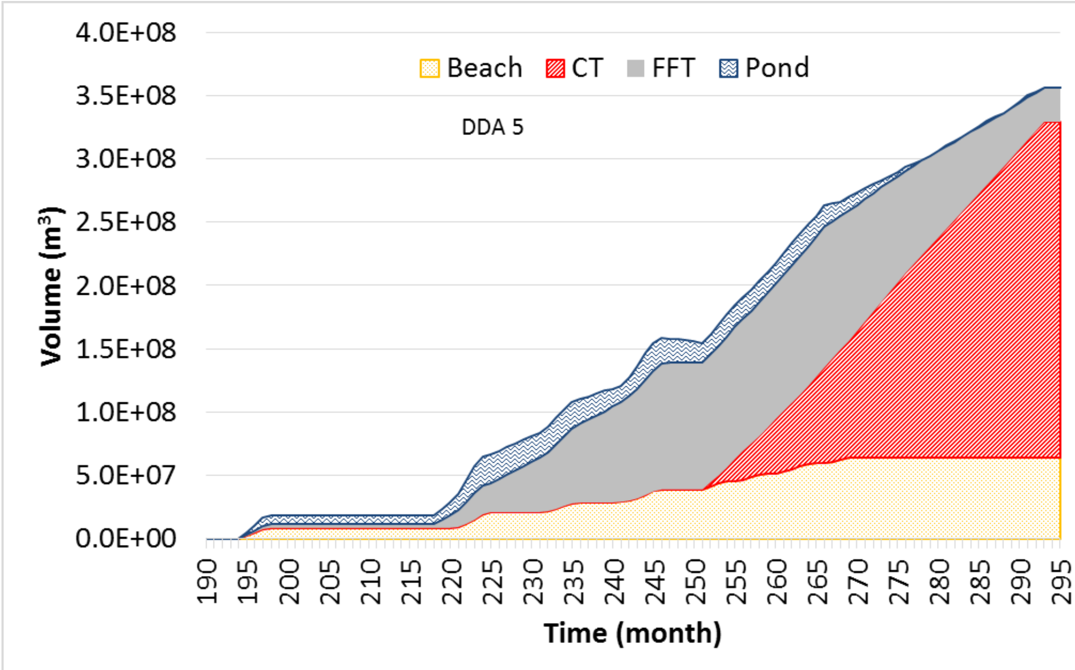


Figure 7.29. Deposit volume summary for DDA 5 (CT-1).

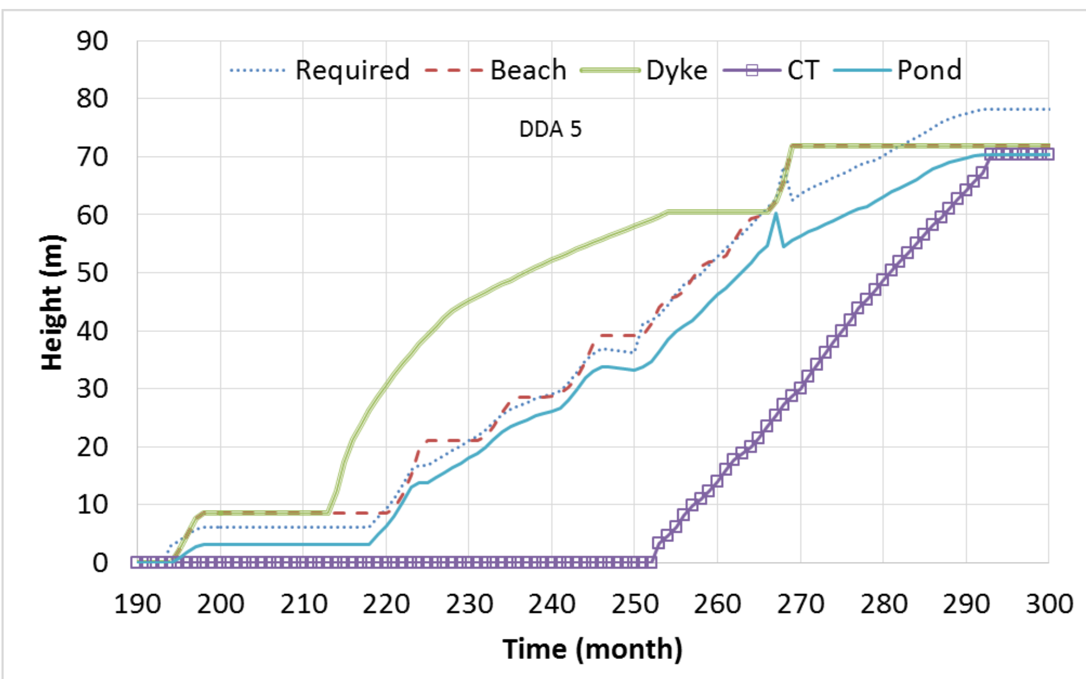


Figure 7.30. Deposit height summary for DDA 5 (CT-1).

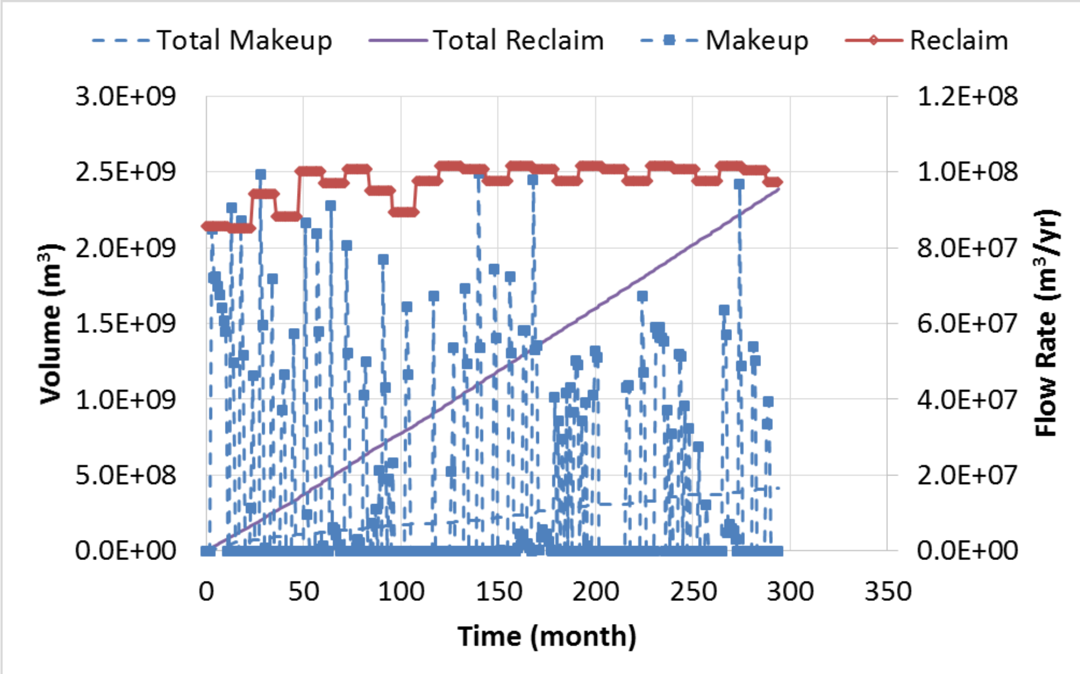


Figure 7.31. Reclaim water and makeup water requirements for preliminary CT simulation CT-2.

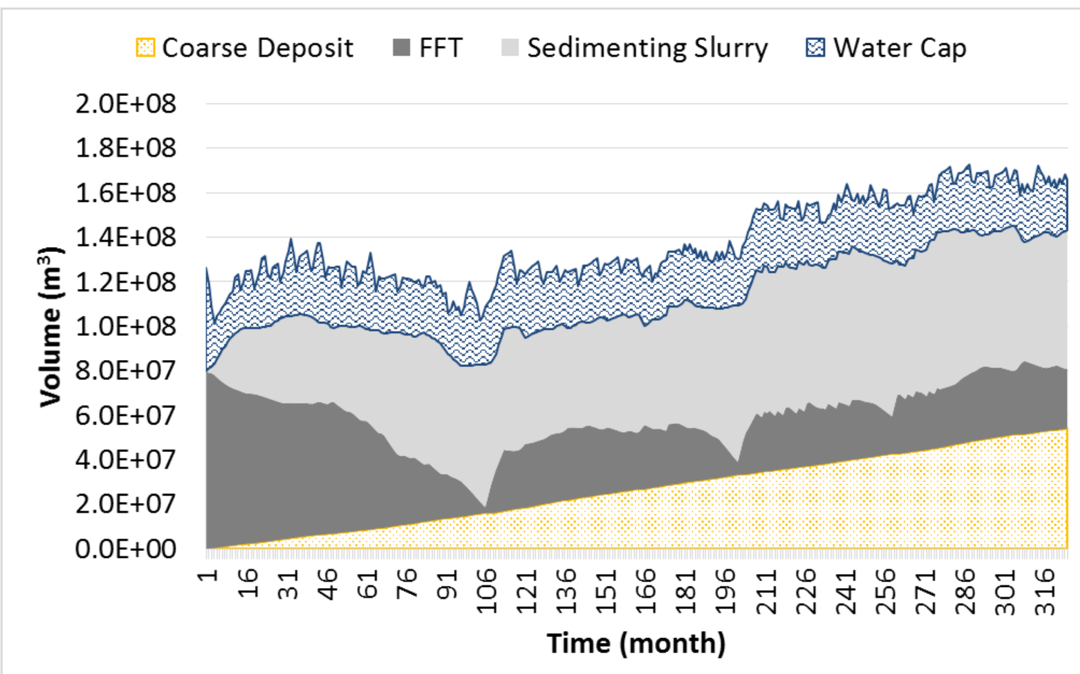


Figure 7.32. ETF volume summary for preliminary CT Simulation CT-2

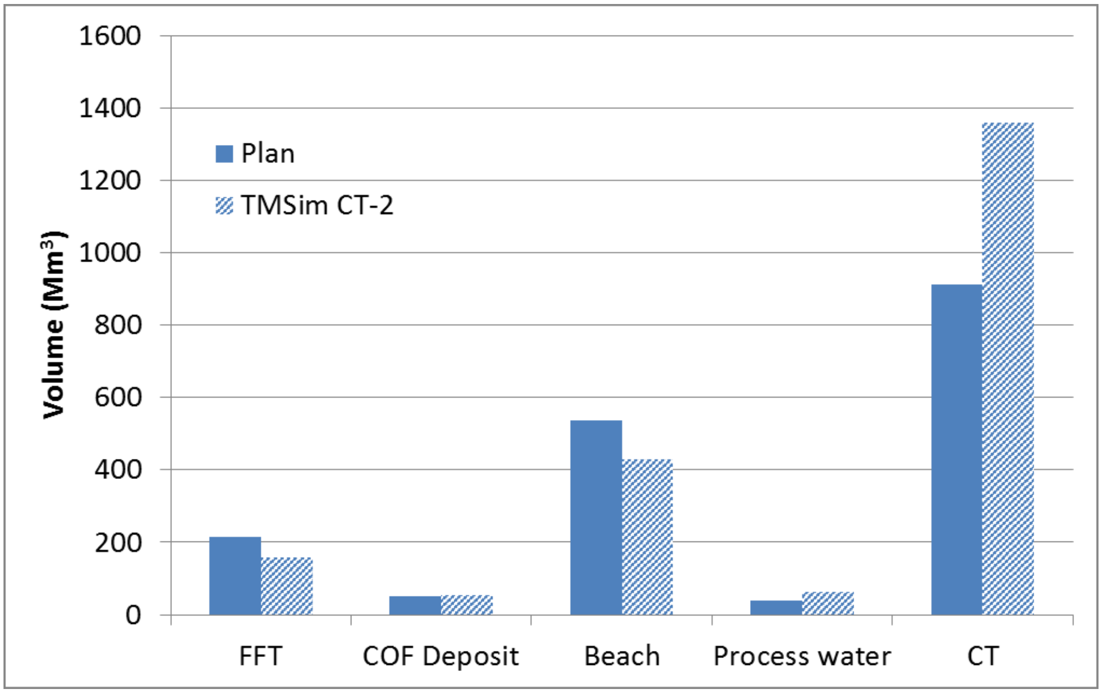


Figure 7.33. Volume comparison of CT-2 with Syncrude Aurora tailings plan

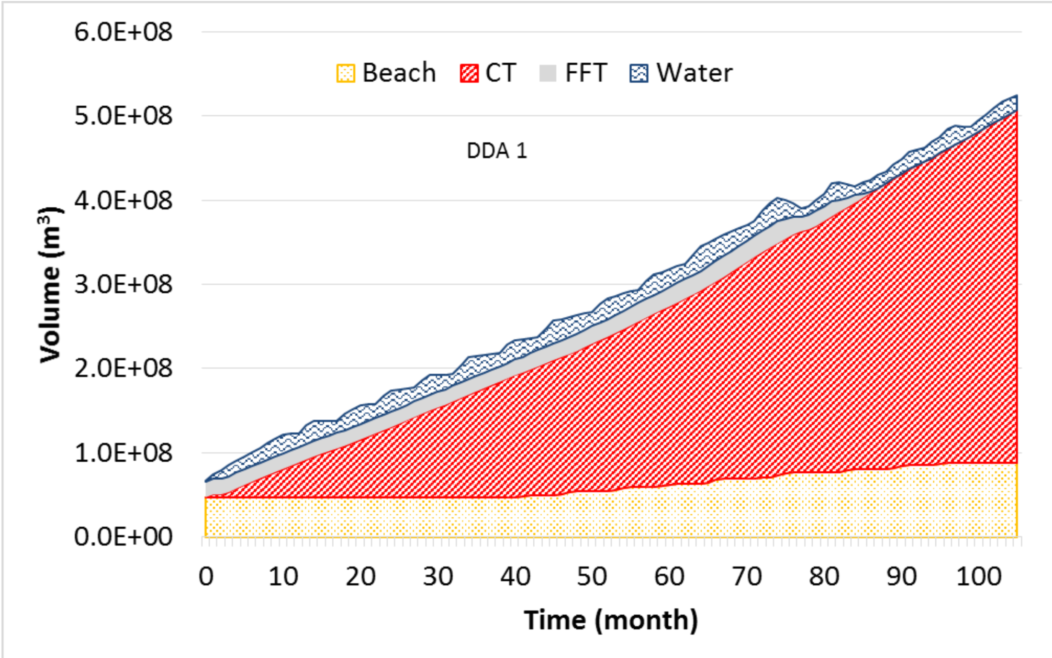


Figure 7.34. Deposit volume summary for DDA 1 (CT-2).

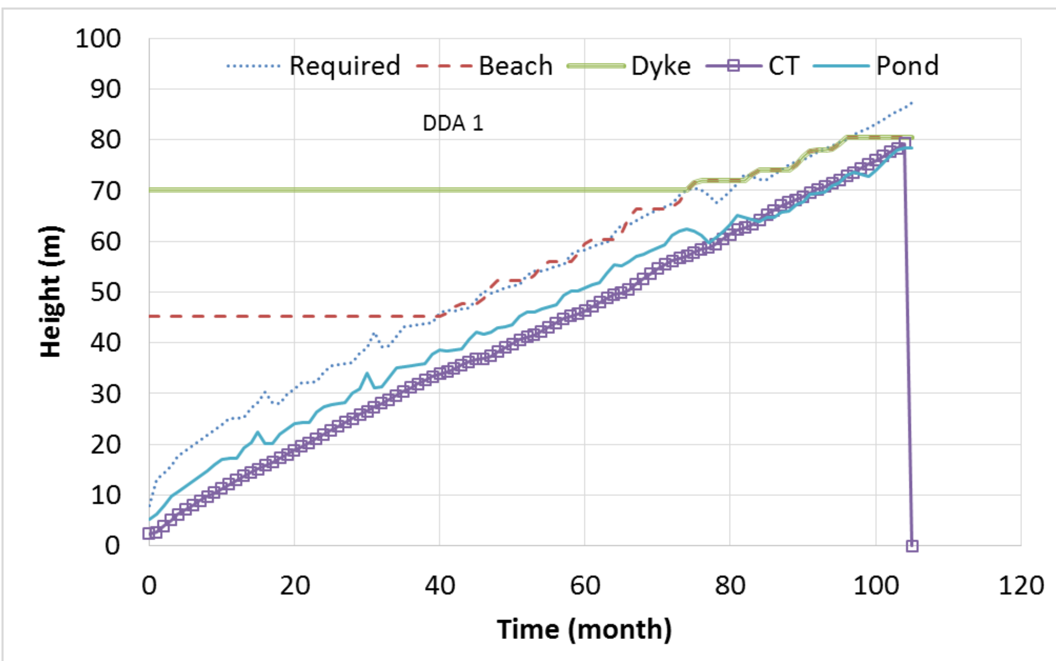


Figure 7.35. Deposit height summary for DDA 1 (CT-2).

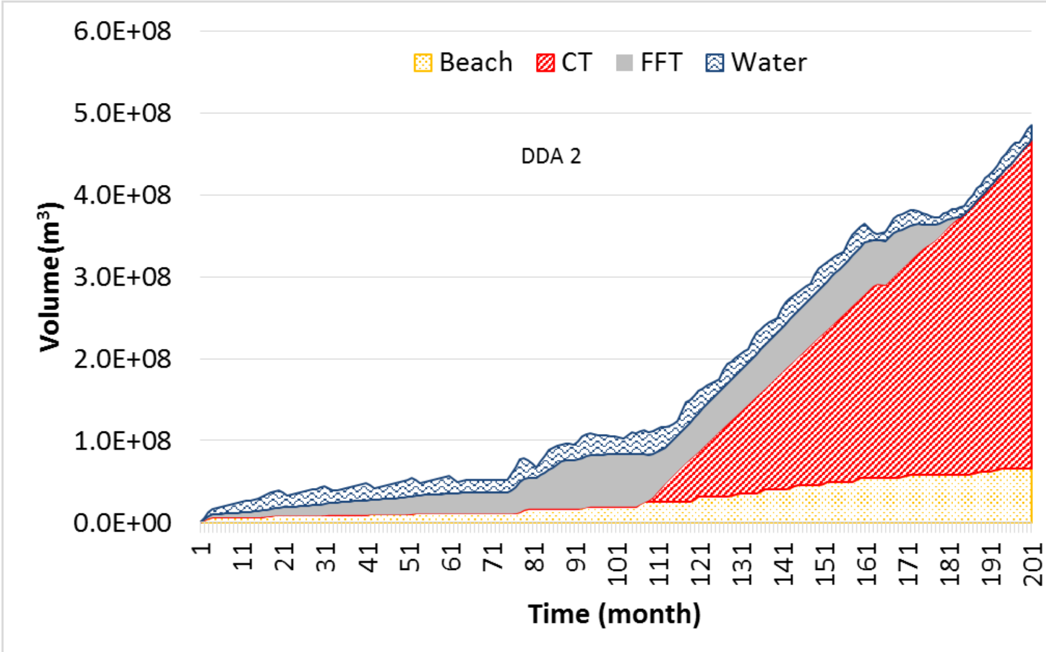


Figure 7.36. Deposit volume summary for DDA 2 (CT-2).

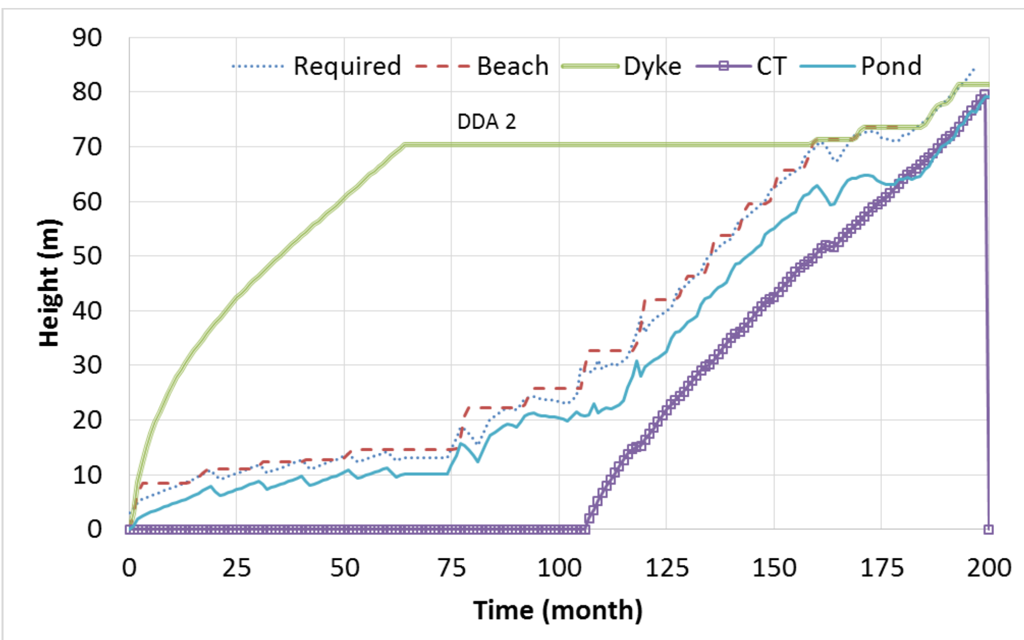


Figure 7.37. Deposit height summary for DDA 2 (CT-2).

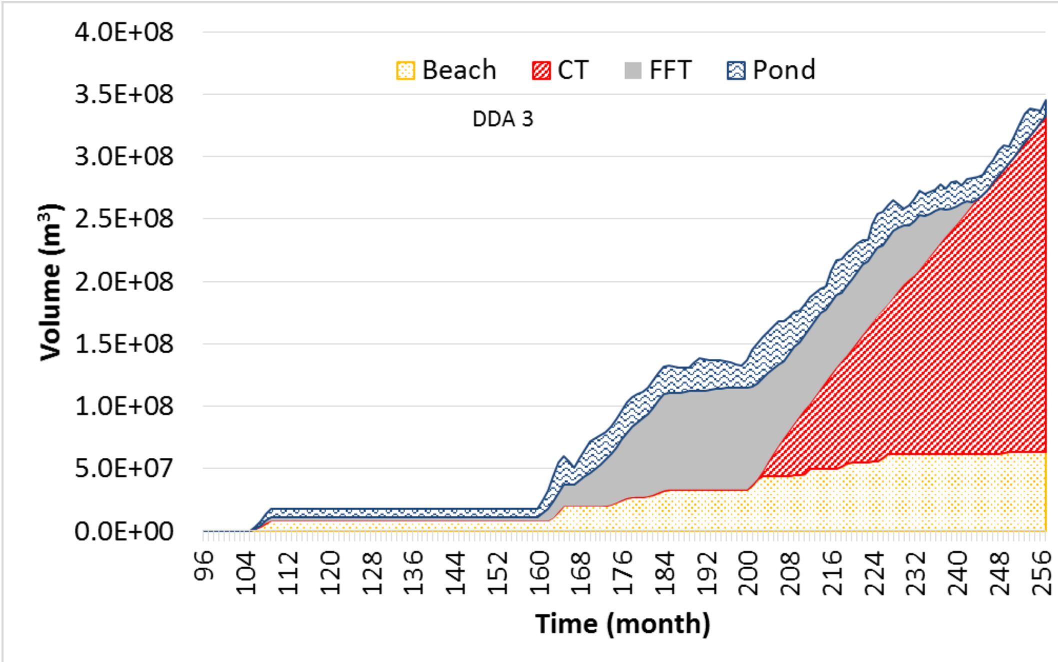


Figure 7.38. Deposit volume summary for DDA 3 (CT-2).

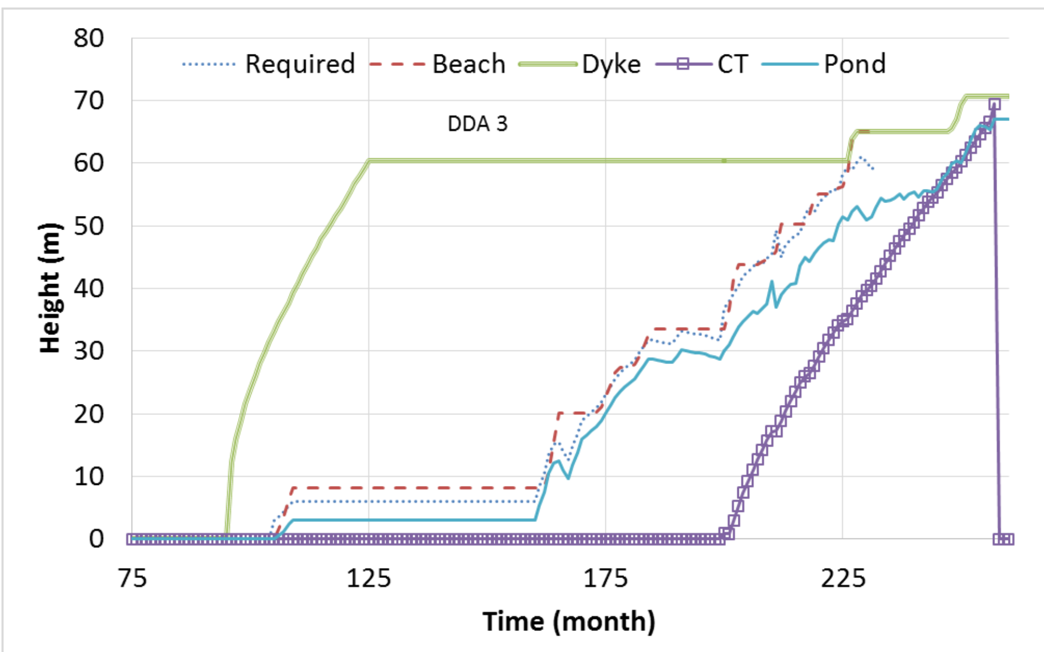


Figure 7.39. Deposit height summary for DDA 3 (CT-2).

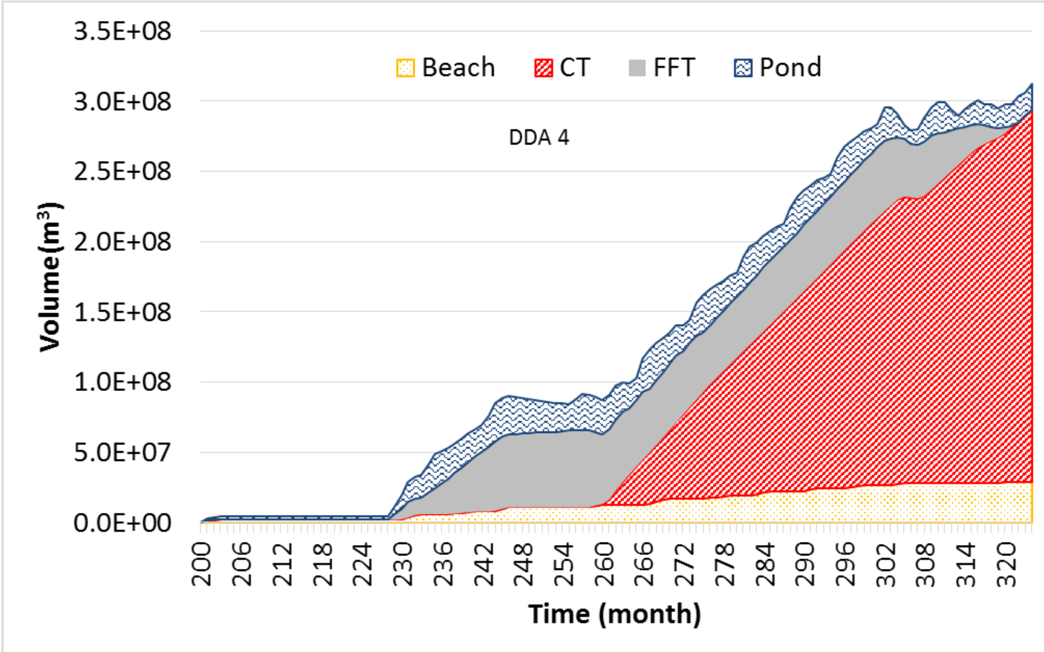


Figure 7.40. Deposit volume summary for DDA 4 (CT-2).

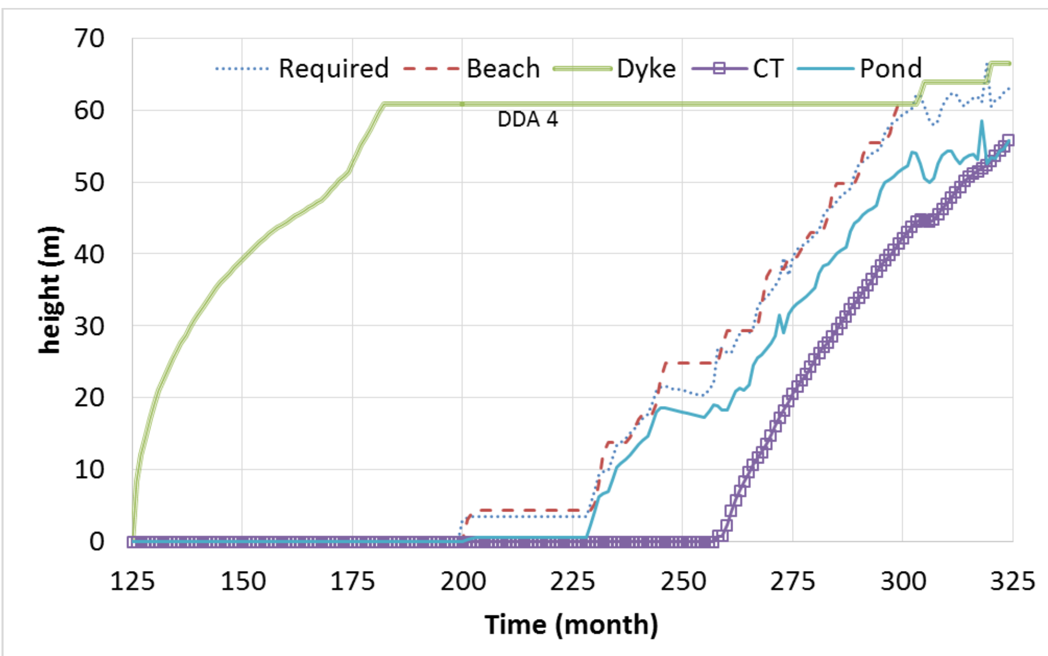


Figure 7.41. Deposit height summary for DDA 4 (CT-2).

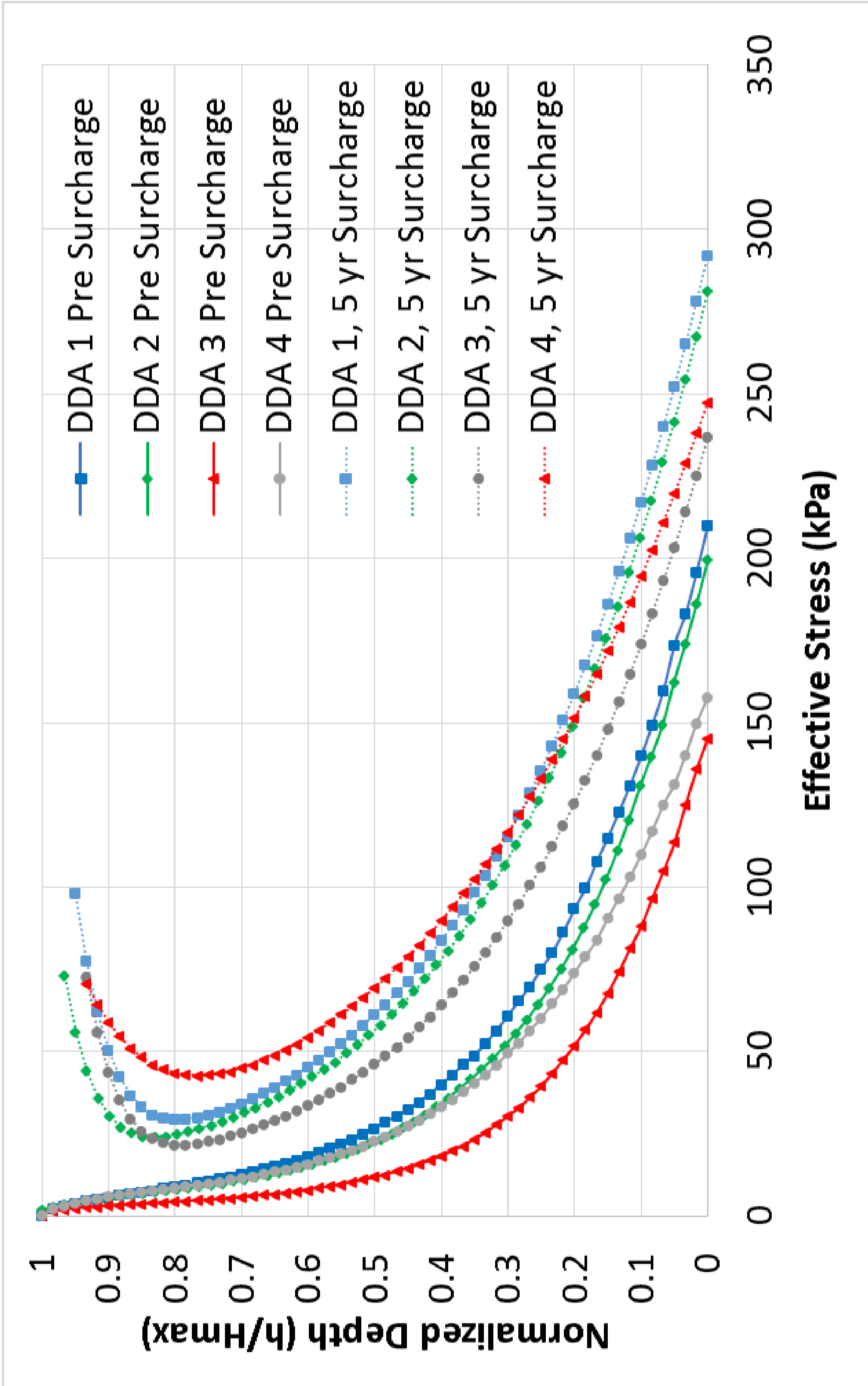


Figure 7.42. Effective stress profiles for CT-2.

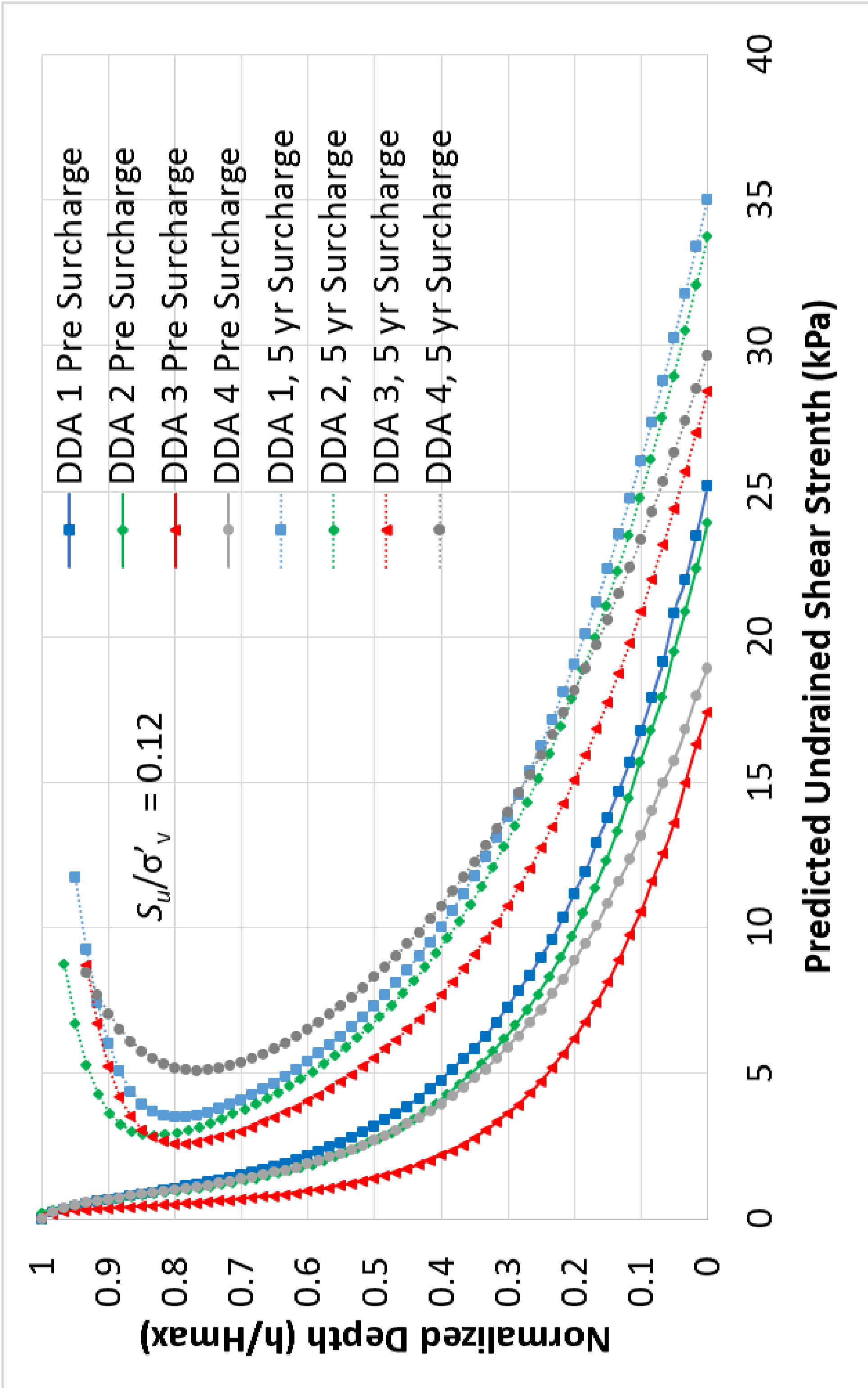


Figure 7.43. Predicted undrained strength profiles for CT-2.

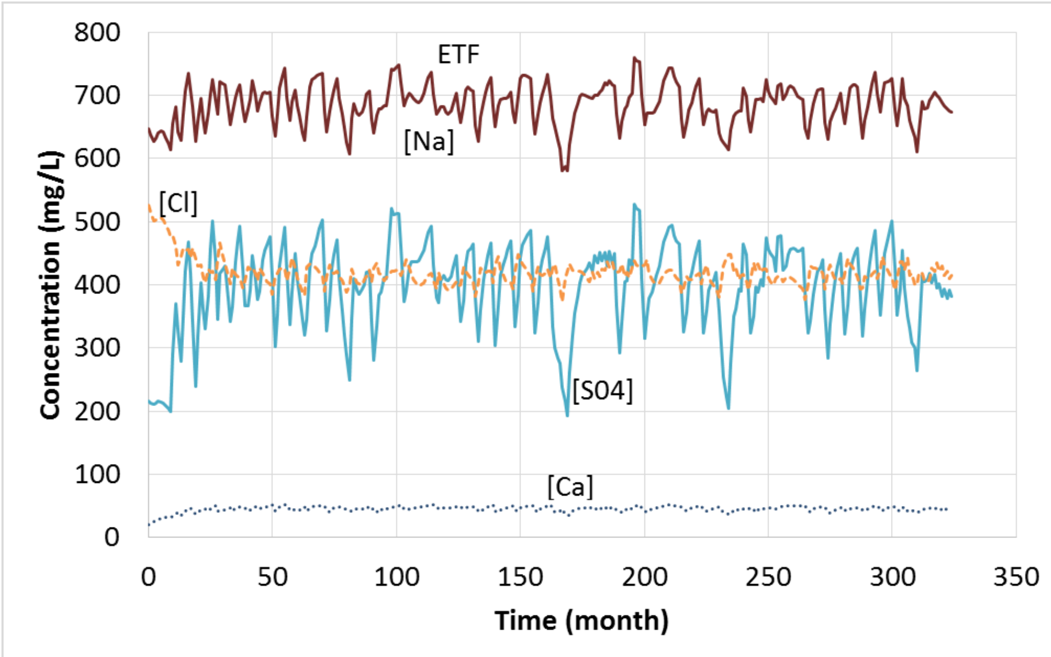


Figure 7.44. Chemical species concentration in the ETF.

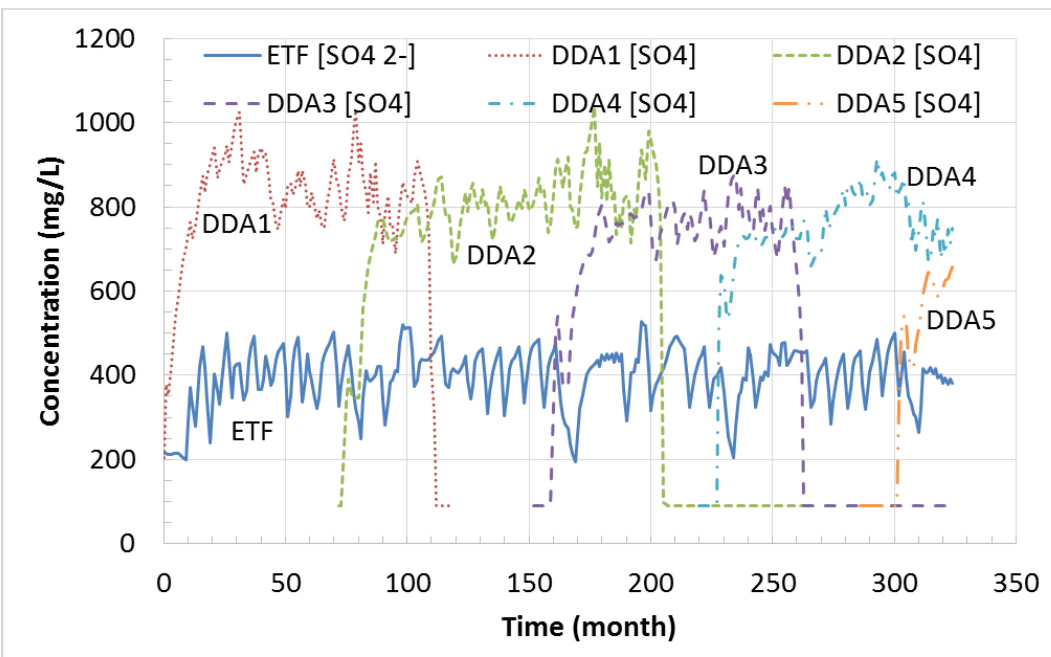


Figure 7.45. Comparison of the S04 concentration in the DDAs and ETF.

7.8 REFERENCES

Allen, E. 2008. Process water treatment in Canada's oil sands industry: I. Target pollutants and treatment objectives. Journal of Environmental Science, **7**: 123-138.

CNRL, 2010. Horizon Tailings Management Plan. Revision 2. November 2010. 90 pages

Environment Canada (EC). 2014. National Climate Data and Information Archive. Accessed July 28, 2014, http://climate.weather.gc.ca/climate_normals/index_e.html

Fitton, T. 2007. Tailings beach slope prediction. PhD Thesis, School of Civil, Environmental and Chemical Engineering, RMIT University, Melbourne, Australia.

Jeeravipoolvarn, S. 2005. Compression Behaviour of Thixotropic Oil Sands Tailings. M.S.c. thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, AB.

Jeeravipoolvarn, S. 2010. Geotechnical behavior of in-line thickened oil sands tailings. PhD Thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, Alberta. pp. 431.

Jeeravipoolvarn, S., Scott, J.D., and Chalaturnyk, R.J. 2009. 10 m standpipe tests on oil sands tailings: long term experimental results and prediction. Canadian Geotechnical Journal, **46**:875-888.

Kupper, A. 1991. Design of Hydraulic Fill. PhD thesis. University of Alberta. Edmonton, Alberta, Canada.

MacKinnon, M.D., Matthews, J.G., Shaw, W.H., and Cuddy, R.G. 2001. Water quality issues associated with Composite Tailings (CT) technology for managing oil sand tailings. International Journal of Surface Mining, Reclamation, and Environment, **15**(4): 235-256.

Matthews, J.G., Shaw, W.H., MacKinnon, M.D., and Cuddy, R.G. 2002. Development of composite tailings technology at Syncrude. International Journal of Surface Mining, Reclamation, and Environment, **16**(1): 24-39.

Miller, W.G., Scott, J.D. and Sego, D.C., 2011. Effect of extraction water chemistry on the consolidation of oil sands fine tailings. CIM Journal **1**(2): 113-129.

Qiu, Y., 2000. Optimum deposition for sub-aerial tailings disposal. PhD thesis, University of Alberta.

Reeves, B. 1996. Environmental Impact Assessment for the Syncrude Canada Limited Aurora Mine. BOVAR Environmental, 1996.

Scott, J.D. 2014. Personal communication.

Sobkowicz, J.C. 2012. The oil sands tailings technology roadmap and action plan: Introduction and key recommendations. Third International Oil Sands Tailings Conference. Edmonton, Alberta. December 2-5, 2012. Pp. 13-23.

Sobkowicz, J.C. and Morgenstern, N.R. 2009. A geotechnical perspective on oil sands tailings. 13th International Conference on Tailings and Mine Waste, Banff, Alberta, 1-4 November 2009. University of Alberta, pp. xvii-xli.

Song, Q., O'Kane, M., Dhadli, N., and Matthews, J. 2011. Deposition thickness and evaporative drying for oil sands tailings in northern Alberta. Mine Closure 2011. Proceedings of the Sixth International Conference on Mine Closure. Lake Louise, Alberta, Canada, 18-21 September 2011. pp. 373-382.

Suthaker, N. N. 1995. Geotechnics of Oil Sand Fine Tailings, Ph.D. thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, AB.

Suthaker, N.N. and Scott, J.D. 1996. Measurement of Hydraulic Conductivity in Oil Sand Tailings Slurries. Canadian Geotechnical Journal, **33**(4): 642-653.

Syncrude Canada Ltd. 1995. Non –segregating tailings: 1995 NST field demonstration summary report. Syncrude Canada ltd. research and development. 201 pp.

Syncrude Canada Ltd. 2010a. Directive 074 Baseline Survey for Fluid Deposits Syncrude Mildred Lake and Aurora North, Year: Baseline (2010). Report submitted to Energy Resources Conservation Board, September 30, 2010. 53 pp.

Syncrude Canada Ltd. 2010b. 2009 Groundwater Monitoring Report. Submitted to Alberta Environment, March 30, 2010.

Syncrude Canada Ltd. 2012. 2012 Annual Tailings Plan Submission Syncrude Aurora North Mine. Report submitted to Energy Resources Conservation Board, September 28, 2012. 43 pp.

8 ASSESSING CROSS FLOW FILTRATION TECHNOLOGY WITH A TAILINGS MANAGEMENT SIMULATION MODEL

8.1 INTRODUCTION

Tailings management at oil sands mines has evolved from fluid storage in single external impoundments to complex multistage mechanical and chemical dewatering processes and storage in several in-pit and external impoundments. To meet closure and regulatory requirements, oil sands tailings management is moving towards transforming fluid tailings and waste materials into geotechnically stable deposits that fulfil regulatory performance criteria and subsequently reclaimed (Sobkowicz and Morgenstern 2009). As evident in the Tailings Roadmap study (Sobkowicz 2012), there are hundreds of potential tailings dewatering and reclamation technologies. The Tailings Roadmap project ultimately developed nine "technology roadmaps" outlining potential technologies applicable to improving oil sands tailings management practices. This project provided a snapshot of potential technologies and their various stages of development. However, new technologies, processes and applications are constantly being developed and improved. There is a need to evaluate these new tailings management technologies. Several of the technology vendors lack a sufficient understanding of oil sands operations (i.e. technology exploited from another industry) or the technologies are conceptual or bench scale and require further research and development. Therefore, their potential application to and impact on the oil sands mines are not fully understood.

A dynamics simulation model, TMSim was developed as potential tool to aid in the evaluation of these tailings management technologies and processes. The TMSim model will provide guidance to the technology developers, mine operators, and regulators on strengths and limitations of these technologies. TMSim was developed to incorporate mine plan data, various stages of dewatering including classification, pre- and post- deposition dewatering, and an impoundment material balance including tailings, process water, construction material and capping materials.

The objective of this chapter is to demonstrate the application of the TMSim simulation tool to evaluate a potential new dewatering technology. All data utilized in the following simulations were collected from publically available sources of information. The Syncrude Canada Ltd. (Syncrude) Aurora North (Aurora) mine was chosen as the model site. The TMSim model will simulate a “new technology”, cross flow filtration (CFF) dewatering, as presented by Beier and Sego (2008) and Zhang (2010). The performance of the CFF-tailings plan will be compared with the existing CT technology simulation results from Chapter 7.

8.2 OIL SANDS MODEL DATA

The model oil sands mine plan developed in Chapter 7 was also utilized for the CFF technology simulations. The model data set is based on information from the Syncrude Aurora mine and tailings management data obtained from the Aurora North Environmental Impact Assessment report (Reeves 1996) the 2012 Annual Tailings Plan report (Syncrude 2012) and the 2010 Baseline Survey for Fluid Deposits (Syncrude 2010). Adaptations and assumptions to the model data set are detailed in Appendix 4. Below is a brief summary of the mine plan detailed in Chapter 7.

The Syncrude Aurora North oil sands mine (started in 2000) uses truck and shovel surface mining technology. A warm water extraction process is utilized to separate bitumen from the ore. Bitumen froth produced at Aurora is pipelined to another Syncrude mine site, Mildred Lake, for further processing and upgrading. The Aurora mine excavates approximately 95.5 Mm^3 of ore and mine waste overburden per year resulting in approximately 200,000 bbl/day of bitumen production. The average ratio of excavated ore to overburden and mine waste is 1.59. The Aurora mine pit covers approximately 48.8 km^2. Based on the total mined volume of ore and overburden from 2000 to 2012 estimated at 800 Mm^3 (Appendix 3) and 2578 Mm^3 calculated for 2013 to 2039, an average pit depth was determined to be 69.2 m (Appendix 3). Based on the average ore to overburden ratio of 1.59, the overburden thickness is approximately 25.7 m and the ore is 43.5 m thick. The mine pit slopes in overburden zones are 3:1 and 2:1 in the ore body formation.

The primary extraction process at Aurora produces a tailings stream consisting of sand, silts, clay, residual bitumen and water known as whole tailings or SCT. A small stream of floatation tailings is also produced during the extraction process. The floatation tailings are incorporated and managed with the whole tailings. An ETF (Aurora settling basin) contains FFT created since the mine started. The ASB currently contains 79.9 Mm^3 of FFT and 46.5 Mm^3 of process water (Syncrude 2012) and is at its maximum construction elevation. The digitized pond surface area is 6.19 km^2. The maximum planned storage volume of the ASB is 129.8 Mm^3 (based on maximum volume calculated from Syncrude 2012). The mine pit also contains 18.6 Mm^3 of FFT, 0.7 Mm^3 of water and 47.3 Mm^3 of SCT.

Properties of several samples of fine tailings originating from Syncrude are presented in Table 8.1. Syncrude tailings sample properties COF is cyclone overflow from a 2008 sampling program (Jeeravipoolvarn 2010), MFT is from Scott (2014) and represents an aged FFT sample, Ore A (marine) and Ore B (non-marine) represent young fine tailings derived from a caustic extraction process (Miller et al. 2011), and the 10 m sample represents the MFT stored in a 10 m column for 30 years at the University of Alberta (Jeeravipoolvarn et al. 2009). The FFT from the ASB can be expected to have fines contents greater than 90%, and clay contents from 30 to 50%.

8.3 CROSS FLOW FILTRATION DEWATERING

A potential alternative tailings management technology for an oil sands mining operation is crossflow filtration (Beier and Sego 2008). Crossflow filtration (CFF) is a pressure driven filtration process that can be used for dewatering slurries of fine particles and can offer improvements over conventional filtration. In CFF, the slurry flows parallel to the filtration membrane developing a cake on the filtration surface (i.e. a porous pipe). However, due to shear of the flowing slurry, the build up of cake will reach an equilibrium thus maintaining a relatively constant filtration rate (Figure 8.1). CFF offers an opportunity to deposit tailings without inducing segregation of the fines from sand, therefore preventing further accumulation of FFT. Early testing also indicates that no chemical additives were needed to achieve filtration, therefore, release water from the CCF tailings will be similar to the process water from extraction. It also provides immediate recycle of process water to the extraction plant. This will reduce the energy costs required for heating process water resulting in Green House Gas reduction associated with the extraction process.

CFF was evaluated in the Tailings Roadmap project and was prioritized as a medium level technology, on par with CT and NST technologies (Boswell et al 2012). CFF has only been applied at the laboratory and bench scale (Beier et al. 2008) for oil sand tailings applications. Therefore it will serve as a sufficient candidate to demonstrate the TMSim modeling capability of a new, developing technology.

The CFF process can be implemented in an oil sands mine to dewater extraction tailings or whole tailings prior to deposition, thus negating the formation and subsequent build up of FFT. Sufficient water must be removed from the extraction tailings during CFF so upon deposition, the tailings do not segregate. For typical extraction tailings at 55% C_w, approximately 50% of the water must be removed to increase the C_w to at least 70% to prevent segregation (Beier and Sego 2008). The amount of water to be removed from a unit volume of tailings is included as Figure 8.2. Upon dewatering to 70% or greater C_w, the CFF-tailings could then be deposited as stacks within the mined out pit. Sufficient overburden material should be available to provide in pit containment dykes for the CFF-tailings. Existing FFT in the ASB maybe dewatered using a separate dewatering technology such as freeze-thaw dewatering. Alternatively, the FFT could be re-incorporated (spiked) into the extraction tailings prior to CFF dewatering. This would provide an opportunity to consume the residual FFT. Currently, there are approximately 100 Mm^3 of FFT at the Aurora site. With approximately 27 years of mine life remaining, the chosen FFT dewatering process would have to dewater approximately 4 Mm^3 of FFT annually to deplete the current stockpile. At the end of mining, any process water stored within ASB will be transferred to the mine pit forming an end pit lake.

8.3.1 CFF Dewatering Model

The following CFF dewatering model is based on the preliminary CFF studies by Beier and Sego (2008). They used a mixture of kaolinite and coarse oil sand beach sand to develop a surrogate whole tailings with a C_w of 55% and a 15% F%. For reference, the average F% of the whole tailings stream at Aurora is 18.9 % +/- 5%. The resulting tailings stream was then pumped through a 3 m section of porous filter pipe. At an in-pipe slurry velocity of 1.7 m/s, Beier and Sego (2008) achieved an average filtration rate of 0.008 L/s/m^2 of filter surface. Zhang (2010) achieved a similar filtration rate (0.005 L/s/m^2) with a mixture of oil sand fine tailings (FFT) and beach sand under the same operating conditions. Therefore, results from surrogate oil sand tailings stream are considered suitable for the CFF design. Both Beier and Sego (2008) and Zhang (2010) also found that the filtration rate did not diminish as the C_w of the slurry was increased. Therefore, the CFF design can assume a constant filtration rate, regardless of the slurry C_w. The CFF will however, depend on the cake properties, specifically the resistance to filtration (i.e. saturated hydraulic conductivity).

Filtration experiments on oil sand tailings were also conducted by Xu et al. (2008) to assess the impact of F% and flocculation on the filtration rate for a vacuum filtration process. Tailings slurries with F% ranging from 4.3% to 83.3% were filtered in a bench scale filtration apparatus. Using the filtration Equation 8.1, the specific resistance to filtration, r (m/kg), can be determined as a function of F% and flocculation condition (Figure 8.3).

[8.1] $$\frac{t}{V} = \frac{\mu r \omega}{2P_f A f^{\,2}} V + \frac{\mu L_m}{P_f A f}.$$

Where t is time (s), V is volume (m^3), μ is viscosity of the filtrate (Pa s), ω is the mass of solids cake formed per unit of filtrate volume passed (kg/m^3), P_f is pressure (Pa), Af is the filter area (m^2), and L_m is the equivalent thickness of the filter medium (kg/m^2). The specific resistance from Beier et al (2008) and Zhang (2010) were also plotted on Figure 8.3 for comparison. Beier et al (2008) and the Zhang (2010) data fit within the range of unflocculated tailings Xu et al (2008) data. Therefore, the unflocculated filtration data from Xu et al (2008) may be suitable to represent the CFF filtration conditions at different F%.

Using the Xu et al (2008) relationship between specific resistance to filtration and F% of the slurry, the volume of filtrate for a unit area (1 m^2) and unit time (1 hr) can be calculated from Equation 8.1. The filtrate volumes were calculated assuming a constant pressure (150 kPa), viscosity of filtrate at 0.001 Pa s, L_m (1.26×10^{-8} kg/m^2). The ω and L_m were calculated from the data and figures provided in Xu et al (2008). Using the filtrate volume and time, filter surface area (FSA) and applied pressure, an equivalent saturated hydraulic conductivity, k_{cake}, of the filter cake can be calculated using the Darcy equation (Equation 8.2).

[8.2] $$Q = k_{cake}\frac{\Delta h}{\Delta L}A$$

Where Δh is the applied pressure in m of water and ΔL is the thickness of the filter cake. Assuming the in pipe slurry velocity is similar to the Beier et al (2008) experiments the estimated cake thickness should be 0.005 m. The calculated equivalent cake saturated hydraulic conductivity versus F% is presented on **Error! Reference source not found.**. A power function (Equation 8.3) representing the unflocculated tailings and Equation 8.4 for flocculated tailings data can be used to estimate the equivalent saturated hydraulic conductivity of the CFF cake as a function of F%.

[8.3] $$k_{cake} = 5E^{-7} * F^{-1.42}$$

[8.4] $$K_{cake} = 2E^{-6} * exp^{-0.13*F}$$

According to Devenney (2009), an average of 0.8 m^3 of process water is required per tonne ore processed. Based on the average mining rate of 118 Million tonnes/year, 97 million tonnes of mineral solids will report to the tailings stream

(Syncrude 2012). Assuming a nominal process water loss of 2.5% (Reeves 1996), the average C_w of extraction tailings stream is 51% at a flow rate of 14,700 m^3/hr. To achieve the desired CFF dewatered C_w of 75%, approximately 46 % of the flow needs to be removed as clear water filtrate or approximately 6800 m^3/hr.

Using the k_{cake} relationship for the unflocculated tailings at 15% F%, and an applied filter pressure of 150 kPa, approximately 58,000 m^2 of FSA would be required to achieve the desired dewatering. This FSA is not practical. However, the flocculated tailings would only require an average of 2500 m^2 of filter area, a more realistic surface area. Based on unpublished work on the CFF system with unflocculated tailings, Sego (2014) has improved the dewatering performance at least an order of magnitude by optimizing the pipe flow velocity and using alternative filter media. Ongoing work by Sego (2014) is expected to further enhance the CFF dewatering rates without the use of flocculants. However, given the lack of available data, the flocculated tailings filtration data will be used to demonstrate the design of the CFF system. The expected F% (15-20%) of the whole tailings to be dewatered by the CFF process is significantly less than the F% of the fluid fine tailings (>90%) discussed in Chapter 6. Although flocculant addition to fluid fine tailings can present challenges with respect to consolidation and strength gain, the influence of flocculant addition on the CFF tailings deposit behavior is expected to be minimal due to the lower F%.

Using the flocculated tailings filtration data, the required FSA per year based on actual yearly tailings flow rate and F% is presented on Figure 8.5. Due to the variability in ore fines content and tailings flow rates, the FSA varies considerably from 1500 to 4000 m^2. An actual CFF system would be based on a static FSA. Using the average FSA, the CFF tailings would meet the target C_w of 75% (+/- 5%) only 40-50 % of operating time. Therefore, there could be years where the system is underutilized and years where the CFF is unable to meet the dewatering demand resulting in lower C_w tailings deposits.

A potential opportunity exists to balance the FSA requirements and consume accumulated FFT. The SFR of the whole tailings stream ranges from ~4:1 –

7.2:1. The whole tailings stream could be spiked with MFT to stabilize the SFR ratio, thus reducing the fluctuations in the FSA requirements. Specifying a target SFR of 4.75:1, the calculated FSA and total flow of whole tailings and FFT is presented on Figure 8.5. There is a significant reduction in fluctuations and are attributed only to the tailings flow rates with an average FSA of 3275 m^2. By spiking the whole tailings, a potential of over 167 Mm^3 of MFT can be consumed by the CFF process. For comparison, an SFR of 5:1 would require an FSA of 3000 m^2, and consume ~120 Mm^3 of MFT.

To ensure sufficient FSA is available and provide a contingency, a design FSA of 3500 m^2 will be used to dewater the MFT spiked extraction tailings at an SFR target of 4.75:1. The total average flow rate with MFT spiking is 15,100 m^3/hr. Based on CFF laboratory work (Sego 2014), an in pipe velocity of greater than 3 m/s will provide optimum filtration. Assuming a diameter of 60 cm, five sections of filter pipe, each 366 m long are required to meet the dewatering requirements.

8.3.2 Impoundment DDAs

The CFF-tailings will be deposited as tailings stacks within the mined out pit. The model DDA design and stage curve details are provided in Appendix 4. Detailed depositional studies have not been completed on the CFF-tailings. Therefore, the Fitton (2007) empirical beach slope estimation (Equation 8.5) will be employed to estimate the slope with a range of +/- 50%. For, initial planning purposes, the estimated beach slope will be 2%, similar to whole tailings beaches.

[8.5] $$i = \frac{26.6 * {C_w}^2}{\sqrt{Q}}$$

Using a target C_w for the CFF-tailings of 75 %, the annual production rate is approximately 71 Mm^3/year. The annual rate was based on an average ore excavation rate of 118 tonne/year, (G_s of 2.65). The average mining rate (overburden and ore) is 100 Mm^3/year, therefore mining operations should progress sufficiently ahead of CFF-tailings deposition. The current (2013) pit limits extent and ultimate mine pit is included in Figure 8.6. The first in pit dyke (Dyke 1) has already been constructed to full elevation. Therefore, in pit deposition in DDA 1 to the full height of the pit can be initiated in 2013. DDA 1 will be in filled in the same manner as the CT process (Figure 7.11 and Figure 7.12), but at a slope of ~2% versus 0.5%.

Upon filling of DDA 1, deposition will move to DDA 2 (Figure 8.6). Dyke 2, constructed to an elevation of 20 m with dimensions and material demand proportional to the CT in-pit dykes (Figure 7.10. Typical CT DDA dyke construction material demand curve.), will bound DDA 2 on the west side. Deposition into DDA 2 will occur along a line extending from the east pit wall. This deposition scenario would represent a series of spigots along the central line. Figure 8.7 details the plan and section view of the deposition process into DDA 2. Once, DDA 2 reaches capacity, CFF tailings deposition will continue along the central line into DDA 3 (Figure 8.7. Deposition plan for CFF tailings into DDA 2 and 3.). DDA 3 will be bounded by the pit walls to the south, Dyke 3 to the north, DDA 2 to the east and Dyke 4 to the west. Dyke 4 has the same design as Dyke 2. Once excavated, DDA 4 will be used as overburden storage, additional process water storage, and emergency tailings storage. Dyke 3 will separate DDA 4 from the rest of the mine pit and is constructed to grade similar to the CT in-pit dykes. At the end of mining operations (2037, Figure 8.6), DDA 5 will be used as an end pit lake filled with process water from the ASB.

Starter beaches will not be required for the CFF-tailings as the material is expected to behave similarly to a SCT beach. Since the dyke construction material demand is considerably lower than the CT case, there should be sufficient overburden material available over the life of the mine. Therefore, all tailings will be directed to the CFF process. When the CFF plant is not operational, straight beaching or capping of previous deposits can be undertaken. FFT runoff from the beaching operations will be directed back to the ASB and ultimately incorporated into the CFF tailings.

Given the excess availability, overburden will be used for capping of the CFF tailings deposits. At least 2 m thick cap will be placed on the tailings deposits.

8.3.3 Deposit Behaviour

As with the CT tailings deposit, Stage 3 consolidation dewatering will be the dominant dewatering process following deposition of the CFF tailings, therefore compressibility and saturated hydraulic conductivity functions are required. To date there have not been any large strain compressibility tests completed on CFF-tailings. However, since the compressibility has limited influence on the rate of settlement (influences the magnitude), estimates from similar tailings materials may be sufficient for modeling purposes. The large strain dewatering compressibility behaviors of CT materials may act as a sufficient surrogate for the CFF-tailings. Therefore, the CT compressibility functions used for the CT materials will also be implemented for the CFF-tailings. Sensitivity analyses can be conducted on the compressibility relationship to understand the influence of the compressibility on the magnitude of settlement of the CFF-tailings. However, since the CFF-tailings are deposited at high C_w, or low void ratios, the amount of settlement following deposition will be significantly less than CT materials.

The saturated hydraulic conductivity, however will play an important role in the rate of settlement of the CFF-tailings deposits (Suthaker and Scott 1996). The CT saturated hydraulic conductivity behavior of different SFR tailings collapse to a single curve based on fines void ratio (e_{fines}). This trend is also expected to be true for CFF-tailings. Therefore, by knowing the saturated hydraulic conductivity of the fines and using the SFR, the saturated hydraulic conductivity of the CFF-tailings can be determined. Several data sets of fine tailings originating from Syncrude were compiled in an effort to assess changes in mineralogy, age, and chemistry on the saturated hydraulic conductivity of the fine tailings. In the following Figure 8.8, the saturated hydraulic conductivity from eight different fine tailings samples are compared. On Figure 8.8, CT represents the Syncrude CT data (Matthews et al. 2002), COF is cyclone underflow from a 2008 sampling program (Jeeravipoolvarn 2010), MFT is from Scott (2014) and represents an aged sample, Ore A (marine) and Ore B (non-marine) represent young fine tailings derived from a caustic extraction process (Miller et al. 2011), and the 10 m sample represents the MFT stored in the 10 m column experiment at the University of Alberta (Jeeravipoolvarn et al. 2009). The material properties of each tailings sample are provided in Table 8.1.

At an e_{fines} below 2, the various tailings samples have similar saturated hydraulic conductivities. Above an e_{fines} of 2, the CT and MFT saturated hydraulic conductivities increase, likely due to the pore fluid chemical differences and clay content. According to Miller et al (2011), saturated hydraulic conductivity is influenced by the water chemistry above void ratios of about 3. Both the caustic extraction fine tailings samples exhibited similar saturated hydraulic conductivity regardless of ore origin. The difference in saturated hydraulic conductivity behavior is reflected in the liquid limit (w_L) and G_s of the various samples (Table 8.1). Generally, the lower permeable samples have a greater w_L. To model the influence of w_L and G_s on saturated hydraulic conductivity, the Kozeny Carmen (KC) equation (Equation 8.6) may be used Chapuis and Aubertin (2003):

[8.6] $$k = Ckc * \frac{g}{\mu w * \rho w} * \frac{1}{Ss^2} * \frac{1}{Gs} * \frac{e^3}{1+e}$$

Where Ckc is a constant, g is the gravitational constant, μw is the dynamic viscosity of water, ρw is the density of water, G_s is the specific gravity of the solids, S is the specific surface, and e is void ratio. For clays, Chapuis and Aubertin (2003) suggest the specific surface can be calculated from the w_L (Equation 8.7).

[8.7] $$\frac{1}{Ss} = 1.3513\left(\frac{1}{wL}\right) - 0.0089$$

Using the data in Table 8.1 and the KC method described by Chapuis and Aubertin (2003), the saturated hydraulic conductivity of each tailings stream is calculated and compared with it's measured saturated hydraulic conductivity on **Error! Reference source not found.**. The best fit Ckc constant was found to be 12. The KC method tends to over estimate saturated hydraulic conductivity at low void ratios and under estimate the saturated hydraulic conductivity at high void ratios. This difference can be attributed to the $e^3/(1+e)$ term. The KC equation was developed by considering a porous material as equivalent to a bundle of tubes for which the Navier-Stokes equation can be utilized (Chapuis and Aubertin 2003). Therefore, the basis of the $e^3/(1+e)$ term is theoretical. Carrier et al. (1983) suggested the k of various tailings samples should be expressed as Equation 8.8.

[8.8] $$k = E * \frac{e^F}{1+e}$$

Where E and F are fitting parameters. For the KC equation, F is equal to 3 and E can be calculated from the material and fluid properties. For hard rock tailings, Chapuis and Aubertin (2003) propose F is equal to 3 plus a positive constant and Bussiere (2007) suggested F equal 5.2. The increase in the F value was required to accounts for the influence of path tortuosity in the hard rock tailings (Bussiere 2007). Therefore, the KC equation could be modified slightly (F parameter) to fit the measured saturated hydraulic conductivity data to account for tortuosity. In log-k space, linear regression (Figure 8.10) was used to determine the parameter F is equal to 4.46, which is slightly lower than the value for proposed for hard rock tailings. This difference can be attributed to the particle size difference between the two tailings types and the resulting influence on tortuosity. The E parameter was calculated using the KC equation with Ckc of 12. Using the updated F parameter of 4.46, the modified KC data was compared with the measured k in **Error! Reference source not found.** and showed an improved correlation. The calculated saturated hydraulic conductivity was within 2 or ½ of the measured value. This error is within the range of laboratory error and scatter of typical k versus e_{fines} plots. Therefore, the modified KC equation can be satisfactorily used to calculate the saturated hydraulic conductivity of Syncrude fine tailings using a value of 4.46 for F and Ckc of 12.

For the TMSim model however, the K-e_{fines} relationship must be expressed in the form of Equation 8.9.

[8.9] $k = Ce^D$

Therefore, the modified Kozeny Carmen equation of with constants of E and F must be transformed to parameters C and D. A series of data sets were created using the modified Kozeny Carmen parameter F = 4.46, and a range of Sg (2.3 to 2.7) and liquid limits (40 to 52.5). A power law function was then fit to the data sets to determine the C and D parameters as functions of Sg and w_L. The D parameter was constant at 3.774 regardless of Sg or w_L. The influence of Sg on the C parameter was minuscule (less than +/ $1e^{-9}$) compared with the impact of w_L. Therefore, the C parameter was computed as a function of w_L (Figure 8.12). For Syncrude fine tailings with an average Sg of 2.5, the k-e_{fines} relationship can be determined with Equation 8.10 and 8.11.

[8.10] $C = 0.0002 w_L{}^{-2.871}$

[8.11] $k = Ce_{fines}{}^{3.77}$

Using the SFR relationship (Equation 8.12) and Equations 8.10 and 8.11, the saturated hydraulic conductivity of the CFF-tailings can be calculated.

[8.12] $e_{fines} = e*(SFR+1)*Sg_{fines}/Sg_{coarse}$

Therefore, the saturated hydraulic conductivity of the CFF-tailings is essentially a function of the fines content and w_L of the tailings. The w_L depends on the amount of clay, mineralogy, amount of bitumen, and chemistry of the pore fluid. For ores with constant mineralogy and pore chemistry, the w_L will be a function of the clay content. Unlike the variability of clay in FFT (Table 8.1), the clay content in ore may be a function of the F% (Masliyah et al. 2011). Figure 8.13 depicts Masliyah et al's (2011) and Ciulavu's (2008) relationships for the clay content of various oil sand ore which can be used to estimate the clay content of extraction tailings. However, sufficient data was not available in Table 8.1 for the Syncrude tailings to link clay content with w_L with sufficient confidence. Therefore, assuming the relationship from Masliyah et al (2011) is valid, and using the maximum and minimum fines content of the Aurora ore, and the maximum and minimum w_L of the Syncrude fine tailings, a relationship between fines content of the ore and the w_L can be established (Equation 8.13). This relationship is not ideal, but with future testing it can be improved.

[8.13] $$w_L = -1.583 * F\%/100 + 71.66$$

Therefore, the 'C' parameter for the CFF tailings can be calculated from equation 8.14.

[8.14] $$C = 0.0002*(-1.583*F\%/100+71.66)^{-2.871}$$

Detailed strength testing of the CFF-tailings product have not been completed. However, a comparable analogue may be the Syncrude CT deposit. Therefore, the approach used for the CT deposit strength profiles will also be adopted for the CFF-tailings. With future testing, the estimates of the strength profiles can be improved.

8.4 SIMULATION RESULTS

The TMSim model was utilized to assess a CFF technology scenario applied to the same mine plan as the CT technology in Chapter 7. Model mine plan assumptions and model input data are included in Appendix 4. The extraction efficiency for the CFF simulation is the same as the CT simulation, therefore it will not be reported again. Several preliminary simulations were completed to assess the CFF technology assumptions and dewatering model. These initial simulations did not include Stage 3 dewatering in order to simplify and speed up the modeling time. Using the results from the preliminary simulations as a guide, a final simulation was then completed incorporating Stage 3 consolidation.

8.4.1 Preliminary Simulations

8.4.1.1 CFF-1

The first preliminary test (CFF-1) utilized the CFF process design described above based on average ore fines content and flow rates. The CFF process will utilize an FSA of 3500 m^2, a static filter pressure of 150 kPa, a constant cake thickness of 0.005 m, and a target SFR of 4.75. The target CFF deposit will achieve a C_w of 75 % and attain a deposition slope of 2%.

Using the assumptions above, the three DDAs had been completely filled by CFF tailings after only 304 months (20 months before end of mining). Although the DDAs filled nearly two years before the end of the mine life, it is acceptable because the tailings did not consolidate during the simulation. It is expected that once consolidation is included, the 3 DDAs will provide sufficient storage for the CFF tailings.

In CFF-1, there was a residual FFT volume of 30 Mm^3 left in the ETF at the end of mining, meaning the FFT spiking of the feed tailings to a target 4.75 was not sufficient to consume the 100 Mm^3 of FFT. Additionally, the actual C_w attained by the CFF process only achieved the target of 75% in 4 of 27 years (Figure 8.14). The lower C_w means less water was removed from the tailings stream and is therefore lost to the tailings deposit. This excess water in the CFF tailings resulted in an increased volume of the deposit and contributed to the early filling rate.

The CFF process design used for CFF-1 was based on an average fines content of ore and not the actual fines content of the feed tailings stream. The design also did not include the mass of residual bitumen in the feed tailings. The residual bitumen arises from bitumen that is not captured during extraction. This bitumen is commonly included with the fines mineral component and thus results in a higher fines content of the feed tailings. The higher fines content also means a greater FSA would be required to meet the target C_w. Figure 8.14 includes the required FSA for the CFF-1 test run. The required FSA was greater than the design 3500 m^2 for nearly the entire simulation. Therefore, the design of the CFF process needs to be revised to ensure the target 75% C_w is achieved and the DDAs can accommodate the CFF tailings deposit.

A modified design will be based on a target SFR of 4.5 to ensure more FFT is consumed during the spiking process. Additionally, the FSA will need to be increased to account for the increased fines content of the feed tailings (results in lower filtration rate). The CFF C_w achieved for different FSAs is shown in Figure 8.15. In the plot, a maximum C_w was set at 75% to demonstrate when the target C_w is not achieved. In the model mine plan, there are two time periods with very high fines contents (120-132 months and 276-288) that result in a low C_w regardless of the FSA. An FSA of 4000 m^2 would nearly always achieve the 75% target. An FSA of 3850 m^2, would meet the target 60 % of the time and be within 2% of the target C_w more than 90% of the time. The updated CFF process design will then utilize an SFR of 4.5 and an FSA of 3850 m^2. The updated CFF design would require additional filter pipe, therefore the design pipe lengths will be increased from 366 to 400 m to account for additional FSA.

8.4.1.2 CFF-2

Using the same model mine plan and assumptions, but with an updated SFR target of 4.5 and FSA of 3850 m^2, a second simulation was carried out. The updated design was able to consume all of the FFT by 258 months (Figure 8.16). Therefore, no residual FFT would need to be transferred to an end pit lake at the end of mining. A total of 1770 Mm^3 of CFF tailings were deposited into the 3 DDAs. However, the final DDA 3 did reach capacity at 306 months (18 months early, Figure 8.17). This simulation demonstrates the updated SFR target and FSA are acceptable.

8.4.1.3 CFF-3

The CFF-2 simulation assumed a constant deposit slope of 2%. In reality, this slope will vary depending on the tailings properties, flow conditions and the deposition method (Fitton 2007). Since limited information is available on the actual slope that can be achieved by the CFF tailings, the Fitton equation (8.5) will be used to estimate the variability of the CFF tailings as a function of C_w. This simulation will then demonstrate the influence of deposit slope on the filling rate of the DDAs. The influence of slope is noticeable on the maximum tailings height for each DDA (Figure 8.18). When the CFF tailings slope is less than the prevailing deposit slope, tailings will accumulate at the lowest point reducing the rate of rise of the stack. If the slope is greater than the prevailing deposit slope, the rate of rise will then increase as tailings are deposited at the top of the stack. From Figure 8.18, we can see there is a steep increase in slope in DDA 3 near the end of filling and this will have contributed to the early filling of the DDA 3. By month 300 (Figure 8.19), the DDA 3 reached its maximum capacity and only 1740 Mm^3 of CFF tailings were deposited.

8.4.1.4 CFF-4

The simulations to this point have utilized a static filtration pressure and cake thickness for CFF process model. In reality, the cake thickness and pressure will vary with time and along the length of the pipe. To assess the influence of pressure and cake thickness on the C_w of the dewatered CFF tailings, several scenarios were assessed where the pressure and cake thickness varied by +/- 50% of the static values. The worst case scenario (Figure 8.20) occurs when the pressure is low and the cake is thick. The average C_w achieved for this scenario was only 57% solids. At this state, the CFF tailings would behave as a segregating slurry and resulting in formation of FFT. CFF tailings produced with a low pressure and average cake thickness would only achieve a C_w of 60% and would likely still segregate. If the pressure could be maintained, but a thicker cake was developed, the C_w improved to 65 %. The CFF may not be segregating at this C_w, but there deposition slope would be considerably less than the target of 2% and likely closer to 0.5% like a CT tailings.

To ensure the CFF process maintains a constant minimum cake thickness, the cross flow pipe velocity needs to be maintained near 3 m/s (Sego 2014). During periods of lower flow rates (thus lower pipe velocities), dewatering tailings and filtrate can be recycled and mixed with the feed tailings stream to boost the flow rate. This feedback process will ensure the minimum velocity is maintained in the system.

To ensure the pressure is maintained for the CFF process, booster pumps may be required. According to Shook et al. (2002), CT tailings have a pressure gradient of about 0.2 kPa/m at a pipe velocity of 3 m/s. Assuming the pipe velocity is maintained along the length of filter pipe, the drop in pressure for the 400 m length of filter pipe may be 80 kPa (54% drop in pressure). To ensure an average pressure along the length of the pipe is near the target 150 kPa, a booster pump is required. Therefore, the pumping system for the CFF process would likely include a feed pump operating at a pressure of at least 170 kPa and at a flow rate to maintain 3 m/s pipe velocity. After 200 m, the pressure will have dropped to ~130 kPa. At this point a second booster can be used to increase the pressure back to 170 kPa. By utilizing two pumps along the length of the filter pipe, the average filter pressure is maintained at 150 kPa. The expected C_w at the booster pump would be approximately 63% (Figure 8.2), similar to the operating density of CT tailings. Therefore, conventional pumping systems would be sufficient for use as a booster.

The CFF process design was then updated to incorporate the feedback system to ensure a minimum velocity is maintained. Additionally, a maximum C_w for the CFF process will be set at 77.5 % to ensure the CFF tailings do not over dewater during the process which may result in plugging of the CFF process piping. A simulation (CFF-4) was carried out using the updated CFF design.

The DDA 3 reached capacity after 266 months when the feedback system was added to the CFF process model. Additional flow from the recycle cycle system resulted in less water being removed from the tailings in the CFF process and low C_w (~56%) at periods of recycle flow. These tailings would be segregating, therefore, the current CFF design must be modified to handle the extra flow rate from the feedback system.

8.4.1.5 CFF-5

The FSA was then increased to 4000 m^2 to accommodate the additional flow from the feedback system. The first DDA 1 reached capacity (375 Mm^3) after 69 months (Figure 8.21). By month 151, the second DDA 2 reached capacity (490 Mm^3). The final DDA 3 reached capacity (895 Mm^3) after 300 months (2 years before end of mining) after a total of 1760 Mm^3 of CFF tailings were deposited in the mine pit (Figure 8.22). All of the FFT on the site was consumed by month 258 and the process water volume in the ETF was maintained at 46.6 Mm^3 (Figure 8.22). The average process water make up rate was only 9.8 Mm^3/yr (Figure 8.23). A total of 240 Mm^3 of make up water from the Mildred Lake settling basin (MLSB) was required over the life of the mine. An average of 68 Mm^3/yr of filtrate is directly available for recycle from the CFF process.

The average C_w of CFF tailings deposited into DDA 1, 2 and 3 respectively was 76.7 %, 75.4% and 76.4% (Figure 8.24). During the final years of filling in each DDA 2 and DDA 3, a period of high fines content ore was mined and resulted in a drop in the CFF dewatering efficiency (drop in C_w slightly below 70%).

The preliminary simulations did not include Stage 3 dewatering (consolidation), therefore, an assessment of the effective stress profile and ultimately, deposit strength, are not possible. Similarly, since no consolidation water was generated, an assessment of the ion concentrations in the ETF pond water is not relevant.

8.4.2 Stage 3 Dewatering

8.4.2.1 CFF-6

Based on the final CFF process design and mine plan assumptions used in CFF-5, a full simulation incorporating Stage 3 dewatering was completed using the compressibility and saturated hydraulic conductivity functions outlined in Appendix 4. Insufficient in-pit storage was available for the CFF tailings deposit in simulation CFF-6. DDA 3 reached full capacity 16 months before end of mining.

The first DDA 1 reached capacity (375 Mm^3) after 70 months (Figure 8.25). By month 153, the second DDA 2 reached capacity (490 Mm^3). The final DDA 3 reached capacity (902 Mm^3) after 308 months (1.5 years before end of mining) after a total of 1763 Mm^3 of CFF tailings were deposited in the mine pit (Figure

8.26). All of the FFT on the site was consumed by month 258 and the process water volume in the ETF was maintained at 44.7 Mm^3 (Figure 8.26). The average process water make up rate was only 8.6 Mm^3/yr (Figure 8.27). A total of 190 Mm^3 of make up water from the MLSB was required for the 308 months of mining.

Effective stress profiles at the end of filling for each DDA are included in Figure 8.28. The vertical depth axis was normalized by the maximum depth of the deposit to allow for a comparison between the three DDAs. The effective stress profiles show that very little consolidation occurred in the tailings deposits. This is reflected in the profiles of the C_w of CFF tailings in DDA 1, 2 and 3 at the end of mining Figure 8.29. Only the lower 25 % of the deposit in DDA 1 and 2 developed an effective stress and increase in C_w. Due to the greater depth of the deposit in DDA 3 and slower loading rate, slightly more consolidation had occurred in the lower 40% of the deposit.

To demonstrate the impact of a 2 m overburden cap (i.e. surcharge load) on the tailings deposit, a surcharge was added to the final deposit profile (Figure 8.28). After 5 years, there is only a slight improvement in the consolidation of the tailings deposits. The surcharge load did very little to improve the consolidation of the deposits. The upper 50% of each deposit still did not consolidate with very little consolidation occurring at the surface.

Using the assumed S_u/σ'_v ratio of 0.12, predicted undrained shear strength profiles were calculated for the tailings deposits at times right after deposition ceased and 5 years after the surcharge loading (Figure 8.30). Due to the low effective stresses, the tailings deposits have very little to no undrained strength. Only the bottom 1-18% of the deposit achieved undrained strengths greater than 5 kPa. The surcharge loading slightly improved the undrained strength profiles. Only the lower 20 to 25% of the deposits were greater than 5 kPa.

Predicted species concentrations in the ETF process water pond are included in Figure 8.31. For clarity, only sodium (Na), calcium (Ca), chloride (Cl) and sulphate ($S04^{2-}$) are included. The concentrations of Na, Cl and $S04^{2-}$ dropped within the first year due to the low volume of make up water and influx of site run off (lower concentrations than ETF pond). The Ca concentration was relatively constant throughout the simulation.

8.4.2.2 CFF-7

A second full simulation (CFF-7) incorporating Stage 3 dewatering was completed using the compressibility and saturated hydraulic conductivity functions utilized for the CT tailings technology outlined in Appendix 3. The saturated hydraulic conductivity function used in CFF-7 is about an order of magnitude greater than CFF-6. Due to the improved consolidation behavior of the CFF tailings in simulation CFF-7, sufficient in-pit storage was available to contain all of the CFF tailings produced over the life of the mine (Figure 8.32).

A comparison of the filling times and tailings rise rates between simulations CFF-5, CFF-6 and CFF-7 is provided in Table 8.2. The improved consolidation properties of CFF-7 resulted in an approximately 7% improvement on the tailings rise rate and fill times over the CFF-6 results. The three DDAs in CFF-7 had sufficient storage for the 1717 Mm^3 of CFF tailings deposited in the mine pit (Figure 8.33). Similar to CFF-6, all of the FFT on the site was consumed by month 258. The process water volume in the ETF was maintained at an average of 45.6 Mm^3 (Figure 8.33). Due to the improved consolidation behavior in CFF-7, the average process water make up rate was 4.5 Mm^3/yr (Figure 8.34), about half of CFF-6. A total of 123 Mm^3 of make up water from the MLSB was required over the life of the mine.

Effective stress profiles for CFF-7 demonstrate the impact of the improved saturated hydraulic conductivity on consolidation (Figure 8.35). In each DDA, effective stress was generated over the entire deposit. This is reflected in the profiles of the C_w of CFF tailings Figure 8.36. In all three DDAs, the C_w of the tailings increased with depth to approximately 85%.

Predicted undrained shear strength profiles were calculated using the assumed S_u/σ'_v ratio of 0.12, right after deposition ceased and 5 years after a 2 m overburden surcharge load was applied (Figure 8.37). Immediately after deposition stopped in DDA 1 and 2, the lower 60% of the deposit reached an S_u of 5 kPa or greater. In DDA 3, almost 90 % of the deposit achieved an S_u of 5 kPa or greater. Five years after the surcharge was applied, the entire deposit was above 5 kPa in all DDAs.

Predicted species concentrations in the ETF process water pond for CFF-7 are included in Figure 8.38. The concentrations of Na, Ca, Cl and S04 were similar to CFF-6.

8.5 DISCUSSION

Several preliminary TMSim simulations were conducted before a practical design and model for the CFF technology was discovered. In fact, for every simulation reported herein, there were at least 5 simulations that were not reported due to errors in the TMSim model. These errors arose from issues such as not having sufficient space for CFF tailings or the ETF would over flow with process water in the early years of the mine plan. These failed simulations, although no data was available for reporting, still provided utility as they helped bound various mine plan assumptions (such as required water/FFT transfer rates from the DDA to the ETF and required impoundment dyke designs).

Simulations CFF-1 through CFF-4 tested the preliminary CFF design and model assumptions against a mine plan. After each simulation, the CFF design was improved based on results of the previous run. Variations in the properties (flow rates and fines contents) of the whole tailings reporting the CFF process was shown to have an impact on the final tailings deposit profile and storage efficiency. With an improved design and implementing adequate process control (i.e. feedback for low flow periods) the TMSim model demonstrated the variations in process conditions could be overcome.

If the CFF technology was only evaluated on its own (not incorporated into a mine plan) it would be difficult to assess the implications of poor dewatering performance resulting in low C_w of the tailings deposit. The preliminary simulations identified that early filling of the DDAs would occur in these instances. These simulations can be used to provide a worst case scenario when evaluating the merits of implementing this technology.

Using an optimized design, simulation CFF-5 was successfully completed and demonstrated that the CFF technology could be implemented as a potential technology for oil sands tailings dewatering. With a robust design (i.e. adequate FSA) and process controls on flow rate and pressure, the CFF process could consume the stockpile of FFT when spiked into whole tailings. Additionally, significantly less make up water (80% less) was required than a mine plan implementing CT technology (CT-1, Chapter 7).

Based on the design utilized in CFF-5, a simulation was then completed with Stage 3 dewatering implemented (CFF-6). Based on the DDA configuration and mine plan, there was insufficient room to contain all of the CFF tailings. This was due to the low saturated hydraulic conductivity of the CFF tailings in CFF-6. Very little dewatering occurred while the CFF tailings were deposited as evident by Figure 8.29. The density only increased (C_w) in the lower 30-40% of the deposits. Due to the low deposit dewatering rate, the average make up water rate was similar to the CFF-5 rate. Although the DDAs filled up before the end of mining, there is still sufficient room within the pit to contain the remaining 2 years of tailings produced. A fourth DDA could be developed to the west of DDA 3 by building a small containment dyke and continuing with the deposition of tailings on top of DDA 3.

When the saturated hydraulic conductivity of the CFF tailings were increased an order of magnitude (CFF-7), sufficient space in the DDAs was available for the CFF tailings. The increase in saturated hydraulic conductivity improved the dewatering rate of the CFF tailings resulting in greater volume change. The densities (C_w) of the deposits increased as tailings were deposited due to the improved consolidation. Due to the increased dewatering rate, more water was available as recycle resulting in an average make up water rate of only 4.5 Mm^3/yr. In all cases, the CFF process liberates approximately 68 Mm^3/yr of filtrate water that is recycled immediately to the extraction process. This water would require significantly less heating than using stored process water on site or water imported from MLSB, resulting in a reduction of green house gas production.

The water quality results from CFF-6 and CFF-7 indicate that the concentrations in the ETF are influenced by dilution from precipitation (site runoff) and the recycle of CFF deposit consolidation water. Since no chemicals were added to the CFF tailings, the species concentrations are fairly stable during mining.

Studies on the depositional and deposit dewatering behavior have not yet been completed on the CFF tailings. The parameters used for the compressibility and saturated hydraulic conductivity were based on engineering judgment. The two simulations CFF-6 and CFF-7 offer a lower and upper bound on the expected dewatering performance for the CFF tailings deposits. If the actual consolidation behavior is confirmed to be similar to CFF-6 tailings, the depositional plan may need to be modified. Since very little dewatering occurs during deposition in CFF-6, the deposit may be prone to liquefaction and slope instabilities. Therefore, the containment dykes would need to provide full containment of the deposit. Additionally, the tailings could not be stacked above grade without containment.

To improve the deposit dewatering performance, the tailings may be modified before deposition. A lower FFT spiking rate (higher SFR target) could be utilized to increase the saturated hydraulic conductivity of the CFF tailings. Chemical additives (flocculants) may also be added to the tailings prior to deposition to enhance the dewatering performance. Additionally, improved drainage within the deposit may also be required. This may include providing coarse sand drainage layers interbedded within the CFF deposit. To create the drainage layers, whole tailings would have to be beached on the CFF deposit and would generate FFT.

Following deposition of the tailings, wick drains and increasing the cap surcharge could also aide in the dewatering of the deposit. The increased costs (accumulation of FFT, chemical additives, and drain installation) and benefit of improving the deposit behavior would need to be balanced with potential cost and benefits of implementing the CFF technology in the first place.

8.6 CONCLUSIONS

The CFF dewatering process described above provides an opportunity to deposit high density tailings stacks requiring minimal containment. Two thirds of the yearly process water demand can be satisfied by immediate recycle from the CFF process resulting in lower green house gas production from heating water. Additionally, if FFT spiking is incorporated, existing inventories of FFT can be consumed and stored in the pore space of the CFF tailings. The closure land scape for a CFF tailings deposit would be dominated by above grade, terrestrial deposits and an EPL without FFT.

However, the simulations were based on laboratory and bench scale test results. Further bench scale and pilot testing is required to confirm and improve the CFF design and model assumptions. For example, the fines content-specific resistance to filtration relationship should be further investigated for tailings with or without flocculation. The dewatering rates presented above provide a minimum target that should be achieved to ensure a satisfactory tailings plan. The influence of bitumen content on filtration rates and potential filter membrane fouling should also be investigated. Additionally, the tailings behavior such as deposit slope and consolidation parameters need to be determined from actual CFF tailings generated from laboratory and pilot testing. Following deposition, potential mitigative measures may be required to improve dewatering may. The cost of utilizing these improvements may out way the benefits of implementing the CFF technology.

The simulations completed for the CFF technology have demonstrated the relevance of the TMSim model for examining a novel technology. Furthermore, the process of compiling the necessary input data required by the TMSim model and the simulations themselves have provided significant insight into the CFF process and their impact on a mine plan. The TMSim simulations presented above provide a baseline for further refinement and sensitivity analyses of the technology and depositional scenarios. As research and development progresses on the CFF process, the model can be refined, providing an improved understanding of the impact of the CFF technology to a mine and tailings plan.

8.7 TABLES

Table 8.1. Syncrude tailings sample properties

Parameter	COF	MFT	10 m	Ore A	Ore B
Fines content (%)	94	96	89	94	96
Clay content (%)	30*	52	45	43	49
Liquid limit (%)	43	50	46	49.5	52.1
Plasticity Index (%)	25	29	25	23.7	25.2
Bitumen content (% - total mass)	3.3	3**	3.1	0.35	0.5
SG	2.52	2.44	2.28	2.55	2.48

Notes: * = non dispersed

** = % of mineral solid

Table 8.2. Comparison of filling time and rise rates for CFF simulations.

DDA	CFF-5	CFF-6	CFF-7
DDA 1			
DDA Start	0	0	0
DDA Full	69	70	75
Rise Rate (m/month)	1.16	1.14	1.06
DDA 2			
DDA Start	70	71	76
DDA Full	151	153	165
Rise Rate (m/month)	1.06	1.04	0.96
DDA 3			
DDA Start	152	154	166
DDA Full	300	308	324
Rise Rate (m/month)	0.64	0.61	0.58

8.8 FIGURES

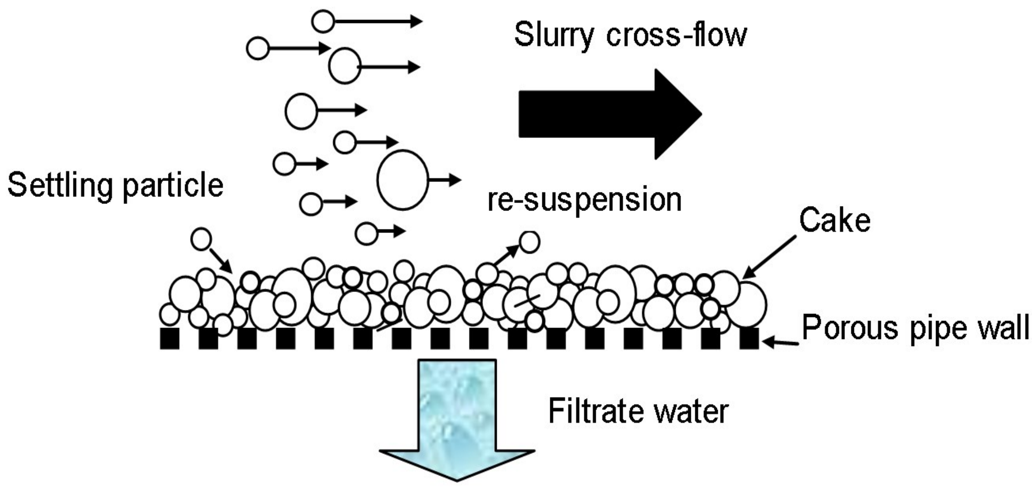


Figure 8.1. Schematic of the cross flow filtration process (modified after Beier and Sego 2008).

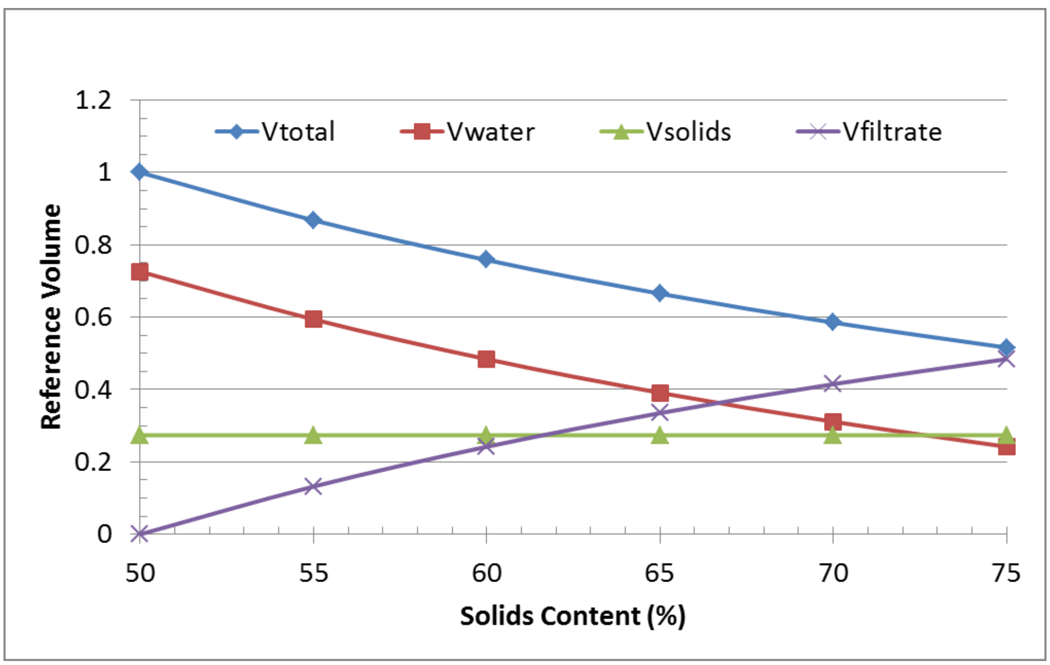


Figure 8.2. Volume reduction of CFF-tailings and corresponding solid and water components.

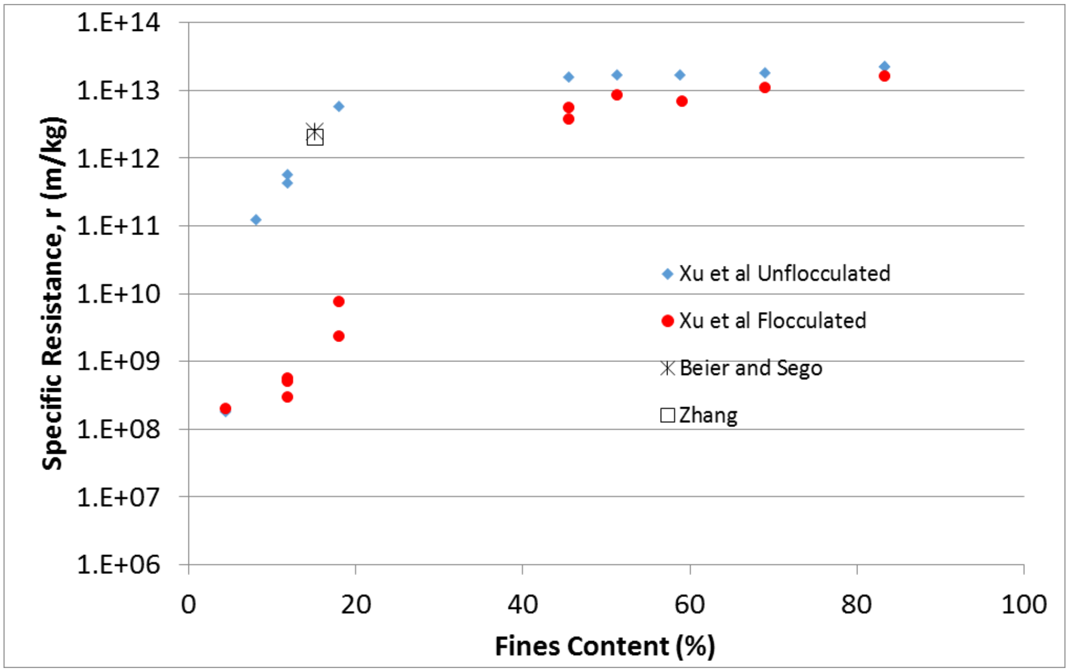


Figure 8.3. Specific resistance to filtration for various tailings streams and fines contents.

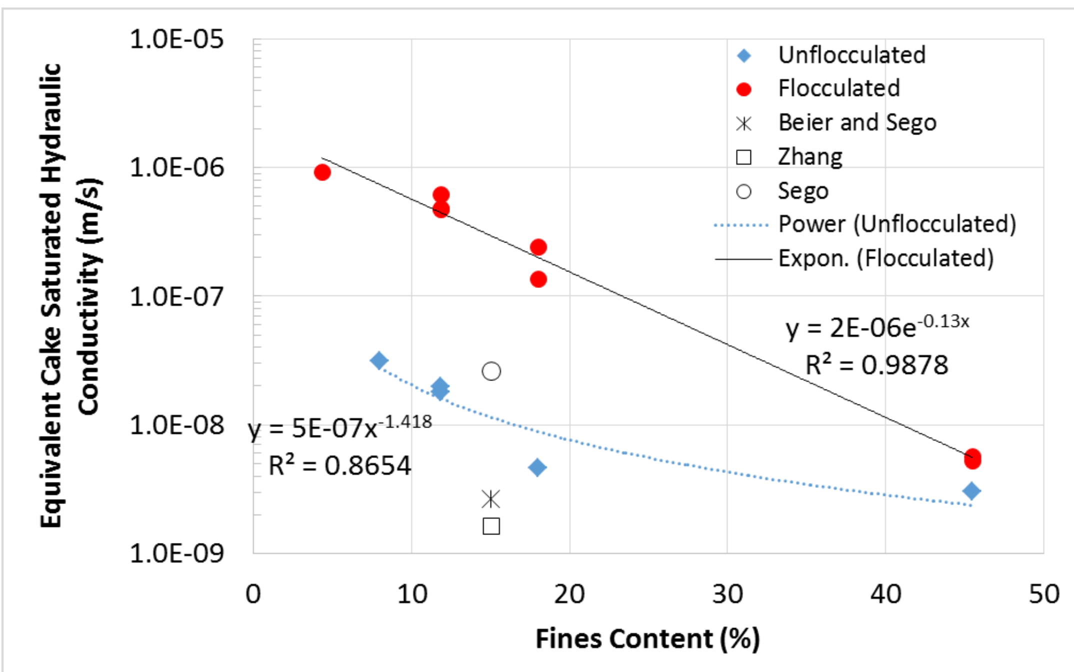


Figure 8.4. Equivalent cake saturated hydraulic conductivity for various tailings streams.

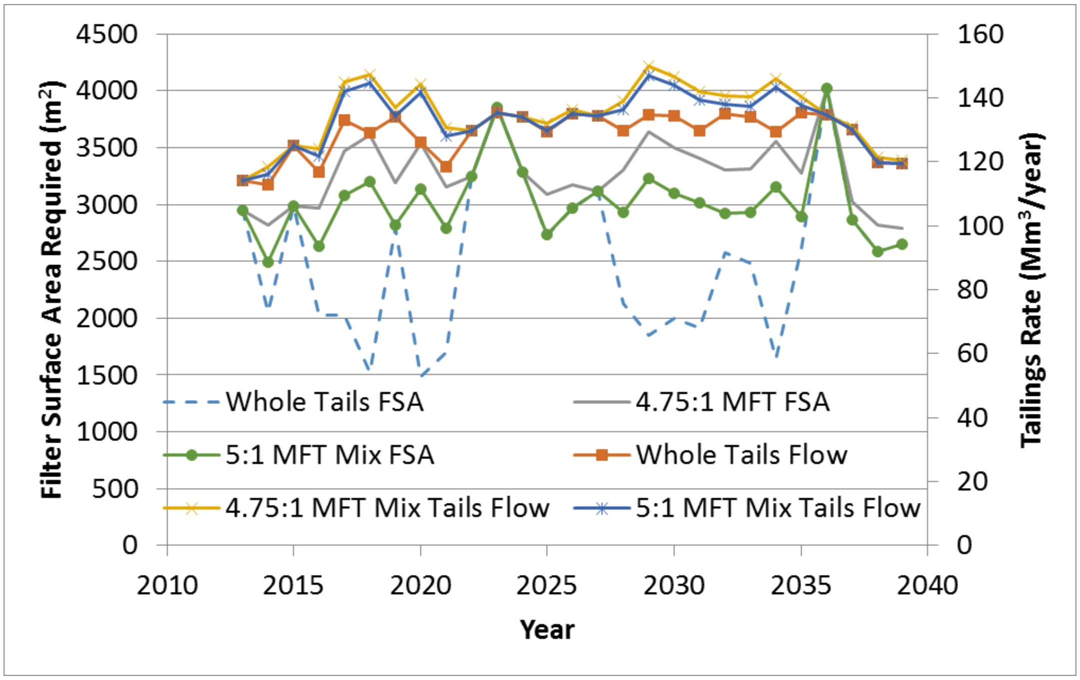


Figure 8.5. Filter surface area and tailings flow rates.

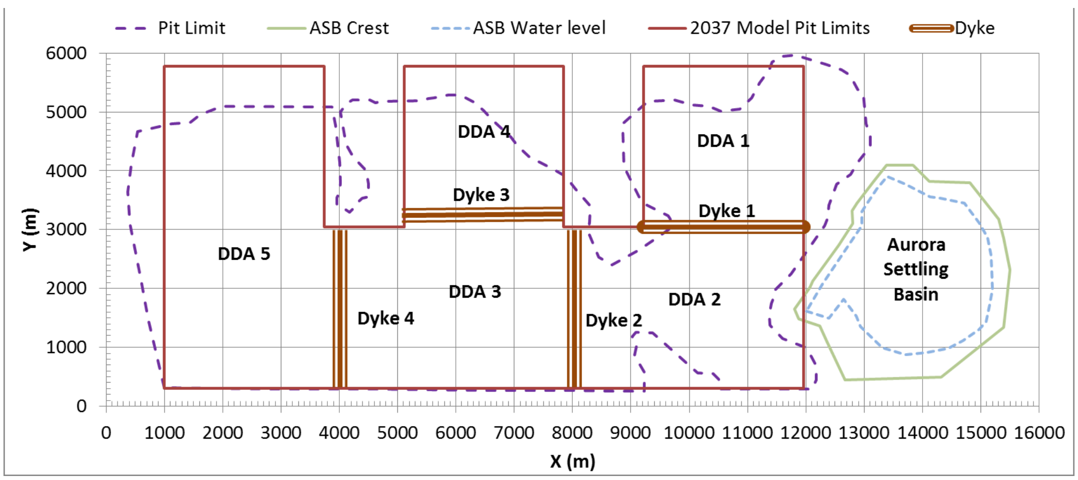


Figure 8.6. Model mine pit and in-pit DDAs for the CFF technology.

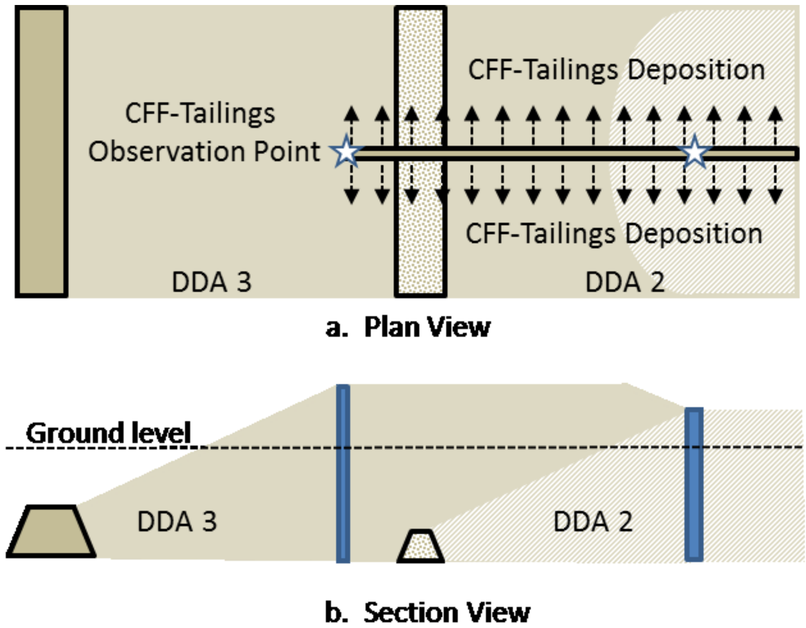


Figure 8.7. Deposition plan for CFF tailings into DDA 2 and 3.

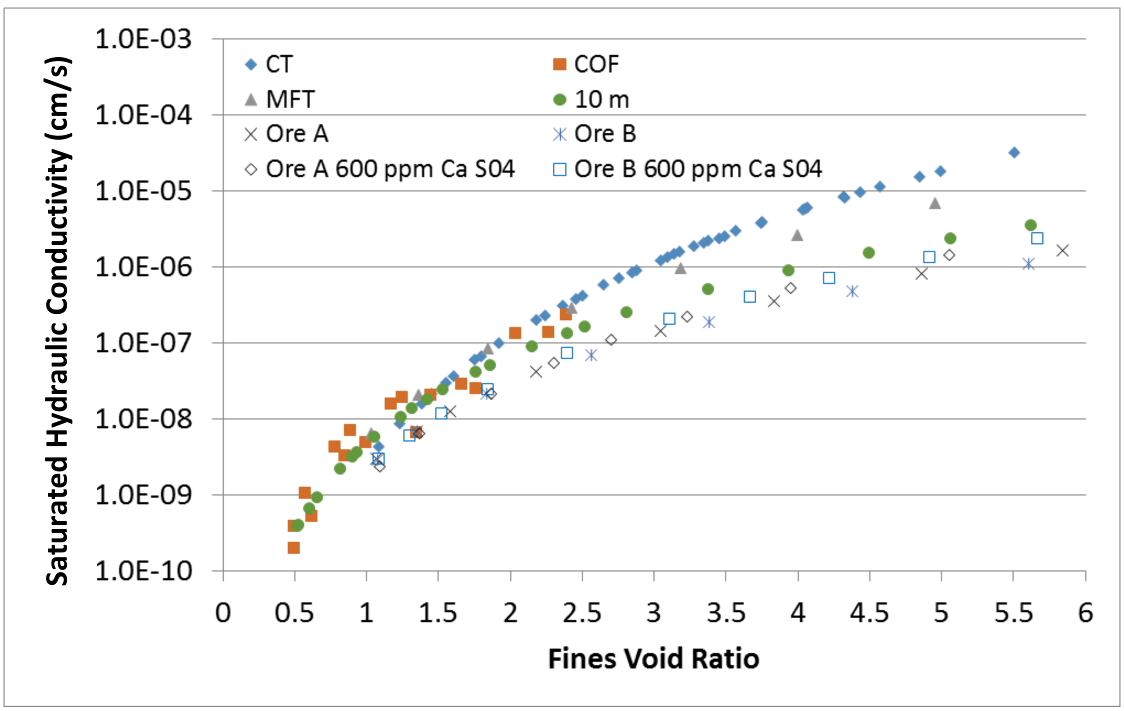


Figure 8.8. Various Syncrude fine tailings saturated hydraulic conductivities.

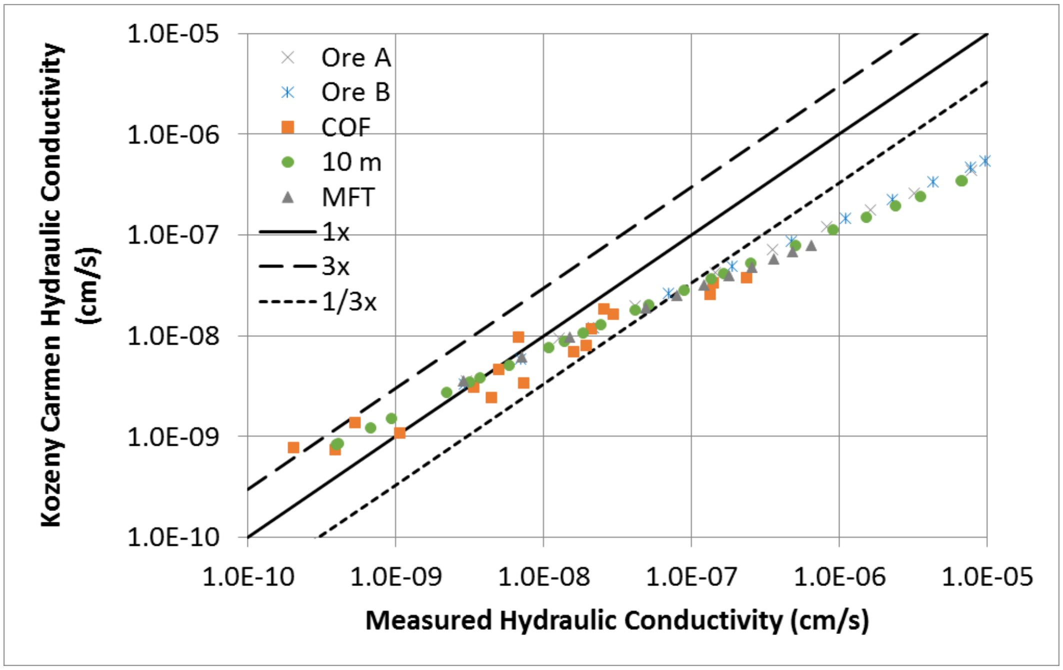


Figure 8.9. Measured versus Kozeny-Carmen calculated saturated hydraulic conductivity for various tailings.

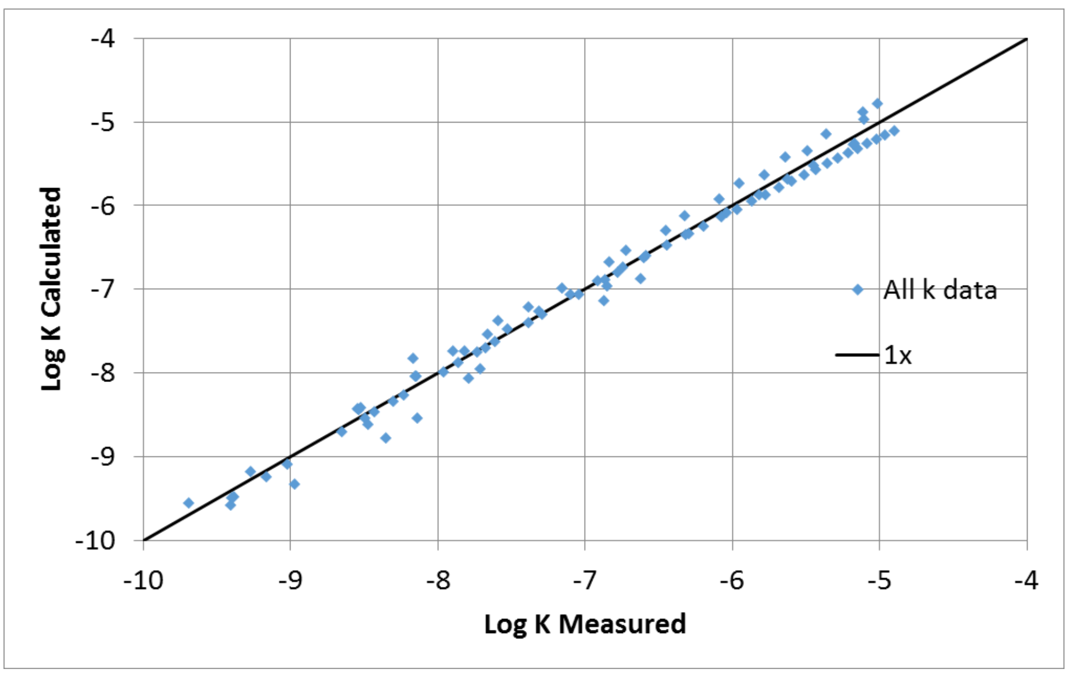


Figure 8.10. Log k space for linear regression analysis of the F parameter.

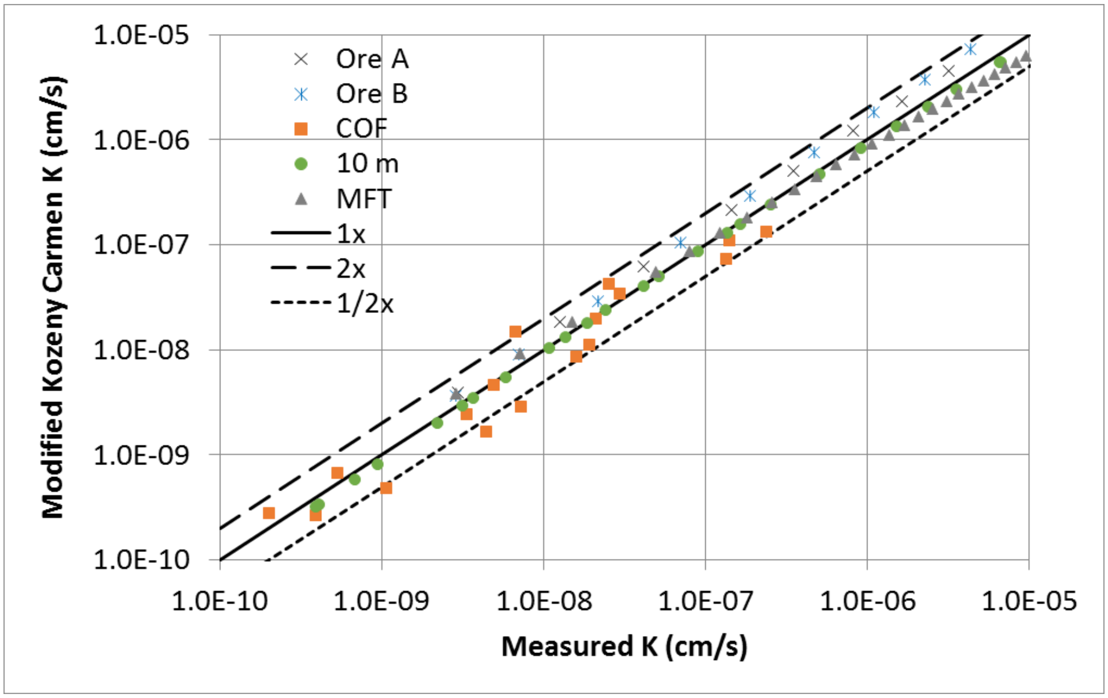


Figure 8.11. Measured versus Modified Kozeny-Carmen calculated saturated hydraulic conductivity for various tailings.

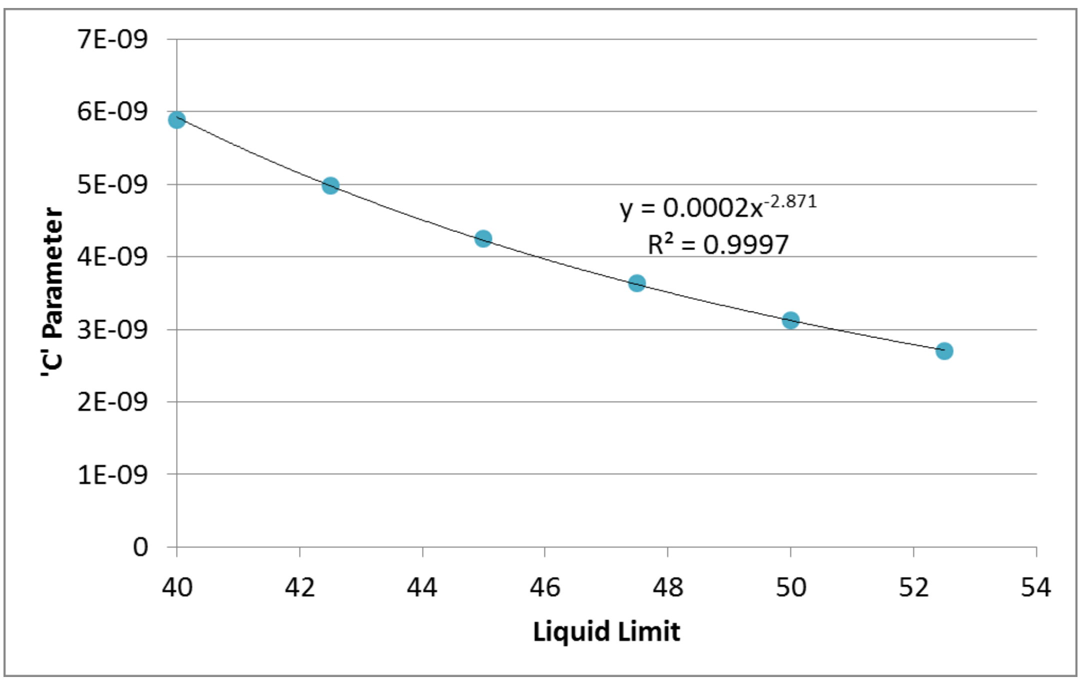


Figure 8.12. C parameter as a function of liquid limit.

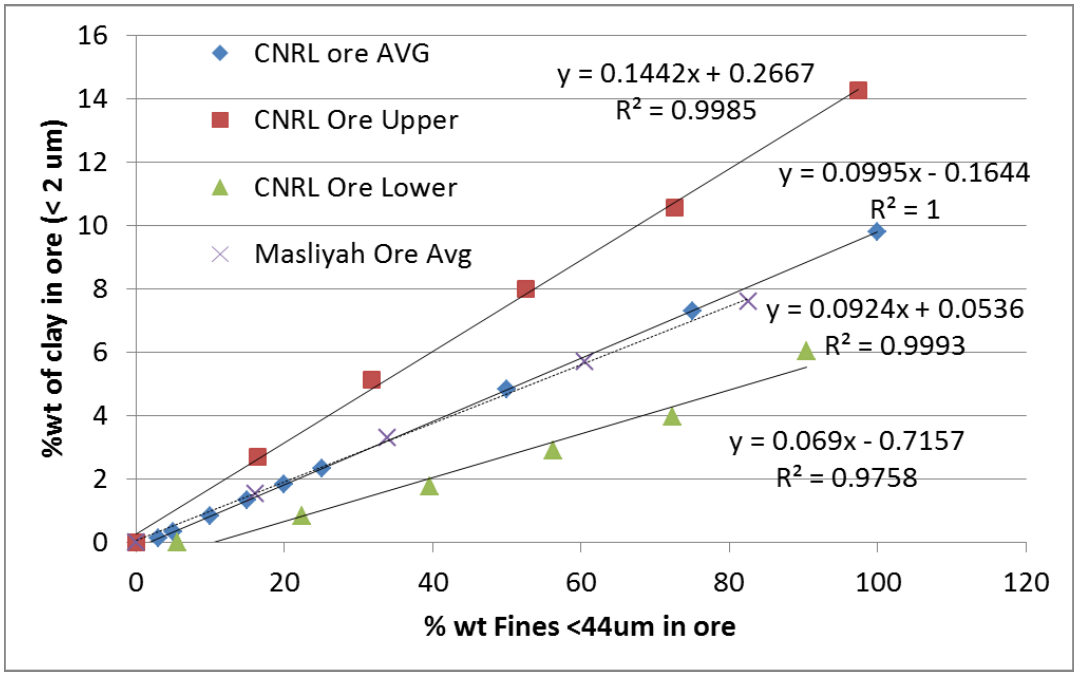


Figure 8.13. Clay content as a function of fines content in ore.

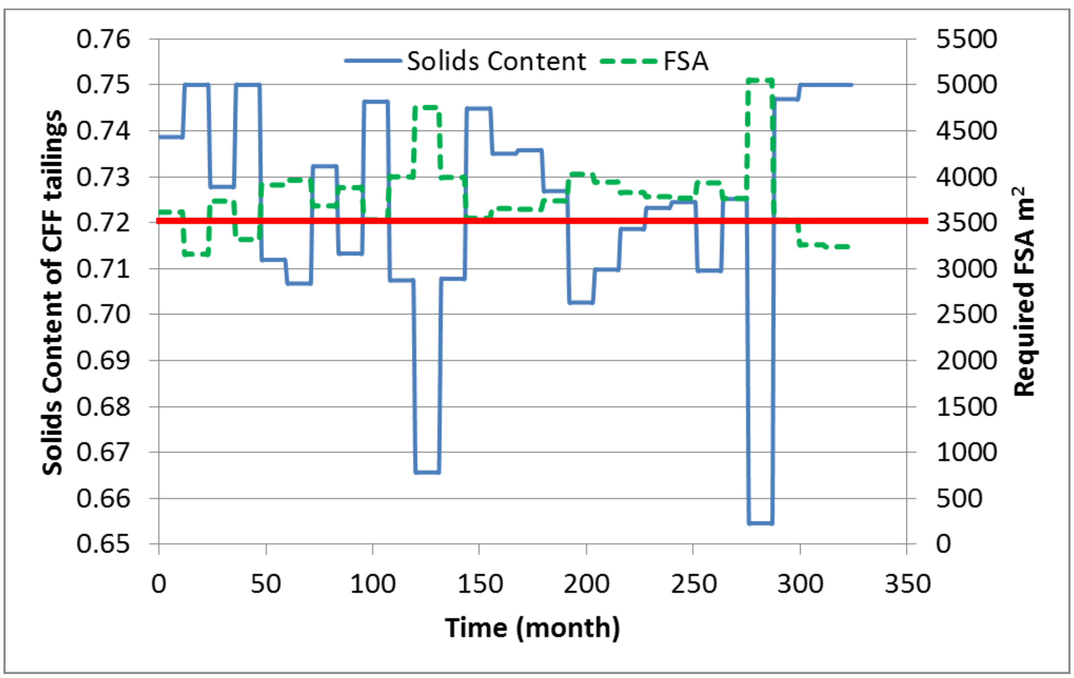


Figure 8.14. Solids Content of tailings and required FSA from model run CFF-1.

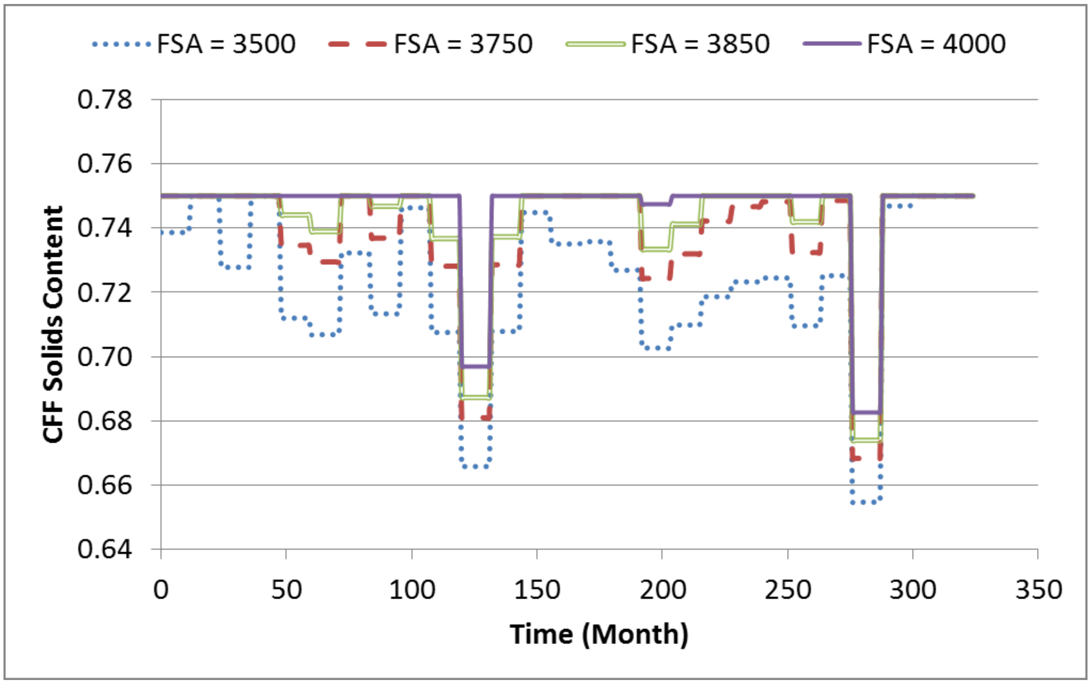


Figure 8.15. Required FSA and resulting solids content of CFF tailings.

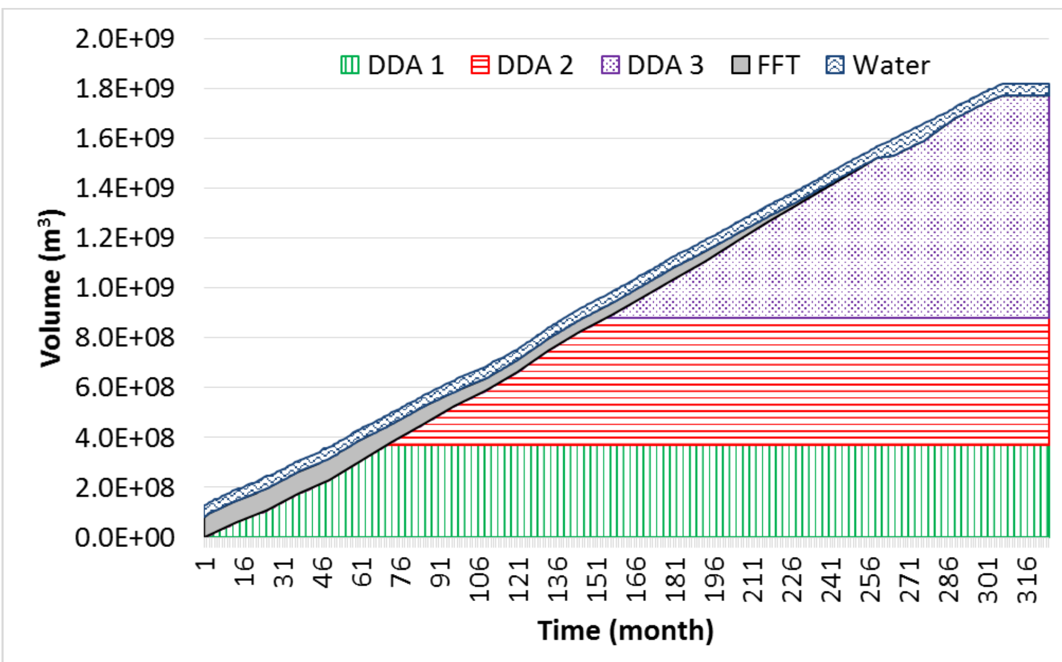


Figure 8.16. Deposit and fluid volumes for simulation CFF-2

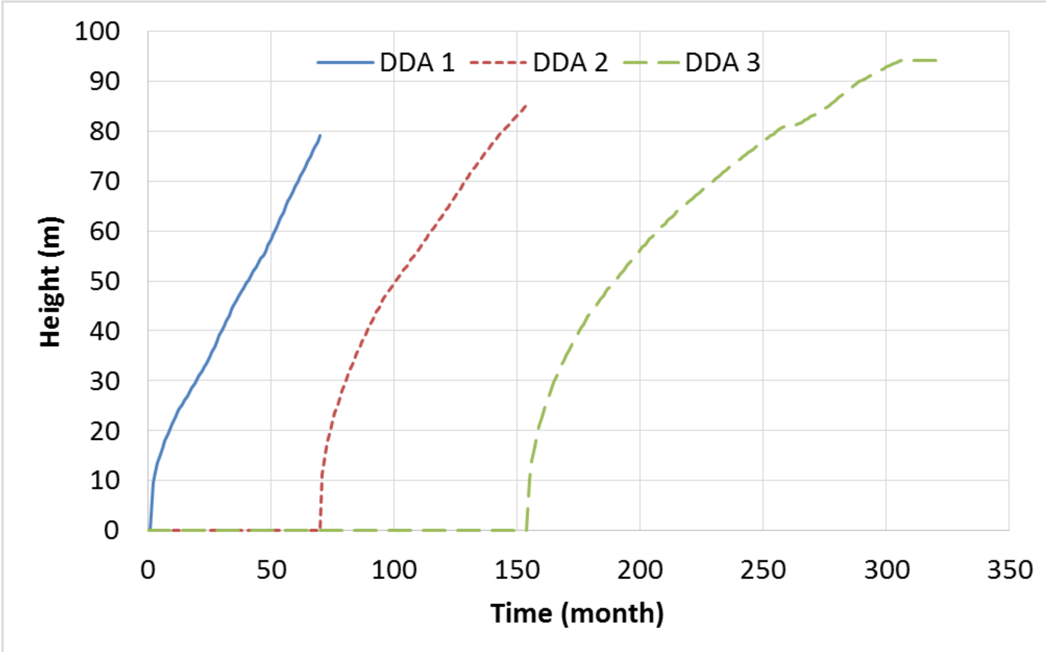


Figure 8.17. Deposit height for simulation CFF-2.

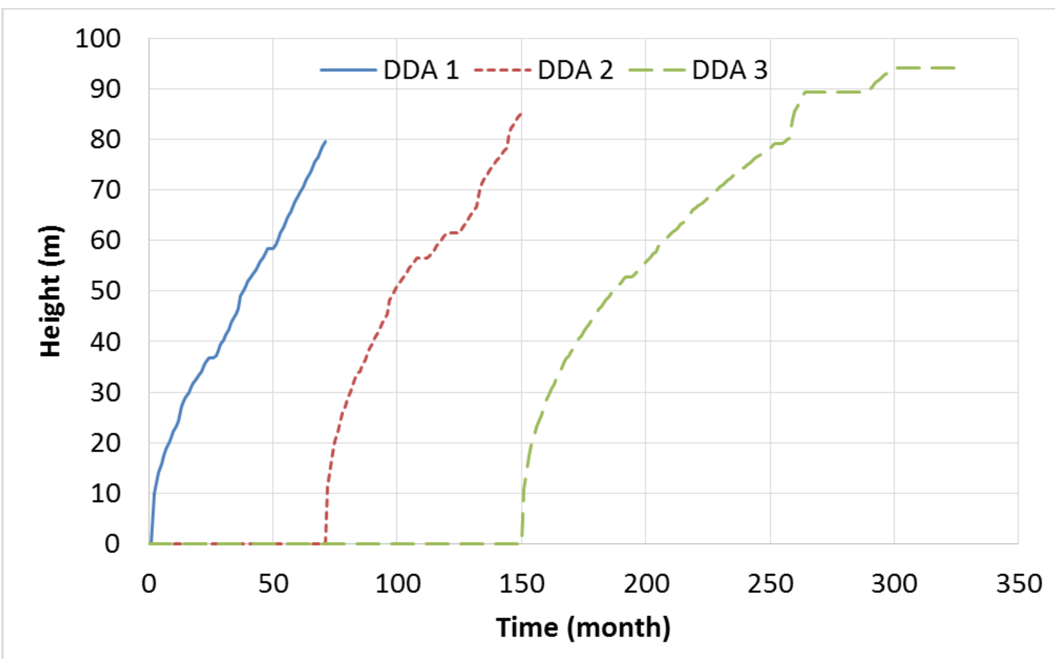


Figure 8.18. Deposit height for simulation CFF-3.

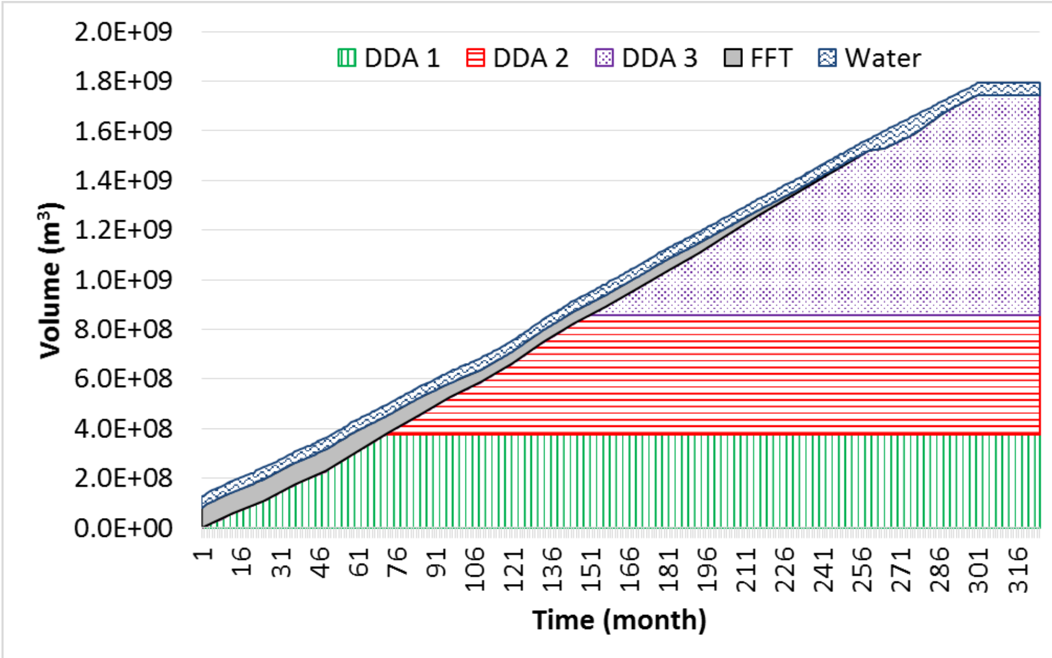


Figure 8.19. Deposit and fluid volumes for simulation CFF-3

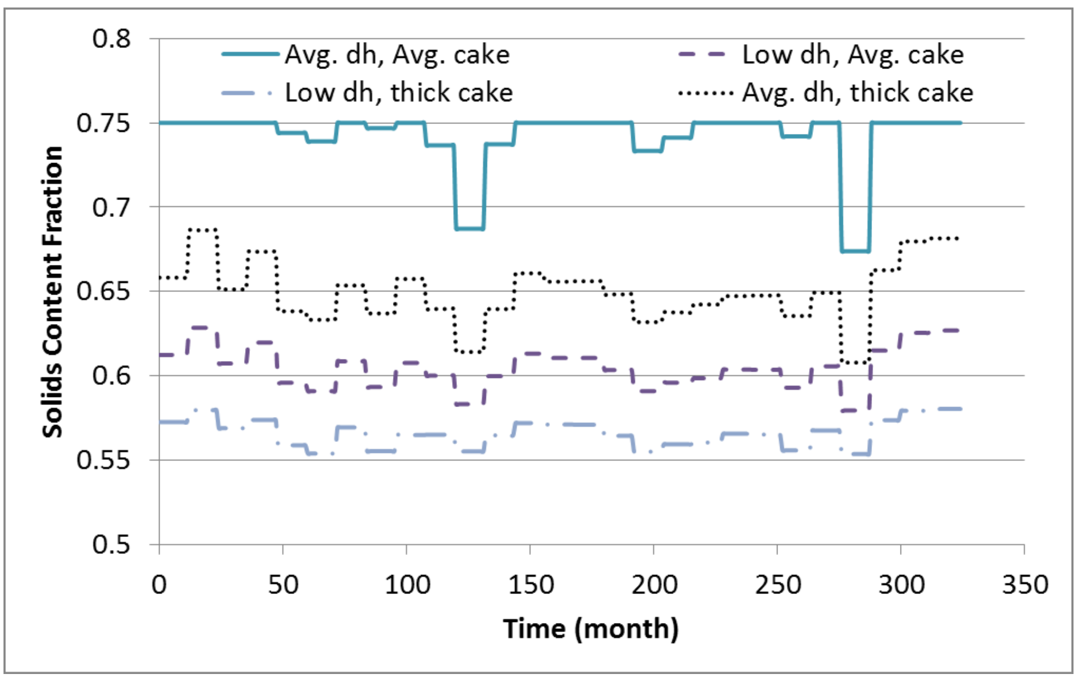


Figure 8.20. Variation of model CFF tailings solids content with pressure and cake thickness.

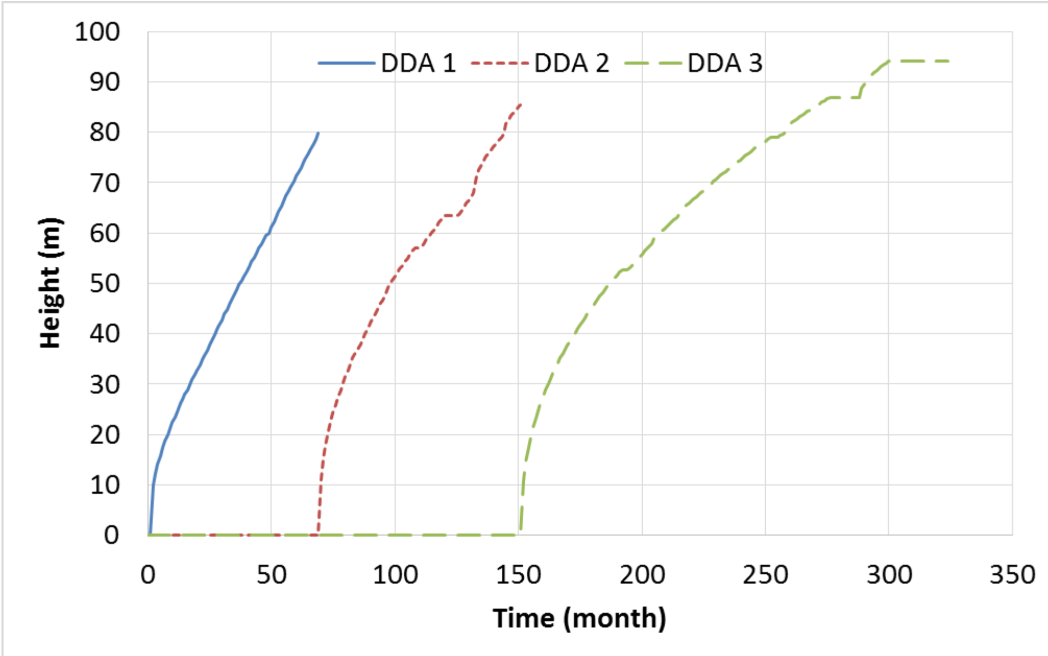


Figure 8.21. Deposit height for simulation CFF-5.

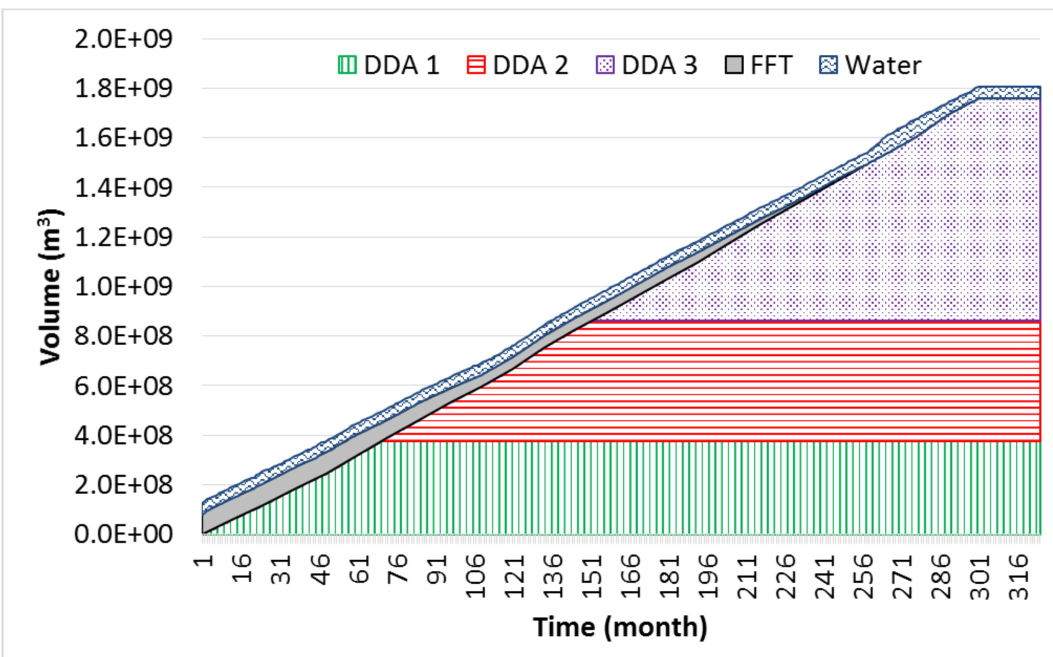


Figure 8.22. Deposit and fluid volumes for simulation CFF-5.

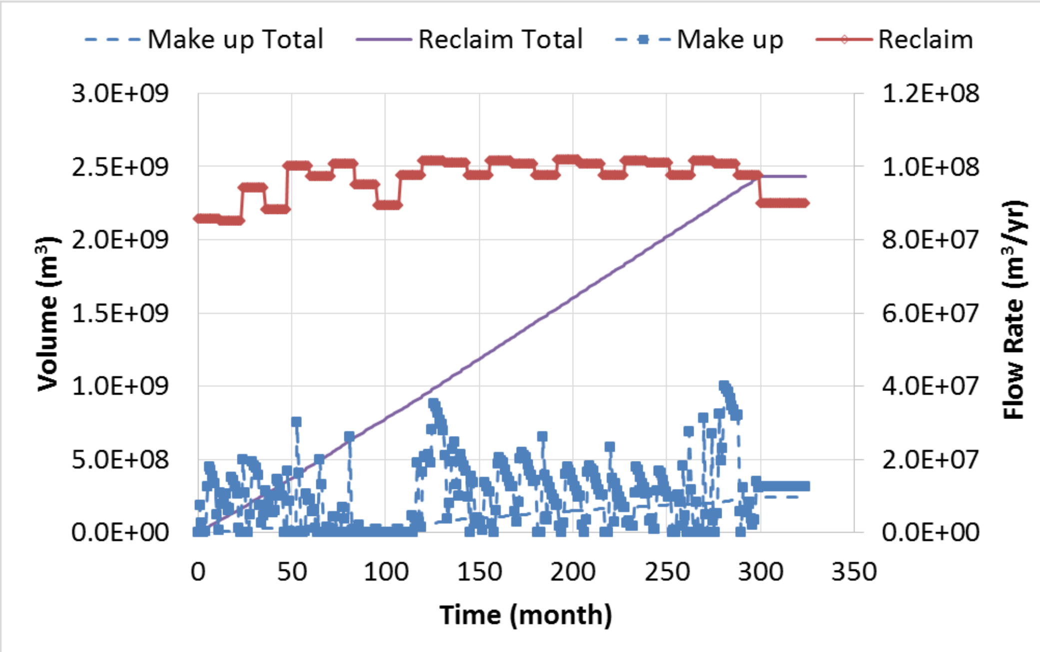


Figure 8.23. Process water reclaim and make up rates for simulation CFF-5.

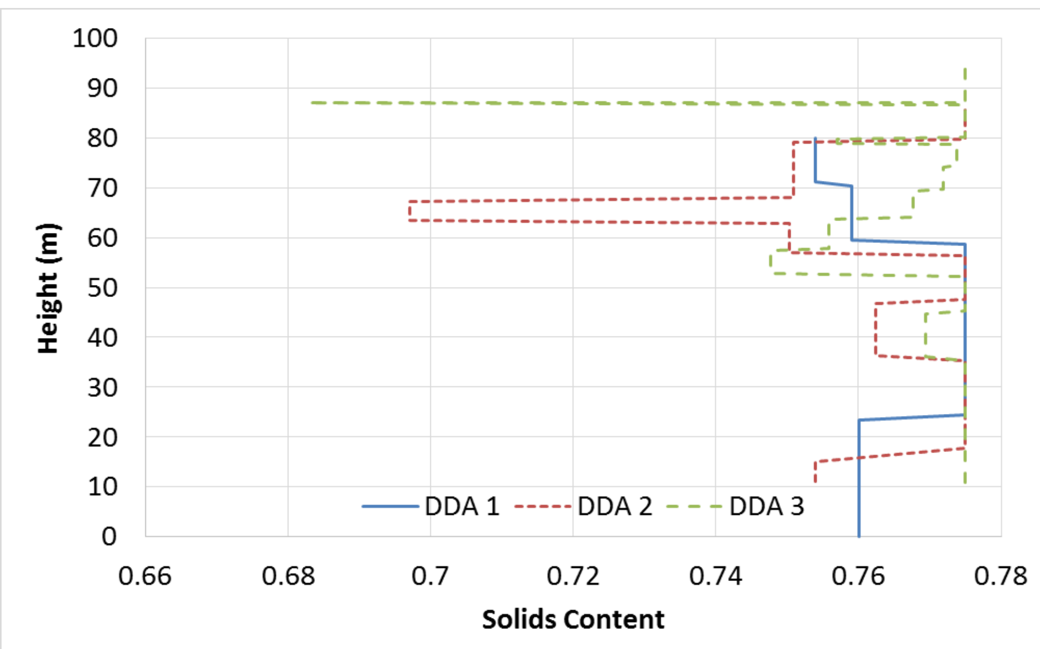


Figure 8.24. Solids content profiles for each DDA for simulation CFF-5.

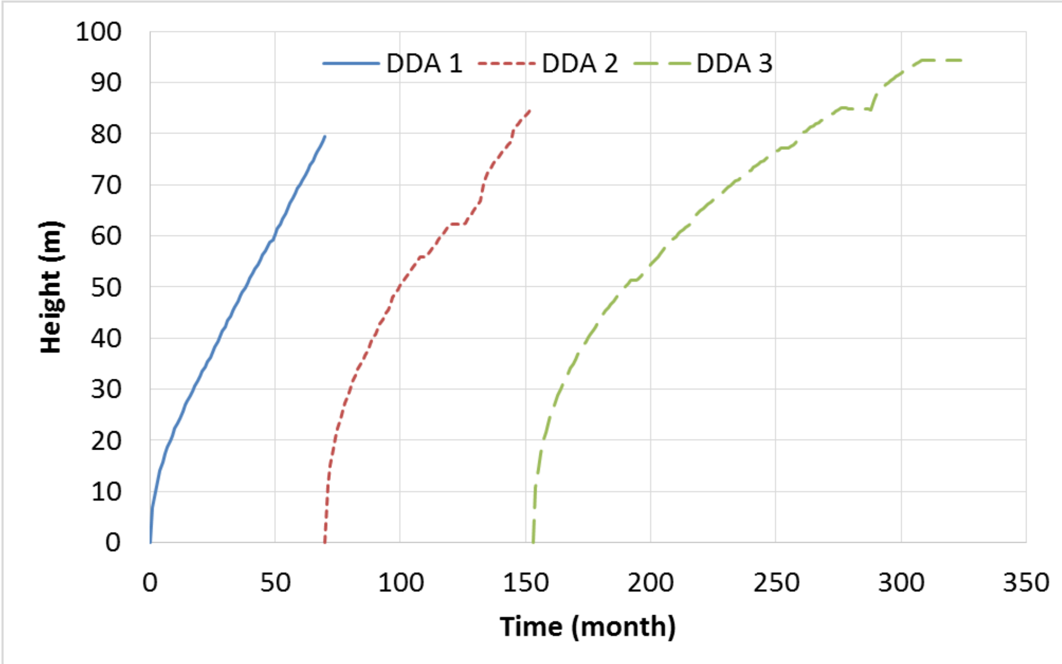


Figure 8.25. Deposit height for simulation CFF-6.

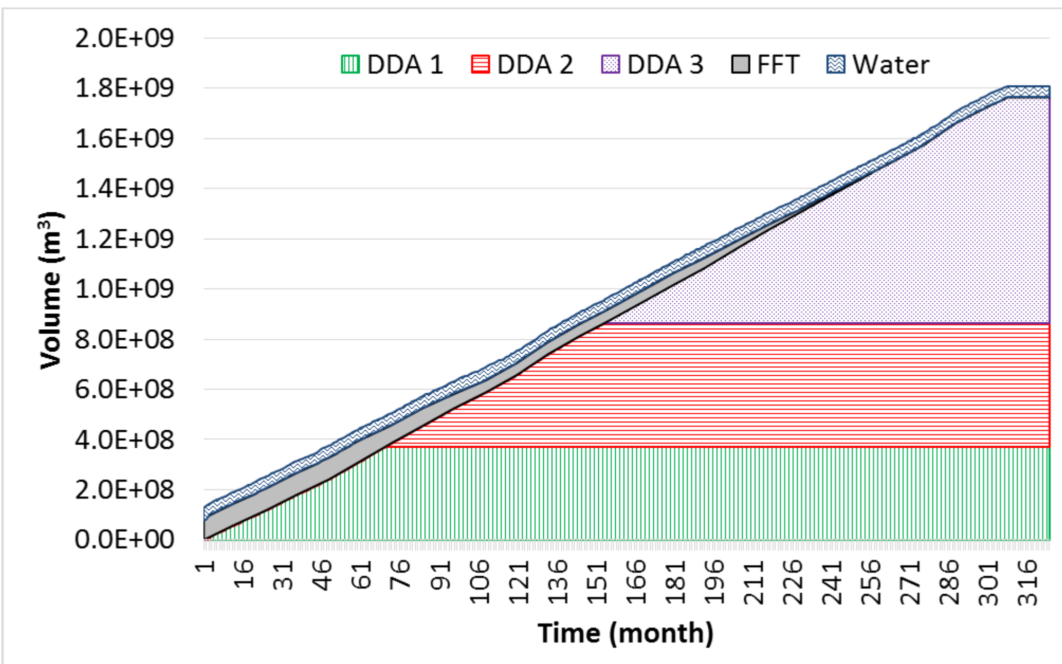


Figure 8.26. Deposit and fluid volumes for simulation CFF-6.

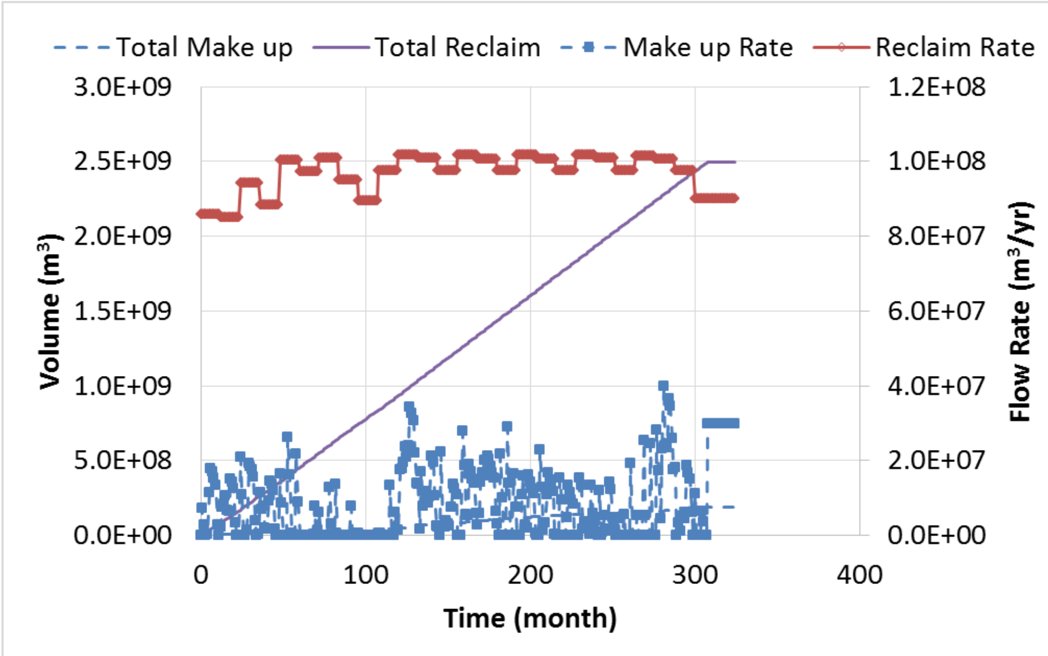


Figure 8.27. Process water reclaim and make up rates for simulation CFF-6.

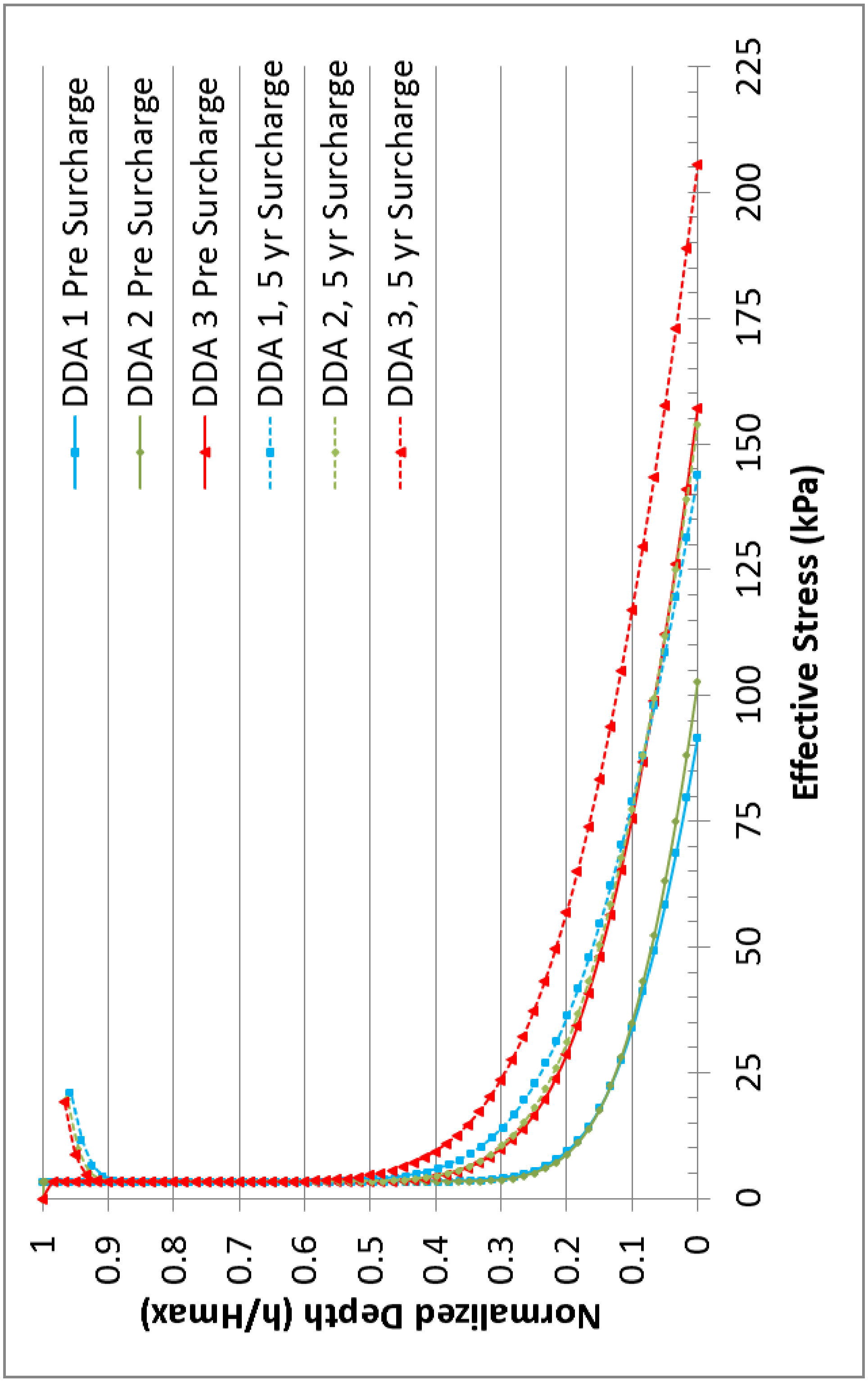


Figure 8.28. Effective stress profiles for CFF-6.

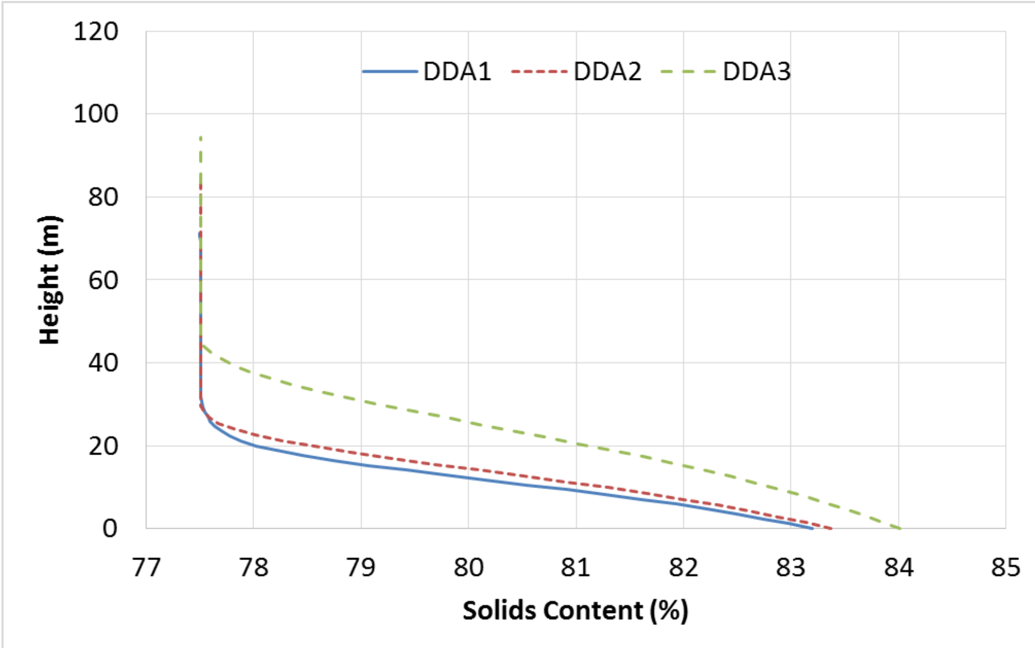


Figure 8.29. Solids content profiles for each DDA for simulation CFF-6.

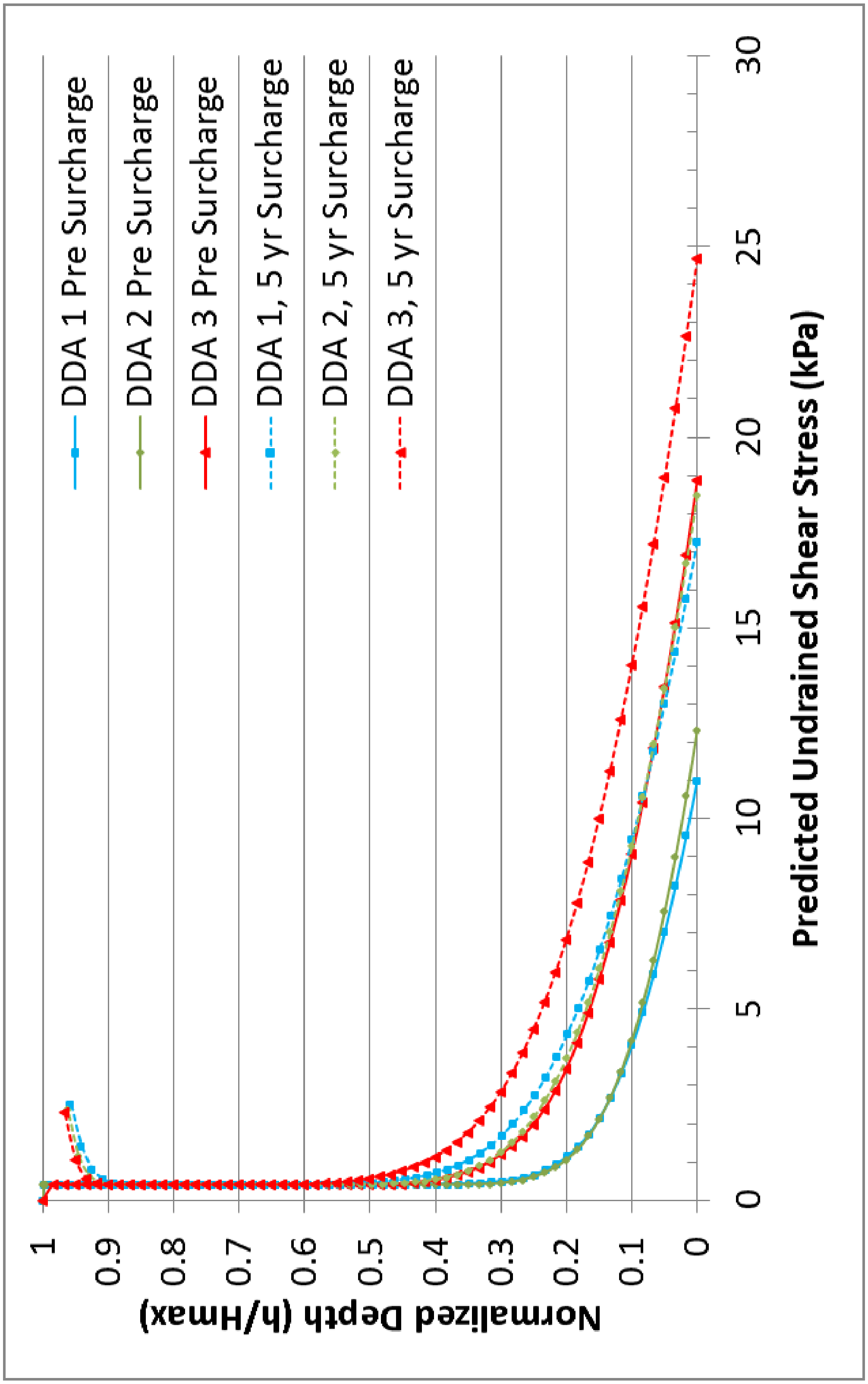


Figure 8.30. Predicted undrained shear strength for CFF-6.

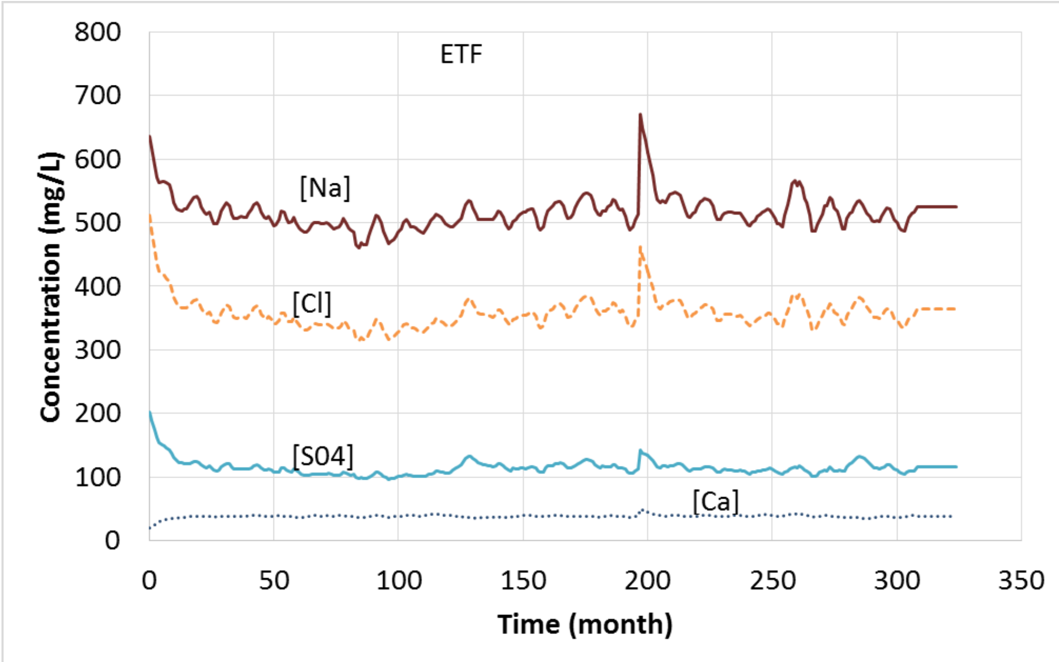


Figure 8.31. Chemical species concentration in the ETF for CFF-6.

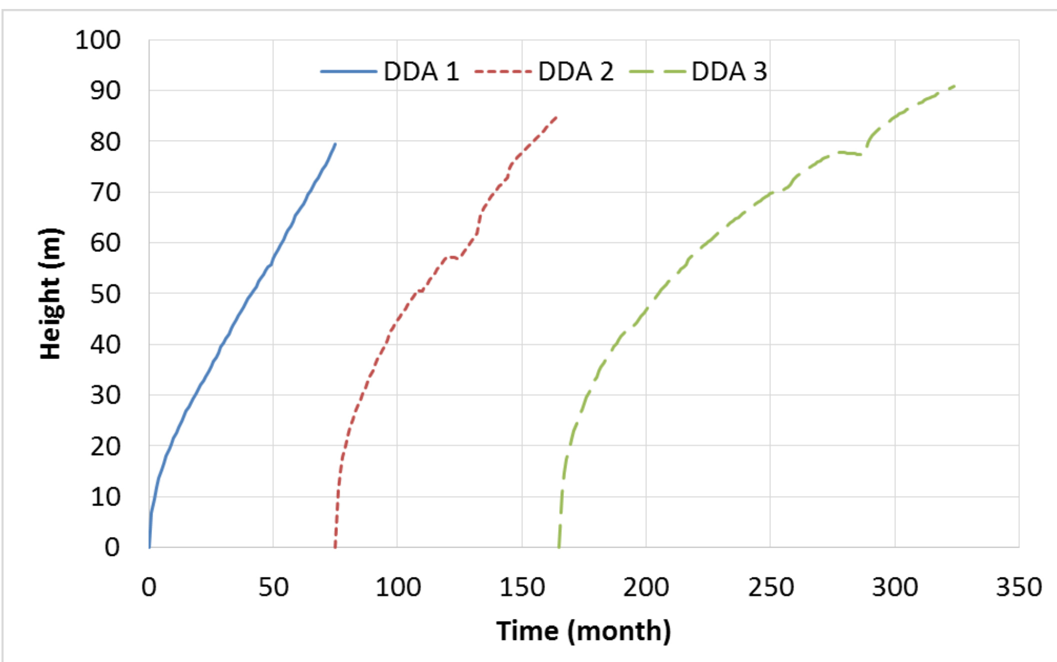


Figure 8.32. Deposit height for simulation CFF-7.

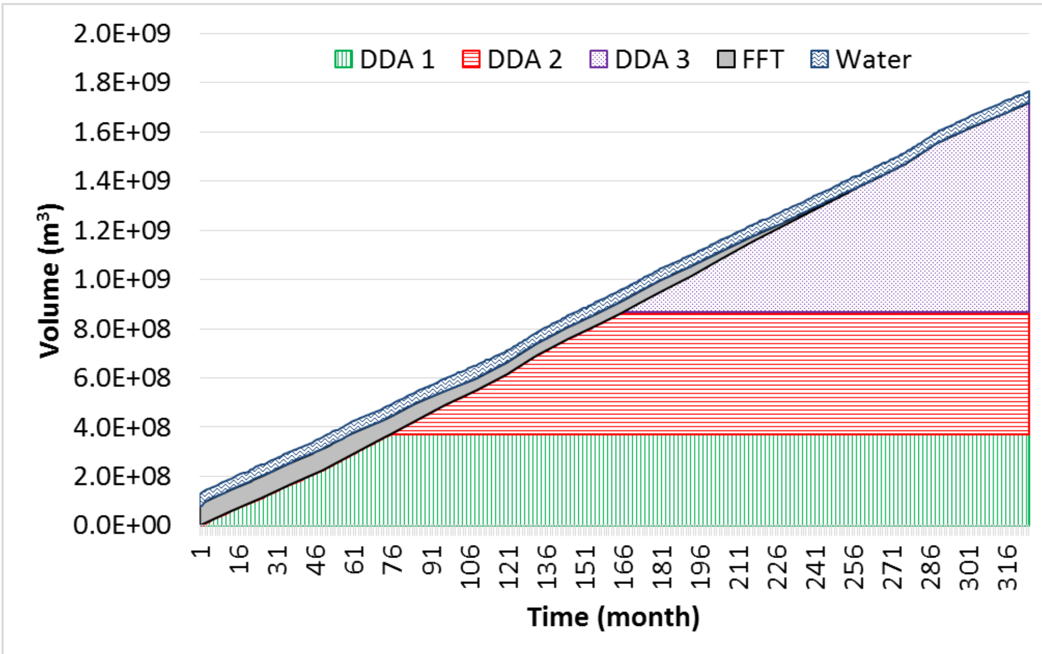


Figure 8.33. Deposit and fluid volumes for simulation CFF-7.

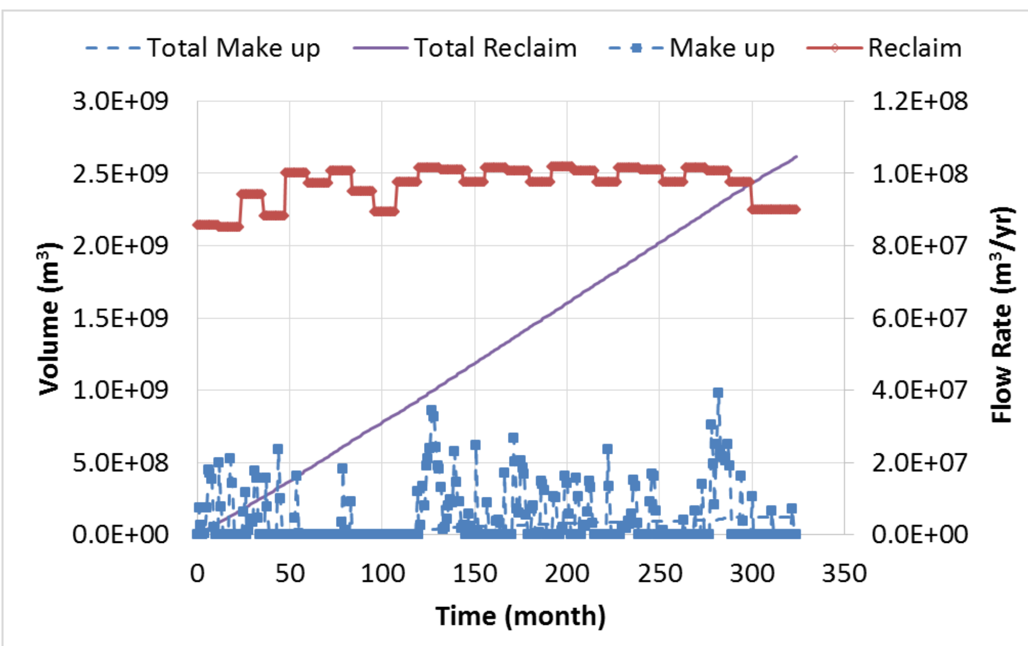


Figure 8.34. Process water reclaim and make up rates for simulation CFF-7.

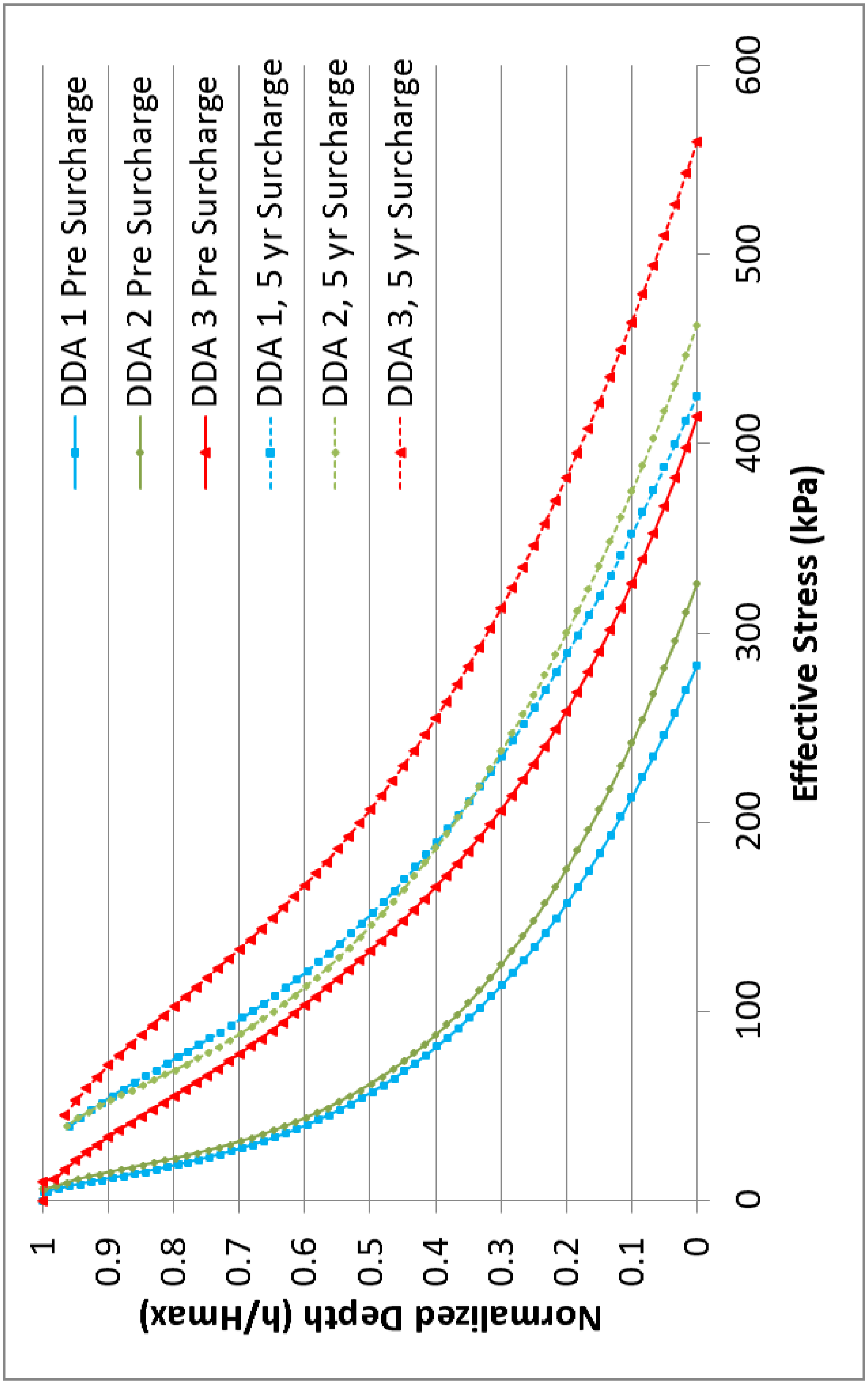


Figure 8.35. Effective stress profiles for CFF-7.

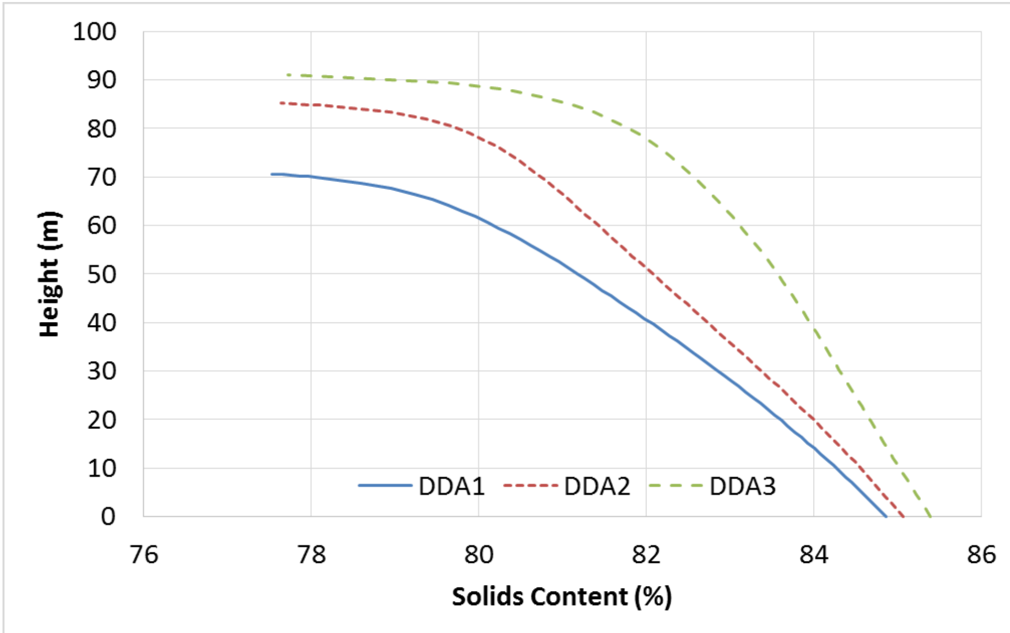


Figure 8.36. Solids content profiles for each DDA for simulation CFF-7.

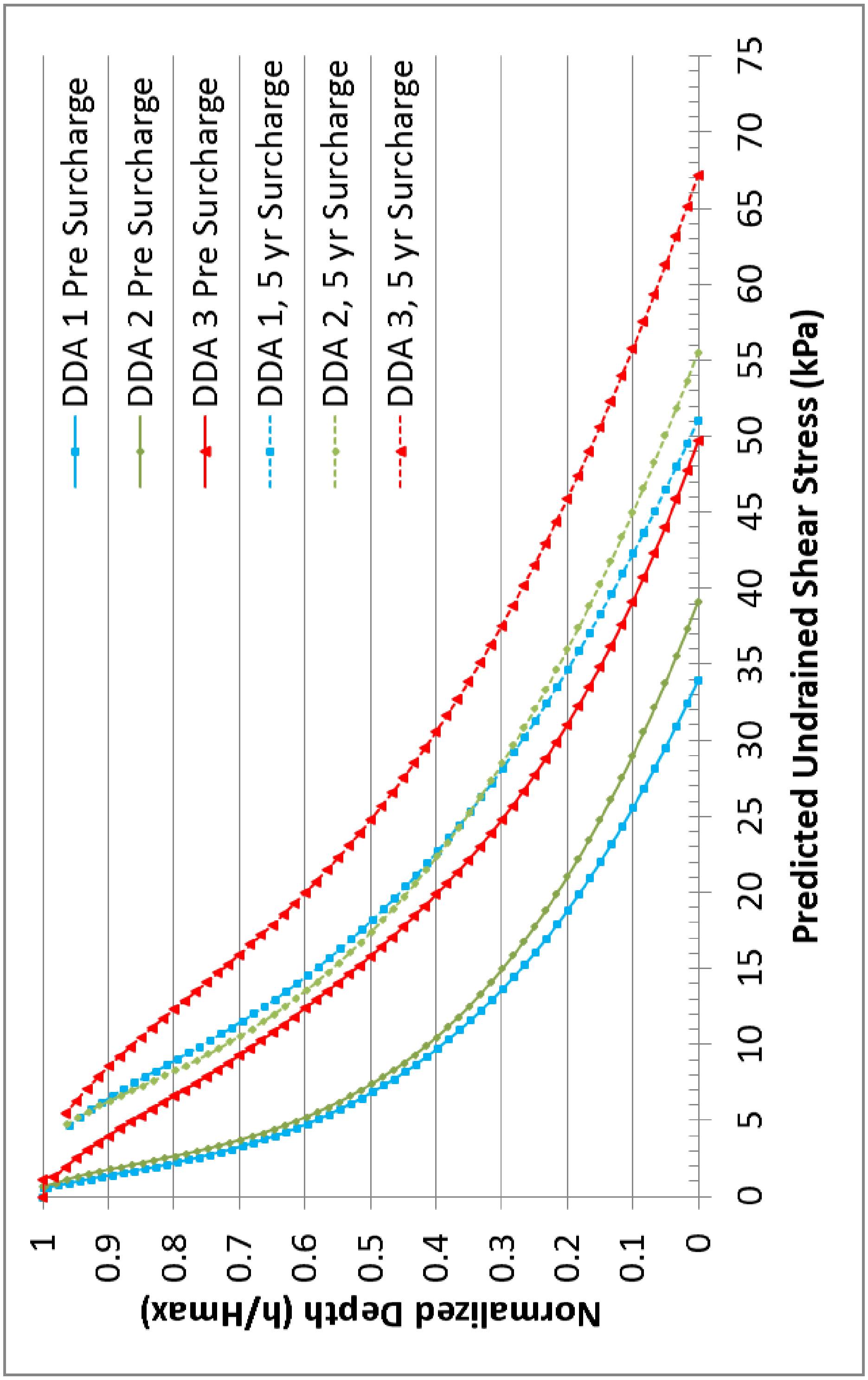


Figure 8.37. Predicted undrained shear strength for CFF-7.

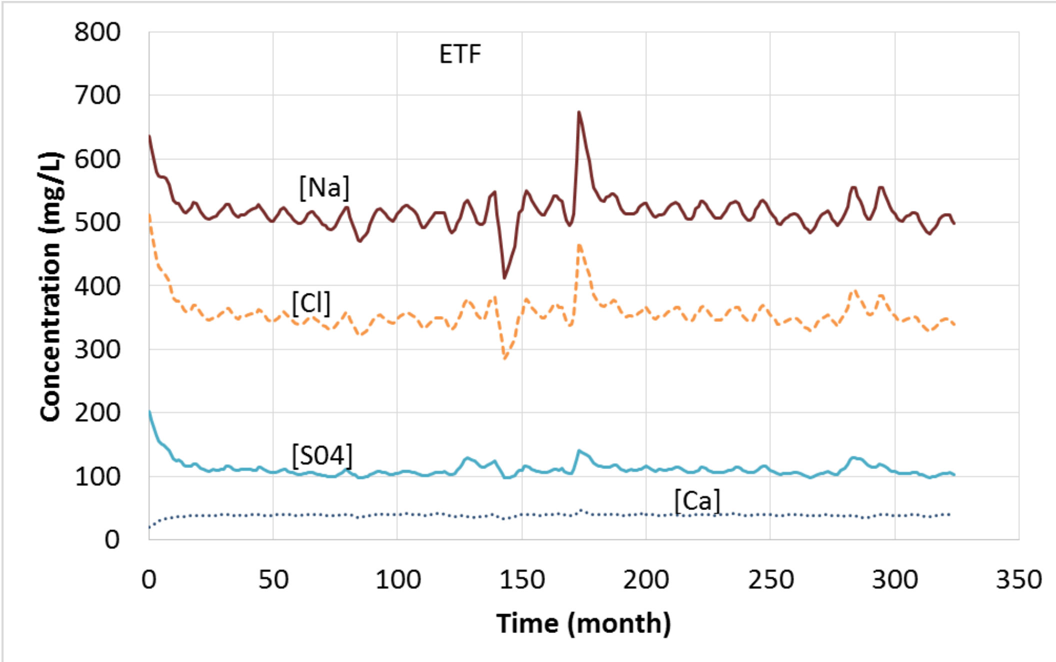


Figure 8.38. Chemical species concentration in the ETF for CFF-7.

8.9 References

Beier, N. and Sego, D. 2008. Dewatering of Oil Sands Tailings Using Cross Flow Filtration. 61st Canadian Geotechnical Conference and 9th joint IAH-CNC Groundwater Specialty Conference, Edmonton, Alberta, 22-24 September 2008. 8 pp.

Boswell, J., Davachi, M., and Sobkowicz, J. 2012. Oil Sands Tailings Technology Deployment Roadmap Project Report – Volume 5 – Component 4 Results. Report submitted Alberta Innovates – Energy and Environment Solutions. 26 June 2012. Pp 343.

Carrier III, W.D., Bromwell, L.G., and Somogyi, F. 1983. Design capacity of slurried mineral waste ponds. Journal of Geotechnical Engineering **109(**5): 699-716.

Chapuis, R.P. and Aubertin, M. 2003. On the use of the Kozeny-Carmen equation to predict the hydraulic conductivity of soils. Canadian Geotechnical Journal **40**: 616-628.

Ciulavu, M. 2008. Clay mineralogy of the Athabasca Oil Sands. Conrad Clay Mineral Workshop. Calgary, Alberta, May 1, 2008.

Devenny, D.W. 2010. A screening study of oil sand tailings technologies and practices. Report prepared for Alberta Energy Research Institute, March 2010. http://www.ai-ees.ca/media/6924/oil_sand_tailings_technologies_and_practices_a.pdf. [cited 4 April 2013].

Fitton, T. 2007. Tailings beach slope prediction. PhD Thesis, School of Civil, Environmental and Chemical Engineering, RMIT University, Melbourne, Australia.

Jeeravipoolvarn, S. 2010. Geotechnical behavior of in-line thickened oil sands tailings. PhD Thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, Alberta. pp. 431.

Jeeravipoolvarn, S., Scott, J.D., and Chalaturnyk, R.J. 2009. 10 m standpipe tests on oil sands tailings: long term experimental results and prediction. Canadian Geotechnical Journal, **46**: 875-888.

Masliyah, J., Xu, Z., and Czarnekci, J. 2011. Handbook on theory and practice of bitumen recovery from Athabasca oil sands – Volume 1: Theoretical Basis. Kingsley Publishing Service. pp. 468.

Matthews, J.G., Shaw, W.H., MacKinnon, M.D., and Cuddy, R.G. 2002. Development of composite tailings technology at Syncrude. International Journal of Surface Mining, Reclamation, and Environment, **16**(1): 24-39.

Miller, W.G., Scott, J.D. and Sego, D.C., 2011. Effect of extraction water chemistry on the consolidation of oil sands fine tailings. CIM Journal 1(2): 113-129.

Reeves, B. 1996. Environmental Impact Assessment for the Syncrude Canada Limited Aurora Mine. BOVAR Environmental, 1996.

Scott, J.D. 2014. Personal communication.

Sego, D.C. 2014. Personal communication.

Shook, C.A., Gilles, R.G., and Sanders, R.S. 2002. Pipeline Hydrotransport with Applications in the Oil Sand Industry. SRC Publication No. 11508-1E02. Saskatoon, Saskatchewan, Canada. 118 pp.

Sobkowicz, J.C. 2012. The oil sands tailings technology roadmap and action plan: Introduction and key recommendations. Third International Oil Sands Tailings Conference. Edmonton, Alberta. December 2-5, 2012. Pp. 13-23.

Sobkowicz, J.C. and Morgenstern, N.R. 2009. A geotechnical perspective on oil sands tailings. 13th International Conference on Tailings and Mine Waste, Banff, Alberta, 1-4 November 2009. University of Alberta, pp. xvii-xli.

Suthaker, N.N. and Scott, J.D. 1996. Measurement of Hydraulic Conductivity in Oil Sand Tailings Slurries. Canadian Geotechnical Journal, **33**(4): 642-653.

Syncrude Canada Ltd. 2010. Directive 074 Baseline Survey for Fluid Deposits Syncrude Mildred Lake and Aurora North, Year: Baseline (2010). Report submitted to Energy Resources Conservation Board, September 30, 2010. 53 pp.

Syncrude Canada Ltd. 2012. 2012 Annual Tailings Plan Submission Syncrude Aurora North Mine. Report submitted to Energy Resources Conservation Board, September 28, 2012. 43 pp.

Xu, Y., Dabros, T., and Kan, J. 2008. Filterability of oil sands tailings, Process Safety and Environmental Protection, **86**(4): 268-276.

Zhang, C. 2010. Cross Flow Filtration of Oil Sand Total Tailings. MSc Thesis submitted to the Department of Civil and Environmental Engineering, University of Alberta. 182 pp.

9 CONCLUSIONS

9.1 SUMMARY

This thesis presents a tailings management simulation model (TMSim) offering a new tool for tailings planners, regulators, and technology developers to evaluate tailings technologies in a virtual environment. The model was developed using publically available data and incorporates the major components of a tailings management system such as the extraction plant, tailings classification, pre- and post-depositional dewatering, the impoundment (dykes, containment, and process water storage) and the environment. To demonstrate the application of TMSim to a range of technologies and tailings plans, several cases were evaluated including a simple metal mine tailings plan and a complex oil sands operation with multiple deposition locations and technology options. Additionally, evaluation of flocculation based technologies for oil sands tailings management identified potential operational and reclamation challenges.

Mining companies manage tailings and impoundments through the implementation of tailings management systems that incorporate all aspects of tailings dewatering and associated storage facilities. These systems are constantly evolving as technology improves and mining and closure plans change due to the evolution of the economic and regulatory environment. Thus, alternative technologies and management practices are constantly under evaluation. Therefore, there is an increasing need for technology evaluation/screening tools. To assess a particular case, performance measures were outlined including storage capacity requirements, deposit characteristics including strength trajectories, process water quality and availability, and flexibility of the plan/technology. A dynamic simulation model (TMSim) was developed to evaluate tailings technologies and incorporate the various processes in a tailings management system. TMSim has the ability to simulate a mine and tailings system over time, demonstrate various outcomes by alternating management practices and technologies, and conduct sensitivity analyses. Given the complexity of tailings management systems, simplifications and assumptions were required in the development of the model. They provide a balance between the numerical complexity of the model with the availability of data and the objective of the modeling exercise. The TMSim model was validated against a metal mine plan to demonstrate it incorporates enough detail to provide a reasonable assessment of the performance measures.

The oil sands region is currently dominated by a wet landscape with several, large above grade containment structures storing fluid fine tailings and process water. In an effort to manage fine tailings and the associated risks to reclamation activities, the Alberta Government elected to regulate oil sand fine tailings. Technologies proposed by the industry to meet the new regulations include combinations of non-segregating (NST) or composite tailings (CT) technologies, coarse sand beaching, fine tailings dewatering techniques including centrifuges, thickened tailings, or chemical dewatering with strategic deposition, and final storage of residual fine tailings in end pit lakes (EPL). Therefore, the final reclaimed landscape for each of the mine sites reviewed will be a combination of terrestrial and aquatic landforms based on the proposed tailings management plans. During the review and evaluation of the oil sands tailings plans and technologies, potential challenges with flocculation based technologies were identified. The addition of flocculants to meet the regulatory strength performance targets may have implications on storage capacity and design, as well as the sensitivity and long-term geotechnical behaviour of flocculated dewatered fine tailings deposits. Mitigative measures may be required to reduce the impact on the overall mine and tailings plans.

Using Syncrude's Aurora North mine site and tailings plan as a guide, a model mine and tailings plan data set was simulated using TMSim. Only publically available data was utilized in the development of the mine plan resulting in some simplifications in the mine plan. The model tailings plan is based on beaching and cyclone classification of whole tailings, CT technology, and management of residual fine tailings with EPLs. These technologies represent the industries general approach to split and manage the fine and coarse tailings fractions separately. The model mine plan and technology assumptions incorporated into the TMSim were shown to be acceptable because of the reasonable agreement with the original Syncrude plan. These results demonstrate TMSim is able to simulate complex, multi-technology, multi-depositional environments and can be used as tool to evaluate design alternatives and technology options for oil sands mine applications.

A novel tailings dewatering technology (crossflow filtration, CFF) was then assessed by TMSim using the model oil sands data set. CFF offers a different approach to managing tailings whereby the coarse and fine fractions of the tailings are managed together without segregation. Whole tailings from the extraction process were dewatered by the CFF process and then deposited as stacks in the mined out pit. Additionally, the current inventory of fine tailings was spiked into the whole tailings prior to CFF dewatering. Filtrate from the CFF process can be recycled immediately. This filtrate fulfils up to two thirds of the yearly process water demand that would otherwise be satisfied from site runoff and make up water from off site sources. Immediate recycle of the water reduces the requirement to heat the recycle water thereby potentially lowering green house gas production. Since the CFF process is a bench scale technology, further testing is required to confirm and improve the CFF design and model assumptions such as dewatering rates and deposit behaviour. End of pipe tailings technologies such as filtration (i.e. CFF) offer the potential to have more terrestrial landforms and fewer below grade aquatic (i.e. wetlands) land forms. Simulations completed for the CFF technology have also demonstrated the application of TMSim for examining a novel technology.

Due to the simplifications and assumptions implemented during the development of TMSim, the model is not intended to be used as a final design tool. It's intended for the evaluation of technologies by diagnosing potential drawbacks and strengths, conducting sensitivity analyses and providing a baseline data set. Not only does TMSim provide the ability to evaluate tailings technologies, but the process of compiling the necessary model assumptions, inputs and UDFs can offer significant insight into the technology. The entire process of formulating a scenario and running the TMSim model helps the user to understand the vulnerabilities and limitations of a technology as demonstrated by the multiple scenarios developed for the CFF technology. All of this information can then be used to strategically guide and support technology development and resource expenditure.

9.2 DETAILED CONCLUSIONS

Conclusions by chapter are as follows:

Chapter 2. Tailings Behaviour and Management

- Management of tailings and waste rock are an integral part of all mining operations.
- Tailings management includes mechanical/chemical dewatering, transport to and construction of geotechnically sound impoundments, depositional behavior, and post depositional dewatering.
- The physical and chemical material properties of the tailings particles and slurry such as particle size, mineralogy, specific gravity, slurry density, pore fluid chemistry, and chemical additives, influence each of the above processes.
- The implementation of technologies depend upon the objectives of the operator/regulator or applicable regulations for the mine site.
- The TMS must also be dynamic to cope with a tailings facility whose geometry and operational considerations change over the life of the mine (decades).

Chapter 3. Tailings Management Simulation Model Development.

- A high level simulation model (TMSim) was developed using publically available data sources.
- It employs process-based, empirical, qualitative and conditional formulations to incorporate the major components of a tailings

management system such as the extraction plant, tailings dewatering Stage 1 (classification), Stage 2 (pre-deposition) and Stage 3 (post-deposition), the impoundment (containment, water cap etc.) and the environment.

- The extraction sub-model utilizes the mining plan, ore characteristics, and feedback (process water quantity and quality) from the other sub-models to calculate the quantity of concentrate extracted and tailings produced.
- Stage 1 dewatering sub-model incorporates both cyclone separation and beach deposition based on empirical formulations. The model also provides flexibility for the user to implement new Stage 1 formulations.
- The Stage 2 dewatering sub-model is highly flexible allowing the user to utilize the built in dewatering models or again implement new dewatering models. These models can be simple built in functions or complex 3rd party software models, depending on the level of detail available to the user and the requirement of the simulation exercise.
- The Stage 3 sedimentation sub-model is based on empirical formulations derived from experimental data.
- The Stage 3 consolidation sub-model incorporates a 3rd party software (FSConsol) as the consolidation modeling engine. The linkage between the TMSim and FSConsol allows the model to dynamically capture evolving tailings properties at each time step. Profiles of void ratio, σ', and S_u with depth are computed at each time step.
- Environmental dewatering processes are also incorporated into the Stage 3 dewatering process. Currently, freeze-thaw dewatering coupled with the Stage 2 consolidation sub-model is included in the TMSim.
- The impoundment sub-model requires the user to specify the stage curves *a priori*. Using the stage curves and tailings properties, a three dimensional deposition model coupled with the one dimensional consolidation sub-model delivers a representative tailings surface at each time step.

- The impoundment sub-model also tracks the various flows of process water in and out of the impoundment including Stage 2 and 3 release water, precipitation/evaporation, seepage, miscellaneous flows, and extraction reclaim.
- The chemical quality of the stored process water is also evaluated using a mixing model.
- At each time step, single or multiple dewatering technologies and deposition locations can be employed based on user input or model constraints.
- Simplifications and assumptions were required in absence of available data and to facilitate modeling.

Chapter 4. Tailings Management Simulation Model Implementation and Validation.

- Few models are available that assess the tailings management process as a system and they do not take into account the dynamic nature of the tailings management process.
- TMSim was developed to evaluate tailings dewatering technologies and management strategies to ensure reclamation and closure goals are met while balancing ongoing storage demand needs.
- An object orientated, systems dynamic modeling software called GoldSim was used as the "simulation engine" for TMSim coupled with Excel VBA, spreadsheet models and FSConsol.
- Individual TMSim model components for sedimentation, consolidation, and deposition were validated using experimental, analytical and numerical data sets.
- Using a metal mine tailings plan, a comparison between TMSim predictions and a design data set was also presented. The mass balance and performance measures predicted by the TMSim agreed well with the design data set.

- The TMSim model developed has demonstrated its validity for application to simulate tailings management systems and technologies.

Chapter 5. Oil Sands Tailings Management.

- An overview of the historical and planned tailings management plans for several oil sands mine sites was presented.
- The tailings management plans were influenced by constraints such as lease geometry, geology, quality of the ore body, and extraction process.
- More than one tailings technology was required to manage the large volumes of tailings arising from the oil sands mining operations.
- The mines plan to move forward with combinations of NST or CT technologies, coarse sand deposition, fines dewatering techniques including centrifuges, TT, or chemical dewatering with strategic deposition, and final storage of residual MFT in EPLs.
- The final reclaimed landscape for each of the mine sites reviewed will be a combination of terrestrial and aquatic landforms based on the proposed tailings management plans.

Chapter 6. Geotechnical Aspects of Flocculation-based Technologies for Dewatering Mature Fine Tailings.

- The oil sands industry faces challenges in finding practical methods to control and reduce the formation of fluid fine tailings or mature fine tailings (MFT).
- It was shown that these deposits of “MFT” behave like natural clay slurries and can be represented by Locat and Demers’ (1988) I_L versus remolded S_u relationship.
- In response to the ERCB’s Directive 074 in 2009, the oil sands industry has undertaken considerable research and development with polymer-based flocculation to augment dewatering and strength gain of the fine tailings stream.

- Analysis of flocculated tailings deposit data demonstrate that it is possible to meet regulatory requirements (S_u of 5 kPa) with polymer addition to fluid fine tailings.
- However, chemically-amended fine tailings can have lower storage efficiencies compared to untreated tailings if the latter can be dewatered to the same target S_u.
- Available data also suggests that these deposits may exhibit sensitive, metastable behavior upon deposition.
- Mitigative measures to ensure the stability of such flocculated MFT deposits can include mixing or storing the flocculated fines within overburden structures. Incorporating high water content flocculated fines into overburden structures may have an influence on the overall stability.
- These mitigation measures require double handling of material, increasing the overall cost of the process.
- Based on available data and published literature, flocculent addition to fine tailings may actually hinder the self-weight consolidation process through the development of a pre-consolidation pressure.
- For disposal scenarios that rely on drainage and self-weight consolidation (deep deposits), significant surcharges may be required to induce compression and dewatering of the underlying tailings deposit.
- Therefore, improvement in current understanding of the storage and DDA design implications, sensitivity and long-term geotechnical behaviour of flocculated dewatered fine tailings deposits is required.

Chapter 7. Application of a Tailings Management Simulation Model to an Oil Sands Mine.

- The TMSim modelling tool was successfully utilized to simulate a model oil sands mine and tailings plan based on CT tailings technology.
- Model mine plan assumptions and UDFs incorporated into the TMSim were shown to be acceptable because of the agreement with the Syncrude plan for preliminary simulations.

- After implementation of Stage 3 dewatering, an acceptable mass balance between the TMSim model and the Syncrude plan was achieved.
- The compressibility and saturated hydraulic conductivity of the CT deposits influenced the filling rate of the DDAs, the make up reclaim rate, and the predicted strength profile of the deposits. Improvements to these properties, such as increasing the target SFR of the CT mix design, would improve the deposit strength. However, decreasing the mass of fines incorporated into the CT deposits would lead to a greater overall FFT volume at the end of mining.
- Based on the agreement with the Syncrude tailings plan, the TMSim model was established to be an effective quantitative tool that can be used in the evaluation of technologies for oil sands mining operations.

Chapter 8. Assessing Cross Flow Filtration Technology with a Tailings Management Simulation Model.

- The CFF dewatering process provides an opportunity to deposit high density tailings stacks requiring minimal containment.
- Two thirds of the yearly process water demand can be satisfied by immediate recycle from the CFF process resulting in potentially lower green house gas production from heating water.
- Additionally, if FFT spiking is incorporated, existing inventories of FFT can be consumed and stored in the pore space of the CFF tailings.
- Further bench scale and pilot testing is required to confirm and improve the CFF design and model assumptions such as: the fines content-specific resistance to filtration relationship with or without flocculation; influence of bitumen content on filtration rates and potential filter membrane fouling; and tailings behavior such as deposit slope and consolidation parameters.
- Following deposition, potential mitigative measures may be required to improve dewatering may. The cost of utilizing these improvements may out way the benefits of implementing the CFF technology.

- The simulations completed for the CFF technology have demonstrated the relevance of the TMSim model for examining a novel technology.
- The process of compiling the necessary input data required by the TMSim model and the simulations themselves have provided significant insight into the CFF process and their impact on a mine plan.
- The TMSim simulations presented above provide a baseline for further refinement and sensitivity analyses of the technology and depositional scenarios.

9.3 RECOMMENDATIONS AND FUTURE WORK

It is recommended that future research on the improvement of the TMSim model focus on combining the Stage 3 dewatering processes into one unified model. Currently, sub-models for freeze thaw, desiccation, and consolidation are separated. The unified model should also have the ability to simulate multiple layers with different material properties. This would enhance the range of technologies and scenarios that TMSim could evaluate. The deposition model could also be enhanced to include a variable surface profile. The model presently utilizes a linear slope calculated at each time step. Incorporation of ice development within the tailings impoundments would extend the application of the TMSim model to mining operations in northern climates.

Additionally, the chemical modeling ability for evaluation of recycle water quality could be enhanced. The TMSim model employs a mixing model that is sufficient for conservative ion species and understanding trends in concentrations. Adding a chemical equilibrium sub model would allow for simulation of chemical reactions and adsorption/desorption processes. These processes are important to understand when the process water quality may influence the extraction process or environment. A zero-discharge criterion is currently included in the TMSim water balance. An option for releasing accumulated/treated water that meets minimum environmental criteria should also be included.

Given the potential implications on storage and stability, more investigation into the long-term geotechnical behaviour of flocculated dewatered fine tailings deposits is required. Research should look at the influence of feed tailings composition (i.e. fines and clay content), flocculants (type and dose), dewatering technology (i.e. centrifuge, freeze thaw, desiccation) on the deposit dewatering behavior (short and long term) and sensitivity of the deposit strength.

The TMSim simulations have also outlined several issues with the current understanding of the CFF technology. Improvements on the influence of feed tailings properties (i.e. fines/clay content, bitumen content) and operating conditions on the filtration rate from the CFF process are required. Additionally, studies are required to confirm the depositional behavior of the CFF tailings including the slope of the deposits, consolidation behavior and development of strength.

BIBLIOGRAPHY

Abraham, C. 1999. Evaporation and Evapotranspiration in Alberta. Report 1912-1985, Data 1912-1996. Water Sciences Branch, Water Management Division, Alberta Environmental Protection. January 1999.

Abusaid, A., Fear, C., McRoberts, E., and Wells, S. 2011. An update to the construction of the Suncor oil sands tailings Pond 5 cover. *In* Proceedings of the 15th Tailings and Mine Waste Conference, Vancouver, British Columbia, 6-9 November 2011. pp. 363-375.

Ahmed, I., Labelle, M., Brown, R. and Lahie, R. 2009. Paste pumping and deposition field trials and concepts on Syncrude's dewatered MFT (centrifuge cake). Tailings and Mine Waste 2009, Banff, Alberta, 1-4 November 2009. pp. 417-427.

Al, T.A. and Blowes, D.W. 1999. The hydrogeology of a tailings impoundment formed by central discharge of thickened tailings: implications for tailings management. Journal of Contaminant Hydrology, **38**(1999): 489-505.

Alberta Energy and Utilities Board (EUB). 2007. Imperial Oil Resources Ventures Limited. Application for an Oil Sands Mine and Bitumen Processing Facility (Kearl Oil Sands Project) in the Fort McMurray Area. Decision 2007-013. 27 February 2007.

Alberta Environment and Sustainable Resource Development (AESRD). 2013. Oil Sands Information Portal. Website accessed May 1, 2013 http://environment.alberta.ca/apps/osip/ .

Allen, E. 2008. Process water treatment in Canada's oil sands industry: I. Target pollutants and treatment objectives. Journal of Environmental Science, **7:** 123-138.

Australian Government. 2007. Tailings Management. Leading practice sustainable development program for the mining industry. Commonwealth of Australia. www.ret.gov.au/sdmining , accessed Aug 8, 2007.

Azam, S. and Scott, J.D. 2005. Revisiting the ternary diagram for tailings characterization and management. Geotechnical News, December 2005, 43-46.

Badiozamani, M. and Askari-Nasab, H. 2012a. Towards integration of oil sands mine planning with tailings and reclamation plans. Proceedings of Tailings and Mine Waste 2012. Keystone, Colorado, October 14-17, 2012, 609-622.

Badiozamani, M. and Askari-Nasab, H. 2012b. An integration of long-term mine planning, tailings and reclamation plans. 3rd international oil sands tailings conference. 103-110.

Banas, L. 1991. Thixotropic behavior of oil sands tailings sludge. M.Sc. thesis, Department of Civil Engineering, University of Alberta, Edmonton, Alberta. pp. 394.

Barbour, S.L., Wilson, G.W., and Arnaud, L.C. 1993. Evaluation of the saturated-unsaturated groundwater conditions of a thickened tailings deposit. Canadian Geotechnical Journal **30**: 935-946.

Barrientos, M. and Barrera, S. 2001. Modeling of tailings impoundment raising. 8th International Conference of Tailings and Mine Waste 2001. Colorado State University, Colorado. Pg 163-170.

Been, K. 1980. Stress strain behaviour of cohesive soil deposited under water. Ph.D. thesis, Oxford University, UK.

Beier, N. and Sego, D. 2007. The Oil Sands Tailings Research Facility. 60th Canadian Geotechnical Conference and 8th joint IAH-CNC Groundwater Specialty Conference, Ottawa, Ontario, 22-24 October 2007. 8 pp.

Beier, N. and Sego, D. 2008. Dewatering of Oil Sands Tailings Using Cross Flow Filtration. 61st Canadian Geotechnical Conference and 9th joint IAH-CNC Groundwater Specialty Conference, Edmonton, Alberta, 22-24 September 2008. 8 pp

Beier, N., Alostaz, M. and Sego, D. 2009. Natural Dewatering Strategies for Oil Sands Fine Tailings. Tailings and Mine Waste 2009 Conference, Banff, Alberta, 1-4 November 2009. Pp. 845-858.

Beier, N.A., Wilson, G.W., Dunmola, A., and Sego, D.C. 2013. Impact of flocculation-based dewatering on the shear strength of oil sands fine tailings. Canadian Geotechnical Journal **50**(9): 1001-1007.

Ben-Awuah, E. and Askari-Nasab, H. 2011. Oil sands mine planning and waste management using mixed integer goal programming. International Journal of Mining, Reclamation and Environment, **25(**3): 226-247,
DOI: 10.1080/17480930.2010.549656

Ben-Awuah, E. and Askari-Nasab, H. 2012. A mathematical programming model for oil sands production scheduling and waste management. Proceedings of Tailings and Mine Waste 2012. Keystone, Colorado, October 14-17, 2012, 593-609.

Ben-Awuah, E., Askari-Nasab, H., and Awuah-Offei, K. (2012). Production scheduling and waste disposal planning for oil sands mining using goal programming. Journal of Environmental Informatics, **20**(1): 20-33.

Beuth, L:, Dykstra, C., and Jacobs, W. 2014. Modelling of hydraulic sand capping on creeping ultra-soft tailings. 4th International Oil Sands Tailings Conference, Edmonton, Ab. 7-10 December 2014, pp. 10.

Blight, G. 2010. Geotechnical Engineering for Mine Waste Storage Facilities. CRC Press, Taylor and Francis Group. Pp 634.

Blight, G., Thomson, R.R., and Vorster, K. 1985. Profiles of hydraulic-fill tailings beaches and seepage through hydraulically sorted tailings. Journal of South African Institute of Mining and Metallurgy. **85**(5): 157-161.

Boratynec, D.J. 2003. Fundamentals of rapid dewatering of composite tailings. MSc Thesis. Department of Civil and Environmental Engineering, University of Alberta, Edmonton, Alberta. pp. 280.

Boswell, J.E.C. and Sobkowicz, J.C. 2010. Environmental assistance for tailings disposal. 63rd Canadian Geotechnical Conference, Calgary, Alberta, 12-16 September 2010, pp. 665-669.

Boswell, J., Davachi, M., and Sobkowicz, J. 2012. Oil Sands Tailings Technology Deployment Roadmap Project Report – Volume 5 – Component 4 Results. Report submitted Alberta Innovates – Energy and Environment Solutions. 26 June 2012. Pp 343.

Boxhill, L. and Hooshiar, A. 2012. A review of design and operations considerations associated with thickening technologies and their application to oil sand tailings. 3rd International Oil Sands Tailings Conference, Edmonton, Alberta, 3-5 December 2012, pp. 237-247.

Bromwell, L.G. 1984. Consolidation of mining wastes. Sedimentation consolidation models: predictions and validation: In proceedings of a symposium, R.N. Yong and F.C. Townsend Eds, ASCE, Newyork: 275-295.

Burger, R. and Concha, F. 2001. Settling velocities of particulate systems: 12. Batch centrifugation of flocculated suspensions. International Journal of Mineral Processing **63**(2001): 115-145.

Burns, R., Cuddy, G., and Lahaie, R. 1993. Dewatering of fine tailings by natural evaporation. Fine Tailings Symposium, Environment Canada, Edmonton, Alberta, 4-7 April 1993, 36 pp.

Bussiere, B. 2007. Colloquium 2004: Hydrogeotechnical properties of hard rock tailings from metal mines and emerging geoenvironmental disposal approaches, Canadian Geotechnical Journal, **44**(9): 1019-1052.

Canadian Natural Resources Ltd. (CNRL). 2010. Horizon tailings management plan. 13-RPT-TA-0001 Revision 2. November 2010.

Canadian Association of Petroleum Producers (CAPP). 2012. Crude Oil, Forecast, markets and pipelines.. Website accessed Jan. 16, 2013. http://www.capp.ca/forecast/Pages/default.aspx

Carrier, W.D., III, Bromwell, L.G., and Somogyi, F. 1983. Design Capacity of Slurried Mineral Waste Ponds. Journal of Geotechnical Engineering, **109**(5): 699-716.

Carrier, W.D. III, Scott, J.D., Shaw, W.H., and Dusseault, M.B. 1987. Reclamation of Athabasca oil sand sludge. Proceedings of Geotechnical Practice for Waste Disposal, Ann Arbor, MI., 15-17 June 1987 pp. 377-391.

Chalaturnyk, R.J., Scott, D., and Ozum, B. 2002 Management of oil sands tailings. Petroleum Science and Technology, **20**(9&10):1025-1046.

Chapra, S.C. and Canale, R.P. 1998. Numerical Methods for Engineers. 3rd Edition. Mcgraw Hill. 924 pages.

Chapuis, R. and Aubertin, M. 2001. A simplified method to estimate saturated and unsaturated seepage through dikes under steady-state conditions. Canadian Geotechnical Journal, **38**: 1321-1328.

Chapuis, R.P. and Aubertin, M. 2003. On the use of the Kozeny-Carmen equation to predict the hydraulic conductivity of soils. Canadian Geotechnical Journal **40**: 616-628.

Chu, A., Paradis, T., Wallwork, V., and Hurdal, J. 2008. Non segregating tailings at the Horizon oil sands project. First International Oil Sands Tailings Conference, Edmonton, Alberta, 7-10 December 2008. 10 pp.

Consortium of Tailings Management Consultants (CTMC), 2012. Oil sands tailings technology deployment roadmaps: Project report. Report prepared for AI-EES and the Oil Sands Tailing Consortium (OSTC). June 29, 2012. 5 Volumes. http://www.ai-ees.ca/reports.aspx , accessed May 1, 2013.

Coulson, J. M., J. F. Richardson, J. R. Backhurst and J. H. Harker, "Particle Technology and Separation Processes,"Chem. Eng. Vol. 2, 4th ed. Pergamon, Chapter 7 (1991).

Ciulavu, M. 2008. Clay mineralogy of the Athabasca Oil Sands. Conrad Clay Mineral Workshop. Calgary, Alberta, May 1, 2008.

Davies, M.P., and Rice, S. 2001. An alternative to conventional tailings management - "dry stack" filtered tailings. Proceedings of the International Conference on Tailings and Mine Waste '01, Vail, Colorado, pp 411-419

Dawson R.F., and Sego D.C. 1993. Design concepts for thin layered freeze-thaw dewatering systems. 46th Canadian Geotechnical Conference, Saskatoon, 1993. pp. 283-288.

Dawson, R.F., Sego, D.C., Pollock, G.W., 1999. Freeze–thaw dewatering of oil sands fine tails. Canadian Geotechnical Journal **36**(4): 587–598.

Devenny, D.W. 2010. A screening study of oil sand tailings technologies and practices. Report prepared for Alberta Energy Research Institute, March 2010. http://www.ai-ees.ca/media/6924/oil_sand_tailings_technologies_and_practices_a.pdf. [cited 4 April 2013].

Dixon-Hardy, D.W. and Engels, J.W. 2007. Guidelines and recommendations for the safe operation of tailings management facilities. Environmental Engineering Science, **24**(5): 625-637.

Dunmola, A. 2012. Personal Communication

Environment Canada (EC). 2009. National Climate Data and Information Archive. Accessed May 11, 2009, www.climate.weatheroffice.ec.gc.ca

Engels, J. 2006. An expert management system for surface tailings storage. PhD Thesis, Mining, Quarry and Mineral Engineering, University of Leeds.

Fair, A. 2008. Past, Present, Future – Industry Perspective. First International Oil Sands Tailings Conference, Edmonton, AB, Canada, 7-10 December 2008. Presentation Only.

Fair, A.E. 2012. Canada's Oil Sands: A collaborative approach to addressing oil sand tailings. World Heavy Oil Congress, Aberdeen, Scotland, 10-13 September 2012, 10 pp.

Fair A.E. and Beier, N.A. 2012. Keynote Paper: Collaboration in Canada's Oil Sands: Fluid Fine Tailings Management. In Proceedings of the 3rd International Oil Sands Tailings Conference, Edmonton, Alberta, 3-5 December 2012, pp. 3-12.

Farnand, B. and Sawatzky, H. 1988. Microfiltration dewatering of deoiled tailings pond sludge. Eastern Oil Shale Symposium, Lexington, Kentucky. 30 November to 2 December 1988 Institute for Mining and Minerals Research, University of Kentucky, Lexington, Kentucky.

Fitton, T. 2007. Tailings beach slope prediction. PhD Thesis, School of Civil, Environmental and Chemical Engineering, RMIT University, Melbourne, Australia.

Flintoff, B.C., Plitt, L.R., and Turak, A.A. 1987. Cyclone modeling: A review of present technology, CIM Bulletin, **80**: 39-50.

Fredlund, M.D., Fredlund, D.G., Houston, S.L., and Houston, W. 2003. Assessment of unsaturated soil properties for seepage modeling through tailings and mine waste. Tailings and Mine Waste '03. October 12-15, 2003, Vail, Colorado, USA. pp. 149-157.

FTFC (Fine Tailings Fundamentals Consortium), 1995. Advances in oil sands tailings research, Alberta Department of Energy, Oil Sands and Research Division, Edmonton, Alberta, Canada.

Gibson, R.E., England, G.L. and Hussey M.J.L. 1967. The Theory of One-dimensional Consolidation of Saturated Clays. I. Finite Non-linear Consolidation of Thin Homogeneous Layers. Géotechnique, 17(3): 261–273.

Government of British Columbia, 2015. Report on Mount Polley tailings storage facility breach. Independent expert engineering investigation and review panel. Province of British Columbia.

Halog, A. and Chan, A. 2008. Developing a dynamic systems model for the sustainable development of the Canadian oil sands industry. International journal of Environmental Technology and Management, **8**(1): 3-22.

Holtz, R.D. and Kovacs, W.D. 1981. An Introduction to Geotechnical Engineering. Prentice-Hall, Englewood Cliffs, NJ, 733 p.

Houlihan, R. and Mian, H. 2008. Past, Present, Future Tailings – Regulatory Perspectives. First International Oil Sands Tailings Conference, Edmonton, Alberta, 7-10 December 2008. 8 pp.

Hyndman, A. and Sobkowicz, J. 2010. Oil Sands Tailings: Reclamation Goals and the State of Technology. 63rd Canadian Geotechnical Conference, Sept 12 - 16, 2010, Calgary, Alberta, 642 – 655.

Imai, G. 1981. Experimental studies on sedimentation mechanism and sediment formation of clay materials. Soils and Foundations, **21**(1): 7-20.

Imperial Oil Resources Ventures Ltd. (Imperial). 2010. 2010 Annual tailings plan submission to ERCB. 30 September 2010.

Jeeravipoolvarn, S. 2005. Compression Behaviour of Thixotropic Oil Sands Tailings. M.S.c. thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, AB.

Jeeravipoolvarn, S. 2010. Geotechnical behavior of in-line thickened oil sands tailings. PhD Thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, Alberta. pp. 431.

Jeeravipoolvarn, S., Scott, J.D., and Chalaturnyk, R.J. 2009. 10 m standpipe tests on oil sands tailings: long term experimental results and prediction. Canadian Geotechnical Journal, **46**: 875-888.

Jewell, R.J. and Fourie, A.B. 2006. Paste and thickened tailings – A guide. 2nd Edition. Australian Centre for Geomechanics, Nedlands, Western Australia, Australia. Pp. 257.

Johnson, R.L., Bork, P., Allen, E.A.D., James, W.H., and Koverny, L.1993. Oil sands sludge dewatering by freeze-thaw and evapotranspiration. Alberta Conservation and Reclamation Council Report No. RRTAC 93-8, 247 p.

Kabwe, L.K., Scott, J.D, Sorta, A.R, Wilson, W., Ulrich, A., Jeeravipoolvarn, S., and Kone, M. 2013. Geotechnical characteristics of the mature fine tailings in a 30-year 10-m standpipe experiment. GeoMontreal2013, Montreal, Quebec, 29 September to 3 October 3 2013, pp. 7.

Kalantari, S. 2011. Linkage of Annual Oil Sands Mine Plan to Composite Tailings Plan. MSc Thesis, Department of Civil and Environmental Engineering, University of Alberta.

Kaminsky, H., Omotoso, O., Etsell, T., and Ivey, D. 2009. Distribution of Clay Minerals in the Process Streams Produced by the Extraction of Bitumen from Athabasca Oil Sands. Canadian Journal of Chemical Engineering **87**(2): 85-93.

Kolstad, D., Dunmola, A., Dhadli, N., O'Kane, M., Song, J. and Masala, S. 2012. Towards the improvement in geotechnical performance of atmospheric fines drying (AFD) deposits at Shell's Muskeg River Mine. 3rd International Oil Sands Tailings Conference, Edmonton, Alberta, 3-5 December 2012, pp. 411-420.

Konrad, J.M., Morgernstern, N.R., 1980. A mechanistic theory of ice lens formation in fine-grained soils. Canadian Geotechnical Journal **17**(4): pp 473–486.

Kossik, R and Miller, I. 2004. A probabilistic total system approach to the simulation of complex environmental systems. Proceedings of the 2004 Winter Simulation Conference. Ingalls, R.G., Rossetti, M.D., Smith, J.S., and Peters, B.A. eds. December 5-8, 2004, Washington, D.C. 5 pp.

Kupper, A. 1991. Design of Hydraulic Fill. PhD thesis. University of Alberta. Edmonton, Alberta, Canada.

Kynch, G.J. 1952. A theory of Sedimentation. Transactions of Faraday Society, **48**: 166-176.

Lahaie, R. 2008. Syncrude Canada Ltd. - New Tailings Concepts. First International Oil Sands Tailings Conference, Edmonton, AB, Canada, 7-10 December 2008 pp. 17. Presentation Only.

Lee, K. 1979. An analytical and experimental study of large strain soil consolidation. DPhil, University of Oxford.

Leoni, M., Karstunen, M., Vermeer, P.A. 2008. Anisotropic creep model for soft soils. Géotechnique, **58**(3): 215-226.

Leroy, J.C.. 1972. How to establish and maintain growth in Canada-cold winters and short growing seasons. In Tailings disposal today. Edited by Aplin, C.L. and Argalls, G.O. Miller Freeman, San Francisco. pp. 411-449.

Locat, J. and Demers, D. 1988. Viscosity, yield stress, remolded strength, and liquidity index relationships for sensitive clays. Canadian Geotechnical Journal, **25**: 799-806. doi: 10.1139/t88-088.

Longo, S., Francoeur, R., Labelle, M., and Wislesky, I. 2011. Oil sands tailings dewatering – can it be done? Paste 2011, Fourteenth International Seminar on Paste and Thickened Tailings, Perth, Australia, 5-7 April 2011. Pp. 213-224.

Lord, E.R.F. and B.A. Isaac, 1989. Geotechnical investigations of dredged overburden at Syncrude oil sand mine in northern Alberta, Canada. Canadian Geotechnical Journal **26**: 132-153.

Lord, E.R.F., D.W. Mimura and J.D. Scott, 1991. Disposal of oil sand fine tails (sludge) in overburden waste dumps. IN: Proceedings of the First Canadian Conference on Environmental Geotechnics, Montreal, Quebec. 129-138

Lord, E.R., Maciejewski, W. Cymerman, G., and Lahaie, R. 1993. Co-disposal of fine tails and overburden utilizing pipeline techniques. Oil Sands – Our Petroleum Future Conference, Edmonton, Alberta, 4-7 April 1993. Pp 20.

Lord, E.R. and Liu, Y. 1998. Depositional and geotechnical characteristics of paste produced from oil sands tailings. Tailings and Mine Waste 1998. Balkema, Rotterdam. pp. 147-157.

MacKinnon, M.D., Matthews, J.G., Shaw, W.H., and Cuddy, R.G. 2001. Water quality issues associated with Composite Tailings (CT) technology for managing oil sand tailings. International Journal of Surface Mining, Reclamation, and Environment, **15**(4): 235-256.

Martel, C.J. 1988. Development and design of sludge freezing beds. U.S. Army Cold Regions Research and Engineering Laboratory, CRREL Report 88-20.

Martens, S., Lappin, T., and Godwaldt, R. 2008. Compaction of upstream construction tailings beaches using dozers. 61st Canadian Geotechnical Conference, Edmonton, AB. 21-24 Septmeber 2008. Pp. 7.

Masala, S. 1998. Numerical Solution of sedimentation and consolidation of fine tailings, M.S.c. thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, AB.

Masala, S. and Matthews, J. 2010. Predicting development of undrained shear strength in soft oil sands tailings. Proceeding of the Second International Oil Sands Tailings Conference 5-8 December 2010, Edmonton, Alberta, Canada. pp 31-41.

Masala, S., Langseth, J., Smyth, K, Dunmola, A., Mckay, D., and Clark, J. 2012. TSRU tailings pilot-scale thickening and deposition trials. *In* Proceedings of the 3rd International Oil Sands Tailings Conference, Edmonton, Alberta, Canada, 3-5 December 2012, pp. 333-342.

Masliyah, J., Zhou, Z., Xu, Z., Czarnecki, J. and Hamza, H. 2004. Understanding water-based bitumen extraction from Athabasca oil sands. The Canadian Journal of Chemical Engineering, **82**: 628-654.

Masliyah, J., Xu, Z., and Czarnecki, J. 2011. Handbook on theory and practice of bitumen recovery from Athabasca oil sands – Volume 1: Theoretical Basis. Kingsley Publishing Services. 468 pp.

Matthews, J. 2008. Past, Present, and Future Tailings – Tailings Experience at Albian Sands Energy. First International Oil Sands Tailings Conference, Edmonton, AB, 7-10 December 2008. Presentation only.

Matthews, J.G., Shaw, W.H., MacKinnon, M.D., and Cuddy, R.G. 2002. Development of composite tailings technology at Syncrude. International Journal of Surface Mining, Reclamation, and Environment, **16**(1): 24-39.

McKenna, G. 2008. Landscape design for oil sands tailings. First International Oil Sands Tailings Conference, December 7-10, 2008, Edmonton, AB, Canada, pp. 51.

McPhail, G. 2008. Prediction of the beach profile of high density thickened tailings from rheological and small scale trial deposition data. Proceedings of the 11th International Seminar on Paste and Thickened Tailings Kasane, Botswana, 5-9 May 2008. Pp. 179-188.

McRoberts, E. 2008 Considerations in oilsands tailings containment. First International Oil Sands Tailings Conference. Edmonton, Alberta. December 2008. 13-23.

MEND. 2001. Mend Manual, Report 5.4.2d, Vol. 4. Prevention and Control. CANMET, Ottawa, Ontario.

Mesri, G. 1975. New design procedure for stability of soft clays. (Discussion). Journal of Geotechnical Engineering, **101**(4): 1090-1093.

Mesri, G. 1989. A reevaluation of $S_{u(mob)} = 0.22\sigma'_p$ using laboratory shear tests. Canadian Geotechnical Journal **26**: 162-164.

Mihiretu, Y., Chalaturnyk, R.J. and Scott J.D. 2008. Tailings segregation fundamentals from flow behaviour perspective. In First international Oil Sands Tailings Conference. Edmonton, AB, 7-10 December 2008, pp. 112-120.

Mikula, R. and Omotoso, O. 2006. Role of clays in controlling tailings behaviour in oil sands processing. Clay Science, **12**(2): 177-182.

Mikula, R.J., Munoz, V.A., and Omotoso, O. 2008. Centrifuge options for production of "dry stackable tailings" in surface mined oil sands tailings management. In Canadian International Petroleum Conference / Petroleum Society's 59th Annual Technical Meeting, Calgary, AB, Canada, pp. 7.

Miller, W.G., Scott, J.D. and Sego, D.C., 2011a. Effect of extraction water chemistry on the consolidation of oil sands fine tailings. CIM Journal **1**(2): 113-129.

Miller, W.G., Scott, J.D. and Sego, D.C., 2011b. Influence of extraction process and coagulant addition on thixotropic strength of oil sands fine tailings. CIM Journal **1**(3): 197-205.

Mitchell, J.K. 1960. Fundamental aspects of thixotropy in soils. Journal of soil Mechanics Foundation Divisions American Society of Civil Engineers, **83**(3): 19-52.

Mitchell, J.K. and Soga, K. 2005. Fundamentals of Soil Behaviour, 3rd Edition, John Wiley and Sons, Hoboken, New Jersey, USA.

Mittal, H.K. and Morgenstern, N.R. 1975. Parameters for the design of tailings dams. Canadian Geotechnical Journal, **12**(2): pp. 235-261.

Mohamedelhassan, E. 2008. Electrokinetic sedimentation and dewatering of clay slurries. First International Oil Sands Tailings Conference, December 7-10, 2008, Edmonton, Alberta, Canada. pp. 153-160.

Morgenstern, N.R. and Scott, J.D. 1995. Geotechnics of fine tailings management; geoenvironment 2000; characterization, containment, remediation, and performance in environmental geotechnics, Geotechnical Special Publication, 46: 1663-1683.

Morgenstern, N.R., Fair, A.E., and McRoberts, E.C. 1988. Geotechnical Engineering beyond soil mechanics – a case study. Canadian Geotechnical Journal, **25**: 637-661.

Mundy, P. and Madsen, B. (2009). The development of centrifugal separation technology for tailings treatment. Tailings and Mine Waste 2009, Banff, Alberta, 1-4 November 2009, pp. 397-403.

Neurohr, G. A. 1983. Use of macrophytes for sludge treatment. In Proceedings of the 6th Symposium on wastewater treatment, Montreal, Quebec, 16-17 November 1983. pp. 413-430.

Newson, T.A. and Fahey, M. 2003. Measurement of evaporation from saline tailings storages. Engineering Geology, 70(2003): 217-233. doi:10.1016/S0013-7952(03)00091-7

Nik, R. M., Sego, D.C. and Morgenstern, 2008. Possibility of Using Centrifugal Filtration for Production of Non-Segregating Tailings. First International Oil Sands Tailings Conference, Edmonton, AB, Canada, 7 – 10 December 2008 pp. 200-208.

Nixon, J.F., Morgernstern, N.R., 1973. The residual stress in thawing soils. Canadian Geotechnical Journal **10**(4), 571–580.

Norton, V. and Wilkie, W. 2004. Clarifying centrifuge operation and selection. Chemical Engineering Progress, **100**:8. Pp 34- 39.

Oil Sands Developers Group (OSDG). 2013. Oil sands project list, updated March 2013. Unpublished report. Retrieved from http://www.oilsandsdevelopers.ca/wp-content/uploads/2013/03/Oil-Sands-Project-List-March-5-2013.pdf March 26, 2013.

Oliveira-Filho, W. and Van Zyl, Dirk (2006) Modeling Discharge of Interstitial Water from Tailings Following Deposition – Part 2: Application. Brazilian Journal on Soil and Rocks, **29**(2):, August.

OSTC. 2012. Technical Guide for Fluid Fine Tailings Management. Oil Sands Tailings Consortium (OSTC), Calgary, Alberta, 30 August 2012. Available from http://www.cosia.ca/uploads/files/Projects/Tailings/Tech_Guide_Fluid_Tailings_Mgmt_30%20Aug%202012_COSIA.pdf [cited 21 March 2013].

Pane, V. and Schiffman, R.L. 1985. A note on sedimentation and consolidation. Gėotechnique, **35**(1):69-72.

Pollock, G. 1988. Large Strain Consolidation of Oil Sand Tailings Sludge, M.S.c. thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, AB.

Pollock, G. 2004. Consolidation Modeling of Oilsand Tailings. CONRAD Tailings Seminar. Alberta Research Council, November 8, Edmonton, AB.

Proskin, S. 1998. A Geotechnical Investigation of Freeze-Thaw Dewatering of Oil Sands Fine Tailings. PhD thesis, University of Alberta.

Proskin, S. 2012. Revisiting 1992-94 field tests of freeze-thaw dewatering of oil sands MFT. *In* Proceedings of the David C. Sego Symposium, Edmonton, Alberta, 26-27 April 2012, pp. 101-112.

Proskin, S., Sego, D. and Alostaz, M. 2010. Freeze-Thaw and Consolidation Tests on Suncor Mature Fine Tailings (MFT). Cold Regions Science and Technology, **63**(3): 110 – 120.

Purcell, B.C., Barlow, J.P., Nordin, D., and Fleming, S. 2000. Tailings disposal at Syncrude Canada's Ltd Aurora Mine. 6th International Conference on Environmental Issues and Management of Waste in Energy and Mineral Production. Calgary, Alberta. 30 May – 2 June. 2000. Pp. 153-163.

Qiu, Y., 2000. Optimum deposition for sub-aerial tailings disposal. PhD thesis, University of Alberta.

Qiu, Y. and Sego, D.C., 1998. Design of sub-aerial tailings deposition in arid regions, in Proceedings of the 51st Canadian Geotechnical Conference, pp. 615 – 622.

Qiu, Y. and Sego, D.C. 2001. Laboratory properties of mine tailings. Canadian Geotechnical Journal **38**: 183-190.

Qiu, Y. and Sego, D. C. 2006. Optimum deposition for sub-aerial tailings disposal: concepts and theories. International Journal of Mining, Reclamation and Environment, **20**:4, 272 — 285.

Reid, D. and Fourie, A. 2012. The effects of "in-line" flocculation on the geotechnical properties of clay tailings. *In* Proceedings of the 16th Tailings and Mine Waste 2012, Keystone, Colorado, Unites States of America, 14-17 October 2012.

Reeves, B. 1996. Environmental Impact Assessment for the Syncrude Canada Limited Aurora Mine. BOVAR Environmental, 1996.

Richardson, J.F., Harker, J.H., and Backhurst, J.R. (2002). Coulson and Richardson's Chemical Engineering Handbook, Volume 2, Particle Technology and Separation Processes. Butterworth Heinemann, 5th Edition.

Ripperger, S. and Altamnn, J. 2002 Crossflow microfiltration – State of the art. Separation and Purification Technology, **26**(2002): 19-31.

Ritcey, G.M. 1989. Tailings Management: Problems and Solutions in the Mining Industry. Elsevier Science Publishers, Amsterdam. pp 970.

Rizzo, D.M., Mouser, P.J., Whitney, D.H., Mark, C.D., Magarey, R.D., and Voinov, A.A. 2006. The comparison of four dynamic systems-based software packages: Translation and sensitivity analysis. Environmental Modeling and Software, **21**: 1491-1502.

Robinksy, E.I. 1978. Tailings disposal by the thickened discharge method for improved economy and environmental control. Tailings Disposal Today. 2nd International Tailings Symposium, Denver Colorado, USA. May 1978. Pp. 75-92.

Rowe, R.K. 2005. Long-term performance of contaminant barrier systems. Geotechnique, **55**(9): 631-678, November 2005.

Rykaart, E.M., 2002. A methodology to describe spatial surface flux boundary conditions for solving tailings impoundment closure water balance problems. PhD thesis, Department of Civil and Environmental Engineering, University of Saskatchewan, Saskatoon, Canada.

Scaffo-Migliaro, A. 2007. Modeling and simulation of an at-face slurry process for oil sands. MSc thesis, Department of Mechanical Engineering, Univeristy of Alberta, Edmonton, Alberta.

Schiffman, R.L., Chen, A.T-F, and Jordan, J.C. 1969. An analysis of consolidation theories. Journal of Soil Mechanics and Foundation division, Proceedings of the American Society of Civil Engineers, **95**(SMI): pp. 285-312.

Schiffman, R.L., Vick, S.G. and Gibson, R.E. 1988. Behavior and properties of hydraulic fills. Hydraulic Fill Structures, Eds: DJ Van Zyl and SG Vick. American Society of Civil Engineers, Colorado State University, pp. 166-202.

Scott, J.D. 2014. Personal communication.

Scott, M.D. and Lo, R.C. 1992. Optimal tailings management at highland valley copper, CIM Bulletin, **85**: 85-88.

Sego, D. 1992. Influence of pore fluid chemistry on freeze-thaw behaviour of Suncor Oil Sand Fine Tails (Phase I). Submitted to Reclamation Research Technical Advisory Committee, Alberta Environment, 35p.

Sego, D.C. 2014. Personal communication.

Shaw, B., Hyndman, A., and Sobkowicz, J. 2010. Consolidation projections for thickened tailings. In Proceedings of the 2nd International Oil Sands Tailings Conference, Edmonton, Alberta, 5-8 December 2010, pp 253-267.

Shaw, M. 2008. Past, Present, and Future Tailings Management – Suncor Energy. First International Oil Sands Tailings Conference, Edmonton, AB, 7-10 December 2008. Presentation only.

Sheeran, D. 1993. An improved understanding of fine tailings structure and behavior. Fine Tailings Symposium, Environment Canada, Edmonton, Alberta, 4-7 April 1993, 11 pp.

Shell Canada Energy (Shell). 2005. Application for approval of the Muskeg River Mine Expansion project. Volume 1: Project Description. 28 April 2005.

Shell Canada Energy (Shell). 2010a. Muskeg River Mine – Annual tailings management plan 2011. Directive 074 – Appendix E requirement. Submitted to ERCB. 29 September 2010.

Shell Canada Energy (Shell). 2010b. Jackpine Mine ERCB Directive 074 - Tailings management plan. Submitted to ERCB. 3 December 2010.

Shell Canada Energy (Shell). 2010c. Muskeg River Mine – 2010 Annual fluid pond status report and compliance report (D74-Appendix D and F). 29 September 2010.

Shook, C.A., Gilles, R.G., and Sanders, R.S. 2002. Pipeline Hydrotransport with Applications in the Oil Sand Industry. SRC Publication No. 11508-1E02. Saskatoon, Saskatchewan, Canada. 118 pp.

Silva, M. J.. 1999. Plant dewatering and strengthening of mine waste tailings. PhD thesis. University of Alberta. Department of Civil and Environmental Engineering. 271 p.

Simms. P. and Grabinsky, M. 2004. A simple method for estimating rates of drying and desaturation of paste tailings during surface deposition. Tailings and Mine Waste 2004, 11-13 October 2004, pp 287-292.

Simms. P., Grabinsky, M., and Zhan, G. 2007. Modelling evaporation of paste tailings from the Bulyanhulu mine. Canadian Geotechnical Journal **44**: 1417-1432.

Simms, P., Williams, M.P.A., Fitton, T.G., and McPhail, G. 2011. Beaching angles and evolution of stack geometry for thickened tailings – a review. 14th International Seminar on Paste and Thickened Tailings, Perth Australia, 5-7 April 2011. Pp. 323-338.

Sisson, R., Lacoste-Bouchet, P., Vera, M., Costello, M., Hedblom, E., Sheets, B., Nesler, D., Solseng, P., Fandrey, A. Van kesteren, W., Tamon, A., and Sittoni, L. 2012. An analytical model for tailings deposition developed from pilot scale testing. 3rd International Oil Sands Tailings Conference, Edmonton, AB, 2-5 December 2012, pp 53-63.

Sobkowicz, J.C. 2010. History and development in the treatment of oil sands fine tailings. *In* Proceedings of the 14th International Conference on Tailings and Mine Waste, Vail, Colorado, USA, 17-20 October 2010. Taylor Francis Group, pp. 11-30.

Sobkowicz, J. 2012. The oil sands tailings technology roadmap and action plan: Introduction and key recommendations. 3rd International Oil Sands Tailings Conference, Edmonton, AB, 2-5 December 2012, pp. 13-22.

Sobkowicz, J.C. and Morgenstern, N.R. 2009. A geotechnical perspective on oil sands tailings. 13th International Conference on Tailings and Mine Waste, Banff, Alberta, 1-4 November 2009. University of Alberta, pp. xvii-xli.

Somogyi, F. 1980. Large Strain Consolidation of Fine Grained Slurries. Presented at the Canadian Society for Civil Engineering, Winnipeg, Manitoba, May 29-30.

Song, Q., O'Kane, M., Dhadli, N., and Matthews, J. 2011. Deposition thickness and evaporative drying for oil sands tailings in northern Alberta. Mine Closure 2011. Proceedings of the Sixth International Conference on Mine Closure. Lake Louise, Alberta, Canada, 18-21 September 2011. pp. 373-382.

Sorta, A.R. and Sego, D.C. 2010. Segregation related to use of Centrifuge in Oil Sands Tailings. *In* Proceedings of the 63rd Canadian Geotechnical Conference, Calgary, Alberta, 12-16 September 2010. Canadian Geotechnical Society, pp. 656-664.

Sparks, T. 2011. Solid-Liquid Filtration. Elsevier, pp 208. http://dx.doi.org.login.ezproxy.library.ualberta.ca/10.1016/B978-0-08-097114-8.01001-7.

Stahl, R.P. 1996. Characterization and natural processes enhancing dry landscape reclamation of fine processed mine wastes. Ph.D. thesis, University of Alberta, Edmonton, Alberta.

Stahl, R.P., Sego, D.C., 1995. Freeze–thaw dewatering and structural enhancement of fine coal tails. American Society of Civil Engineering Journal of Cold Regions Engineering **9**(3): 135–151.

Suthaker, N. N. 1995. Geotechnics of Oil Sand Fine Tailings, Ph.D. thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, AB.

Suthaker, N.N. and Scott, J.D. 1996. Measurement of Hydraulic Conductivity in Oil Sand Tailings Slurries. Canadian Geotechnical Journal, **33**(4): 642-653.

Suthaker, N.N. and Scott, J.D. 1997. Thixotropic strength measurement of oil sand fine tailings. Canadian Geotechnical Journal, **34**(6): 974-984.

Syncrude Canada Ltd. 1995. Non –segregating tailings: 1995 NST field demonstration summary report. Syncrude Canada ltd. research and development. 201 pp.

Syncrude Canada Ltd. (Syncrude). 2010a. 2010 Annual tailings plan submission Syncrude Aurora North (Leases 10, 12 and 34). 30 September 2010.

Syncrude Canada Ltd. (Syncrude). 2010b. 2010 Annual tailings plan submission Syncrude Aurora South (Lease 31). 30 September 2010.

Syncrude Canada Ltd. (Syncrude). 2010c. Directive 074 Baseline survey for fluid deposits Syncrude Mildred Lake and Aurora North. 30 September 2010.

Syncrude Canada Ltd. 2010d. 2009 Groundwater Monitoring Report. Submitted to Alberta Environment, March 30, 2010.

Syncrude Canada Ltd. (Syncrude). 2012. 2012 Annual tailings plan submission Syncrude Aurora North (Leases 10, 12 and 34). 30 September 2012.

Theriault, J.A. Frostiak, J., and Welch, D. 2003. Surface disposal of paste tailings at the Bulyanhulu gold mine, Tanzania. Sudbury 2003, Mining and the Environment III. Sudbury, Ontario, 25-28 May 2003, pp. 1-8.

Toorman, E.A. 1999. Sedimentation and self-weight consolidation: constitutive equations and numerical modeling. Géotechnique, **49**(6):709–726.

Toorman E. and Gudehus G. 1998. Sedimentation and self-weight consolidation: general unifying theory, Discussion. Geotechnique, **48**: 295-298.

U.S. Environmental Protection Agency (EPA). 1994. Technical Report. Design and Evaluation of Tailings Dams. Unpublished report # EPA530-R-94-038, US EPA Office of Solid Waste, Washington DC. August 1994. Last accessed May 13, 2013 at http://www.epa.gov/osw/nonhaz/industrial/special/mining/techdocs/tailings.pdf

Usher, S.P. and Scales, P.J. 2005. Steady state thickener modeling from the compressive yield stress and hindered settling function. Chemical Engineering Journal, **111**(2005): 253-261.

Vermeer, P.A., Dykstra, C., Jacobs, W., Beuth, L. 2015. Creep-Consolidation Analyses of Alberta Mature Fine Tailings. Prepared for publication.

Vick, S.G. 1990. Planning Design and Analysis of Tailings Dams. BiTech Publishers Ltd. Pp. 369.

Volker, A. 1982. Polders: An ancient approach to land reclamation. Nature and Resources **18**: 2-13.

Wallace, D., Tipman, R., Komishke, B., Wallwork, V., and Perkins, E. 2004. Fines/water interactions and consequences of the presence of degraded illite on oil sands extractability. The Canadian Journal of Chemical Engineering, **82**: 667-677.

Wang. T., Feng, X., Xu., Z. and Masliyah, J. 2010. Polymer aids for settling and filtration of oil sand tailings. The Canadian Journal of Chemical Engineering, **88**:6. 403-410.

Wels, C. and Robertson, M. 2003. Conceptual model for estimating water recovery in tailings impoundments. Tailings and Mine Waste '03. Pp. 87-94.

Wells, P.S. and Riley, D.A. 2007. MFT drying – case study in the use of rheology modification and dewatering of fine tailings through thin lift deposition in the oil sands of Alberta. Paste 2007, Tenth International Seminar on Paste and Thickened Tailings, Perth, Australia, 13-15 March 2007. Pp. 271-284.

Wells, P.S., Revington, A., Omotoso, O. 2011. Mature fine tailings drying – technology update. Paste 2011, R.J. Jewel and A.B. Fourie (eds), Australia Centre for Geomechanics, Perth, ISBN 978-0-9806154-3-2.

Wickham, M., Johnson, B., and Johnson, D. 2004. Dynamic systems-modeling approach to evaluating hydrologic implications of backfill designs. Mining Engineering, **56**(1): 52-56.

Wills, B.A. 2006. Wills' mineral processing technology. Elsevier, Great Britain.

Wilson, G.W., Wickland, B., and Fines, P. 2002. Concepts for co-mixing waste rock and tailings. Symposium on Mining and the Environment: Challenges and Prospects. Rouyn-Noranda, Quebec. 3-5 November 2002.

Wilson, G.W., Fredlund, D.G, and Barbour, S.L. 1994. Coupled soil-atmosphere modelling for soil evaporation. Canadian Geotechnical Journal **31**: 151-161.

Wilson, G.W. 2010. Keynote Address: Environmental Dewatering of Fluid Fine Tailings. Second International Oil Sands Tailings Conference, Edmonton, AB, December 5-8, 2010. Presentation only.

Wilson, G.W., Kabwe, L.K., Donahue, R., and Lahaie, R. 2011. Field performance of in-line flocculated fluid fine tailings using thin lift deposition. Mine Closure 2011. Lake Louise, AB, 18-21 September 2011. pp. 473-481.

Wu, S., Sego, D., and Naeth, M. 2010. Evapotranspiration dewatering effect on CT deposits by grasses. Proceedings of the 63rd Canadian Geotechnical Conference, Calgary, AB. 12-16 September 2010. 8 pp.

Xu, Y., Dabros, T., and Kan, J. 2008. Filterability of oil sands tailings, Process Safety and Environmental Protection, **86**(4): 268-276.

Yan, D., Parker, T., and Ryan, S. 2003. Dewatering of fine slurries by the Kalgoorlie Filter Pipe. Minerals Engineering, **16**(2003): 283-289.

Yuan, S. and Lahaie, R. 2009. Thickened tailings (Paste) technology and it's applicability in oil sands tailings management. Tailings and Mine Waste 2009, Banff, Alberta, 1-4 November 2009, pp. 829-844.

Zhang, C. 2010. Cross Flow Filtration of Oil Sand Total Tailings. MSc Thesis submitted to the Department of Civil and Environmental Engineering, University of Alberta. 182 pp.

Zhang, C., Alostaz, M., Beier, N., and Sego, D. 2009. Cross Flow Filtration of Oil Sand Tailings. Tailings and Mine Waste 2009 Conference, Banff, Alberta, 1-4 November 2009. Pp. 799-812.

Zhang, Y. 2012. Laboratory study of freeze-thaw dewatering of Albian mature fine tailings. M.Sc. Thesis Department of Civil and Environmental Engineering, University of Alberta, Edmonton, AB.

Zhang, Y. and Sego, D. 2012. Freeze-thaw enhancement of Albian Mature fine Tailings (MFT). *In* Proceedings of the 3rd International Oil Sands Tailings Conference, Edmonton, Alberta, Canada, 3-5 December 2012, pp. 267-274.

APPENDIX 1 – TMSIM SUB MODEL DETAILS

HYDROCYCLONE SUB-MODEL

Using data from the Syncrude Aurora mine tailings plan (Report tables 3.1, 3.3, 5.1 and 5.2 in Syncrude 2012), the following empirical relationships were developed for hydrocyclone separation. The mass of fines (Q_{tail_fine}) reporting to the cyclone in the feed tailings stream (Q_{tail}) based on the feed fines content (F_{tail}) is:

[A1.1] $$Q_{tail\_fine} = Q_{tail} * F_{tail}$$

Therefore, the mass of sand reporting to the cyclone is:

[A1.2] $$Q_{tail\_sand} = Q_{tail} * (1 - F_{tail})$$

Based on the reported fines content of the ore and the mass of fines (Q_{CUF_fine}) and sand (Q_{CUF_sand}) reporting to the cyclone underflow (Syncrude 2012), the following empirical relationships were developed (Figure A1- 1and Figure A1- 2)

[A1.3] $$Q_{CUF\_fine} = Q_{tail\_fine} * (0.094 - 0.334 * F_{ore})$$

[A1.4] $$Q_{CUF_{sand}} = Q_{tail\_sand} * (0.819 + 0.693 * F_{ore}),\ for\ F_{ore} < 0.158$$

$$= Q_{tail\_sand} * (0.929),\ for\ F_{ore} \geq 0.158$$

Therefore, the total mass of solids reporting to the cyclone underflow is simply:

[A1.5] $$Q_{CUF} = Q_{CUF\_fine} + Q_{CUF\_sand}$$

Knowing the mass of tailings feed (including sand and fines) the mass of solids (Q_{COF}, Q_{COF_sand}, and Q_{COF_fine}) in the cyclone overflow is simply the remainder.

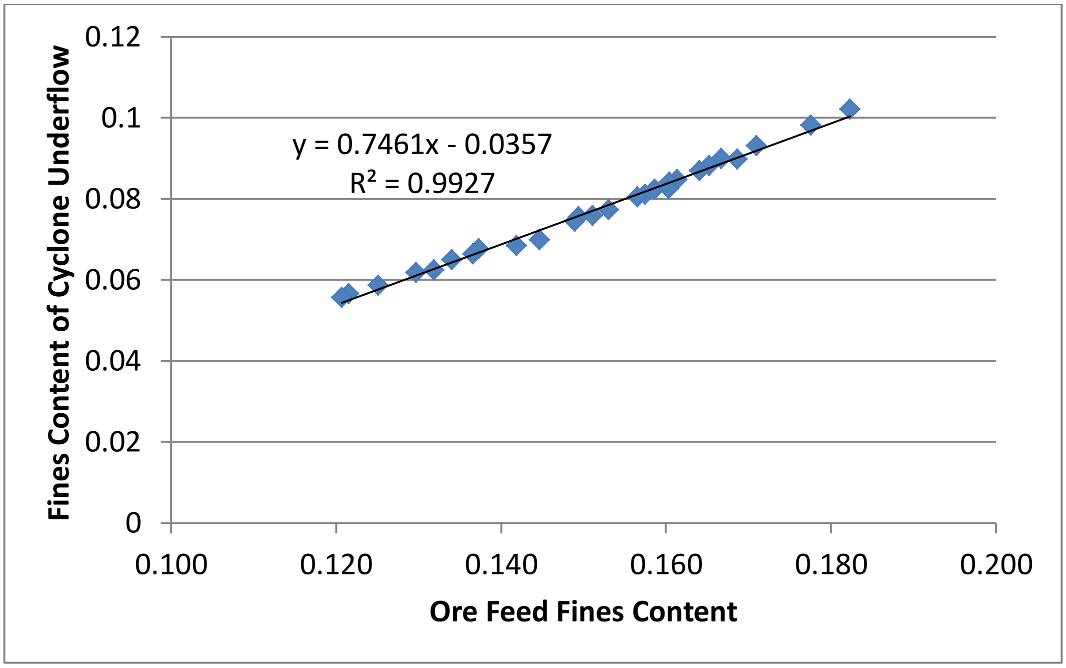


Figure A1- 1. Fines content of the cyclone underflow.

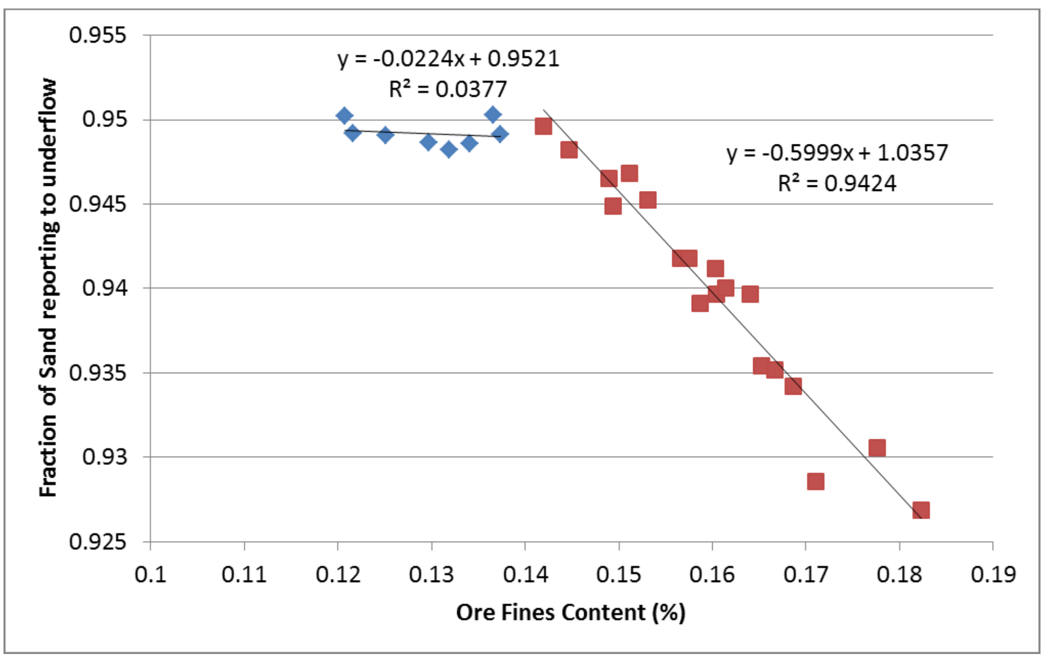


Figure A1- 2. Fraction of sand reporting to the cyclone underflow.

The total volume of the cyclone overflow (Vol_{COF}) be then be calculated based on the solids content of cyclone overflow using equation A1.6.

[A1.6] $$Vol_{COF} = M_{COF} * \left[\frac{\left(\frac{1}{Cw}-1\right)}{Sg\ water} + \frac{1}{Sg\ cof}\right]$$

Using the specific gravity and mass of the sand and fines in the overflow the volume for each component can be calculated including the volume of water reporting to the overflow. The volume of water reporting to the underflow is simply the remainder from the feed water.

BEACHING SUB-MODEL

Syncrude Aurora mine tailings plan (Report tables 3.1, 3.3, 5.1 and 5.2 in Syncrude 2012), the following empirical relationship was developed for to determine the beach sand capture rate (Figure A1- 3). It is expected that a function or static value for fines capture has already been determined. Based on the F_{ore}, the total sand captured in both beach and cell deposition (SC) can be determined from:

[A1.7] $$SC = 1.046 - 1.245 * F_{ore},\ for\ Vol_{tail} < 18.5\frac{Mm^3}{y}$$

$$= 1.035 - 0.830 * F_{ore},\ for\ Vol_{tail} > 18.5\frac{Mm^3}{y}$$

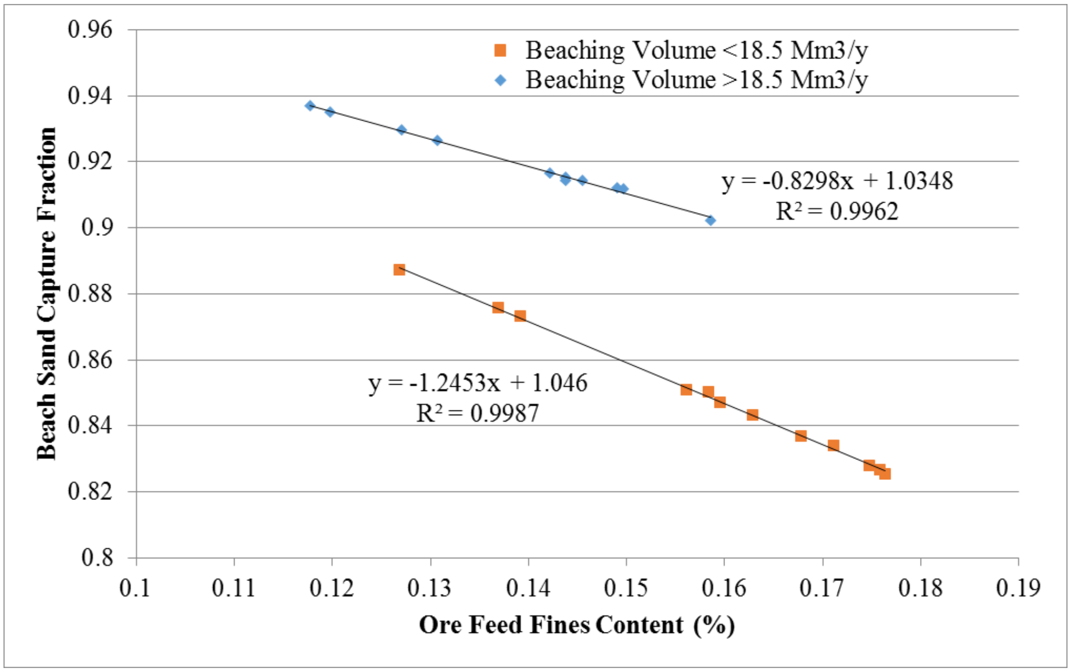


Figure A1- 3. Beaching sand capture for both cell and straight beaching.

FREEZE THAW SUB-MODEL

The following outlines the freeze-thaw dewatering supplemental calculations implemented by the TMSim model for oil sand tailings. The formulation is based on the work of Dawson, Sego and Pollock (1999), Martel (1998), and Proskin (1998). The first step is to specify the expected thaw strain (ε_{th}) expressed as a fraction, and thawed material constitutive coefficients (A, B, C, and D) for effective stress and saturated hydraulic conductivity calculations. If these values are not determined experimentally, they may be estimated from the literature. Freeze-thaw experimental studies published by Dawson et al. (1999), Proskin (1998), and Zhang (2012) have contributed to a database of material properties covering a range of oil sand fine tailings materials from various production sites and extraction processes (Table A1- 1. Characterization Data for freeze-thaw tailings experiments).

Table A1- 1. Characterization Data for freeze-thaw tailings experiments

Site	Solids Content (%)	Clay Fraction (%)	Bitumen Content	w_P	w_L	I_p	Source
Syncrude	28-30	60	1	25	55	30	Dawson et al. 1999
Suncor-1	30.3	49	1				Dawson et al. 1999
OSLO	30	45-50	2-4	20-25	50-55	30	Dawson et al. 1999
Suncor-2	29	NA	NA	20	45	25	Proskin 1998
Albian	36.6	58	1.29	27	54	27	Zhang 2012

NA = not available

The expected ε_{th} for the various tailings materials is presented in Figure A1- 4. Based on the tailings type, deposited void ratio (e_0) and expected frozen bulk density, a reasonable estimate for the expected ε_{th} can be obtained. Therefore, the thawed void ratio (e_{th}) can be calculated with equation A1.8.

[A1.8] $$e_{th} = e_0 * (1 - \varepsilon_{th})$$

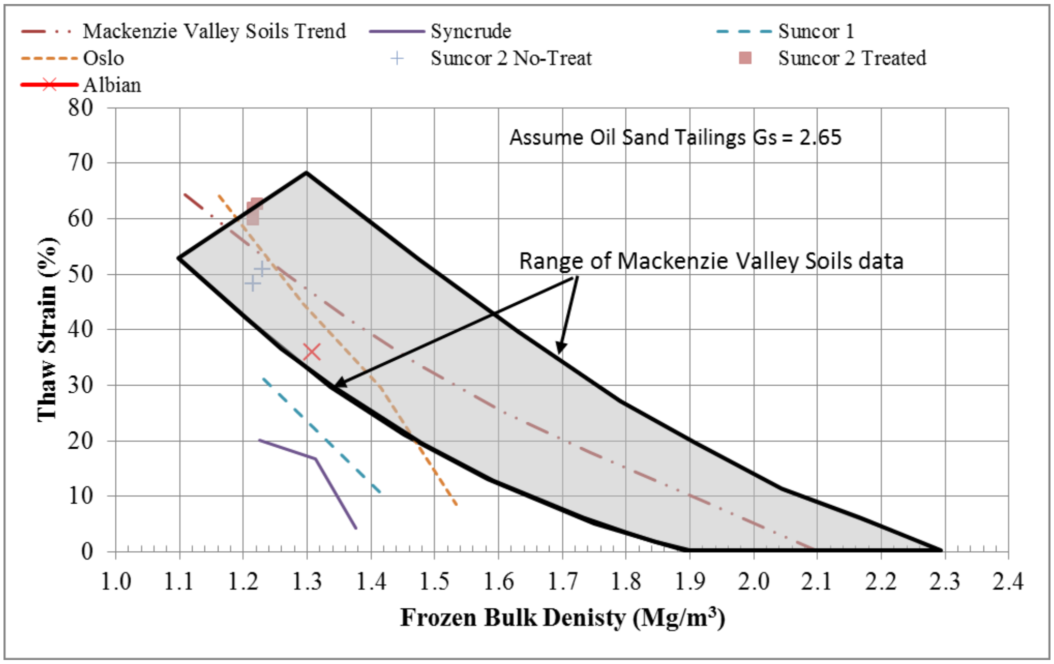


Figure A1- 4. Thaw Strain of various fine grained oil sand tailings.

The compressibility and saturated hydraulic conductivity relationships for the various tailings streams are plotted on the following two Figure A1- 5 and **Error! Reference source not found.**) using a normalized void ratio (e_{norm}) calculated as follows:

[A1.9] $$e_{norm} = \frac{e}{e_{0.5}}$$

Where e is the void ratio at a particular stress after thaw and $e_{0.5}$ is the void ratio immediately after thaw under a loading of 0.5 kPa stress. The presented data demonstrate that the normalized compressibility behavior of the tailings streams is similar at stress levels below 10 kPa, but saturated hydraulic conductivity behavior significantly varies. This variation can be attributed to source of the fine tailings, is extraction and material handling history and addition of chemicals for various treatments (i.e. flocculants).

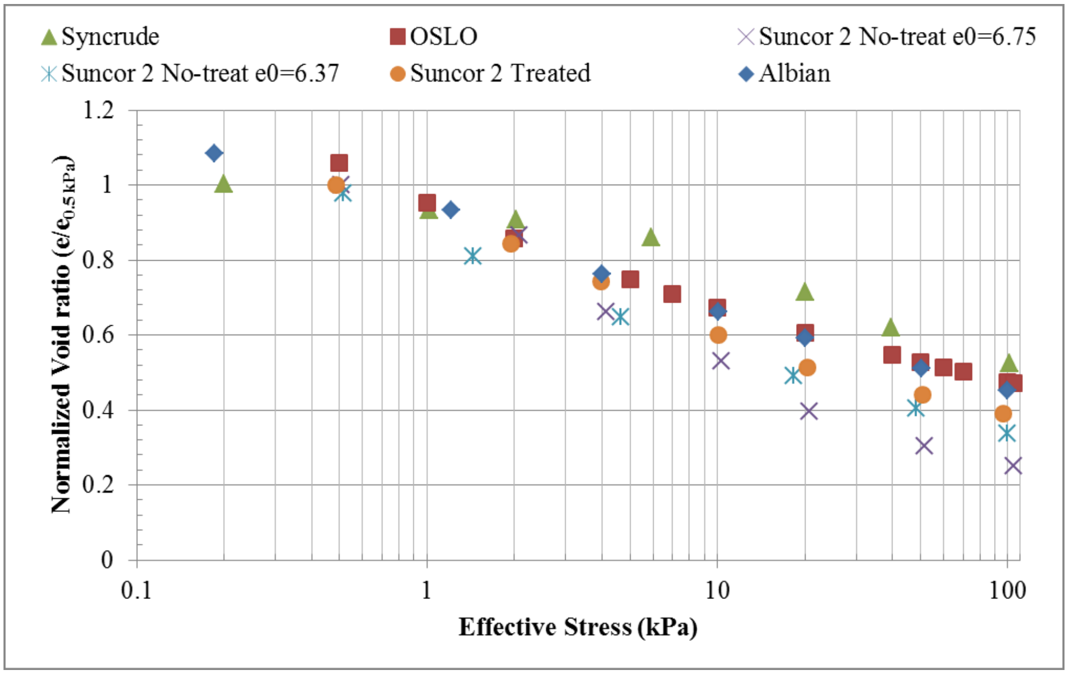


Figure A1- 5. Normalized post thaw compressibility data.

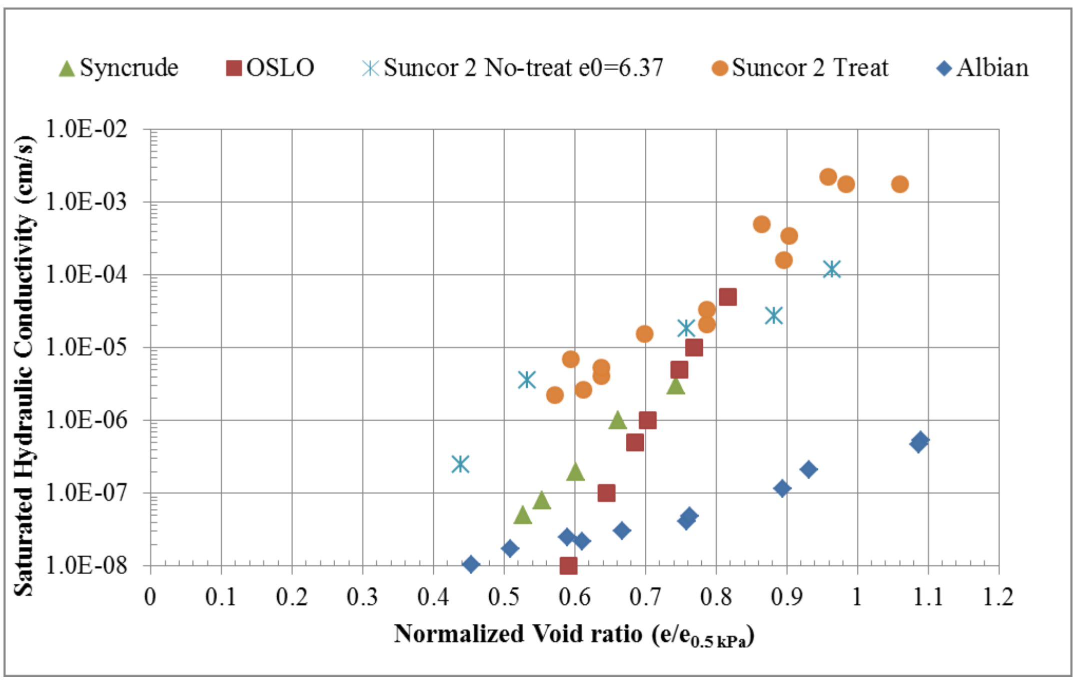


Figure A1- 6. Normalized post thaw saturated hydraulic conductivity data.

The above figures can be used to determine compressibility and saturated hydraulic conductivity functions of the form:

[A1.10] $$e_{norm} = A_{norm}\sigma'^{B}$$

[A1.11] $$k = C_{norm}{e_{norm}}^{D}$$

Where A_{norm} and C_{norm}, are normalized coefficients and must be transformed to A and C in order to be used by the consolidation software. Therefore, the $e_{0.5}$ value must be determined. Through back analysis of the published data, a relationship (Equation A1.12) between $e_{0.5}$ and e_{th} was developed for the several of the tailings streams (Table A1- 2):

[A1.12] $$e_{0.5} = m_{th} * e_{th}$$

Table A1- 2. Thawed void ratio coefficient.

Material	m_{th}
OSLO	0.370
Suncor 2 no treat	0.303
Suncor 2 treat	0.289
Albian	0.411

Now the coefficients A and C can be calculated as follows:

[A1.13] $$A = e_{0.5} * A_{norm}$$

[A1.14] $$C = C_{norm} * \left(\frac{1}{e_{0.5}}\right)^{D}$$

The second step is to determine the freezing and thawing depths based on the site climatic conditions. The model utilized is based on Dawson et al. (1999) and Martel (1988). Dawson et al (1999) demonstrated that is possible to freeze significantly more material at an oil sands mine in one winter than can be thawed the following spring and summer. Therefore, at oil sands mines, thawing will control the design and operation of a freeze-thaw process. The thawing model presented by Dawson et al. (1999) is based on Martel's (1988) work and modified to allow two way heat flow (upward convection and downward conduction). The model assumes water is decanted as thaw progresses and the frozen layer is at or near 0 °C. The upper thaw depth (d_{thaw_u}) can be determined as follows:

$$[A1.15] \qquad d_{thaw\_u} = \sqrt{\left(\frac{K_{TT}}{h_c(1-\varepsilon_{th})}\right)^2 + \frac{2K_{TT}P_T(T_A+\frac{\alpha I}{h_c})}{L(1-\varepsilon_{th})}} + \frac{K_{TT}}{h_c(1-\varepsilon_{th})}$$

Where K_{TT} is the thermal conductivity of the thawed tailings, L is the latent heat of fusion, h_c is the heat transfer coefficient, P_T is the thawing time, T_A is the ambient air temperature, T_G is the ground temperature, α is the solar absorptance of the fine tailings, and I is the solar insolation during thaw. For 30% solids tailings streams, Dawson et al (1999) calculated K_{TT} as 2.2 W/m °C and L as 2761 W day/m^3. For the Fort McMurray, Alberta, area they also estimated I as 208 W/m^2 and h_c of 20 W/m^2. The lower thawed depth (d_{thaw_L}) can be determined as follows:

$$[A1.16] \qquad d_{thaw\_L} = \sqrt{\frac{2P_T T_G K_{TT}}{L(1-\varepsilon_{th})}}$$

REFERENCES

Dawson, R.F., Sego, D.C., Pollock, G.W., 1999. Freeze–thaw dewatering of oil sands fine tails. Canadian Geotechnical Journal **36**(4): 587–598.

Martel, C.J. 1988. Development and design of sludge freezing beds. U.S. Army Cold Regions Research and Engineering Laboratory, CRREL Report 88-20.

Proskin, S.A. 1998. A geotechnical investigation of freeze–thaw dewatering of oil sands fine tailings. Ph.D. Thesis, Department of Civil and Environmental Engineering, University of Alberta, Edmonton, AB.

Syncrude Canada Ltd. (Syncrude). 2012. 2012 Annual tailings plan submission Syncrude Aurora North (Leases 10, 12 and 34). 30 September 2012.

Zhang, Y. 2012. Laboratory study of freeze-thaw dewatering of Albian mature fine tailings. M.Sc. Thesis Department of Civil and Environmental Engineering, University of Alberta, Edmonton, AB.

APPENDIX 2 - TMSIM MODEL STRUCTURE

GOLDSIM CODE

The following figures provide the model structure and influence diagrams for the GoldSim model.

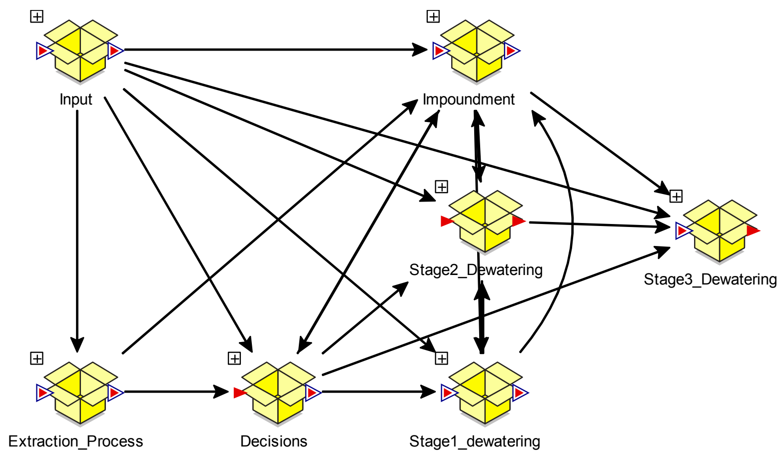


Figure A2- 1. TMSim top level model structure

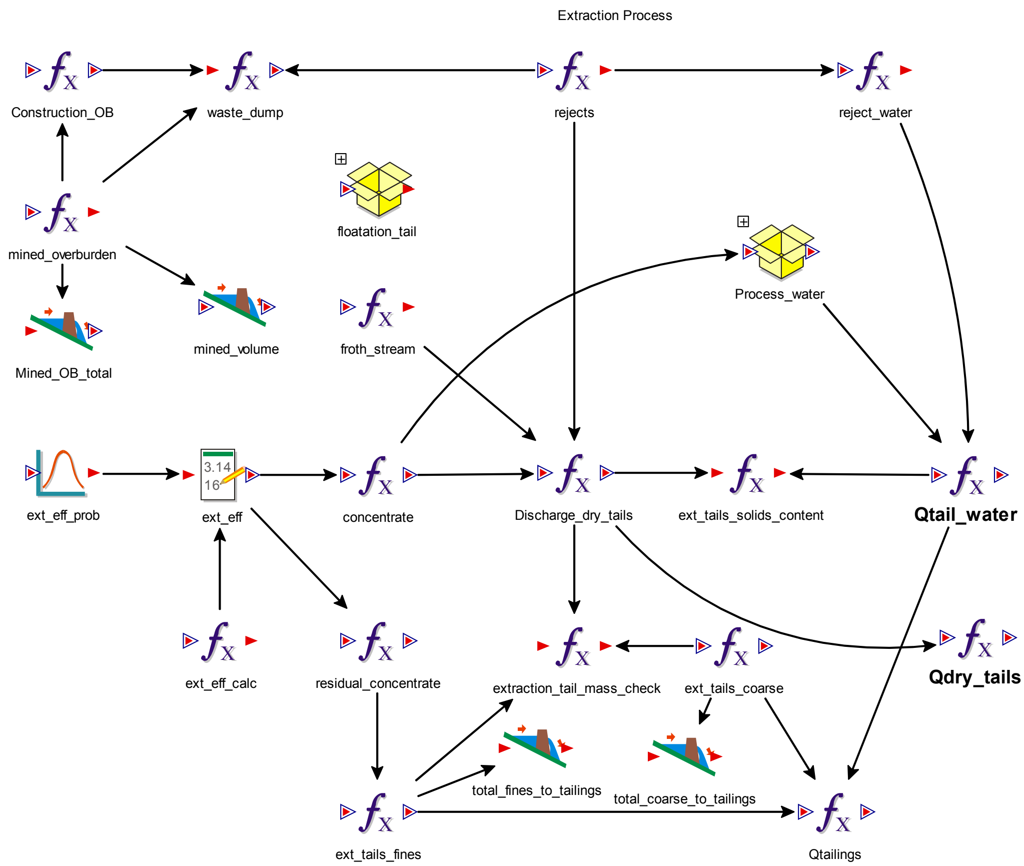


Figure A2- 2. Extraction_Process sub model

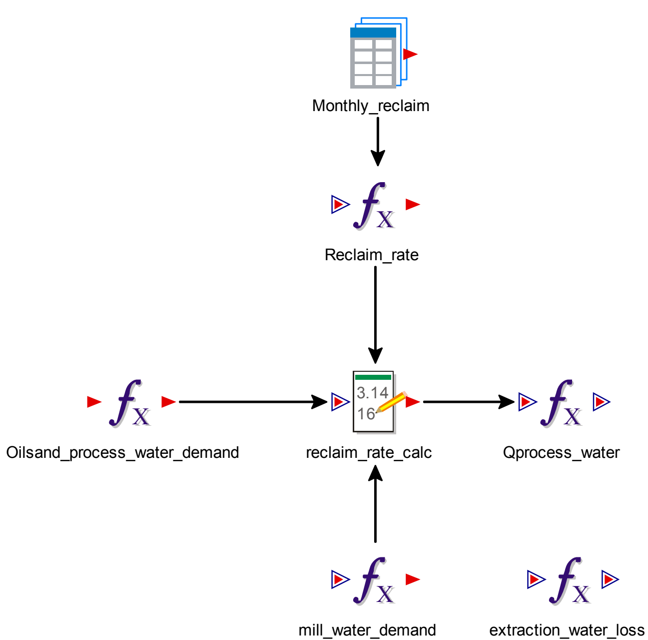


Figure A2- 3. Process_water sub model

DECISIONS CONTAINER SUB MODELS

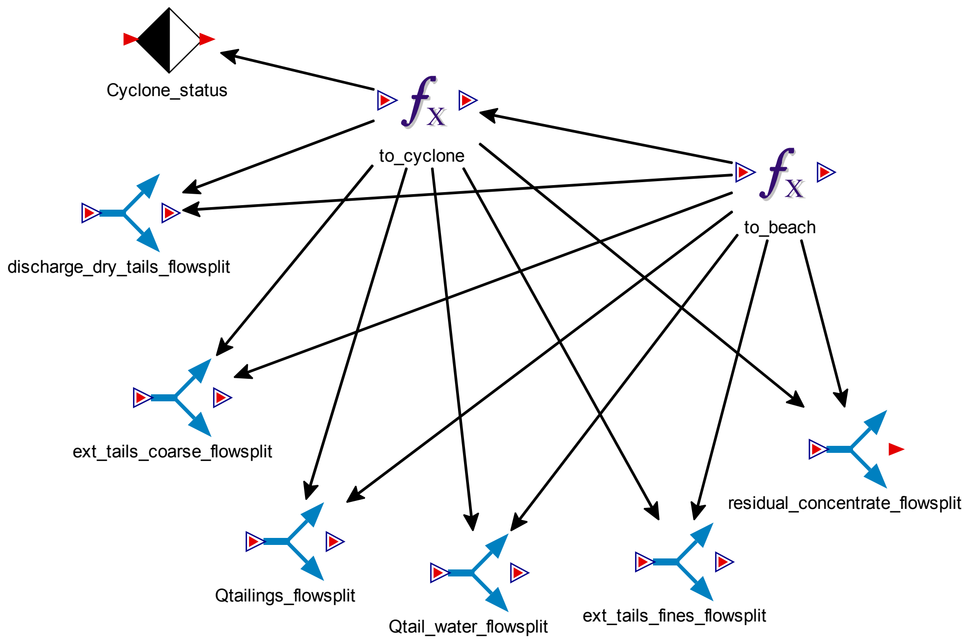


Figure A2- 4. Flow_split model

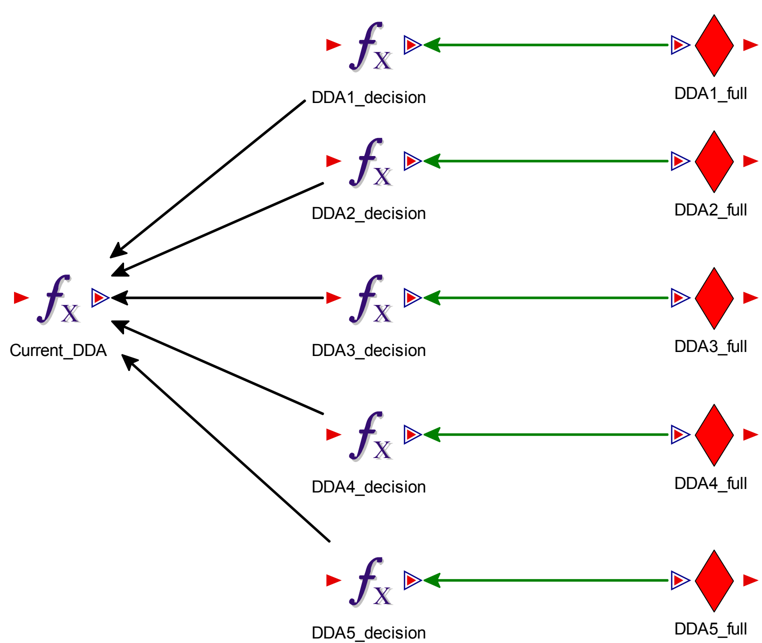


Figure A2- 5. Deposition_Location

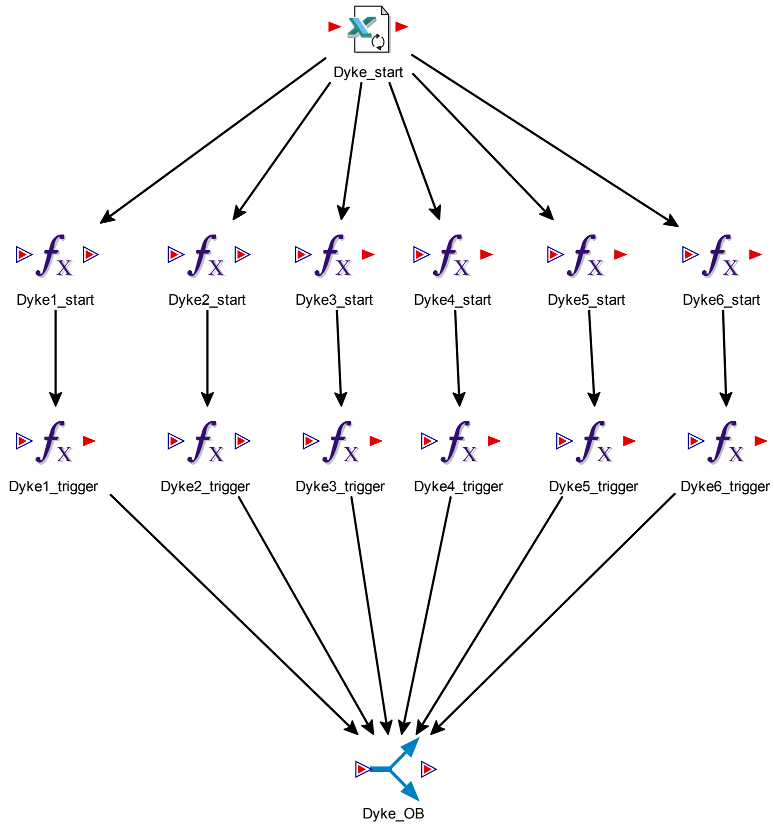


Figure A2- 6. Overburden_Location model

IMPOUNDMENTS CONTAINER SUB MODELS

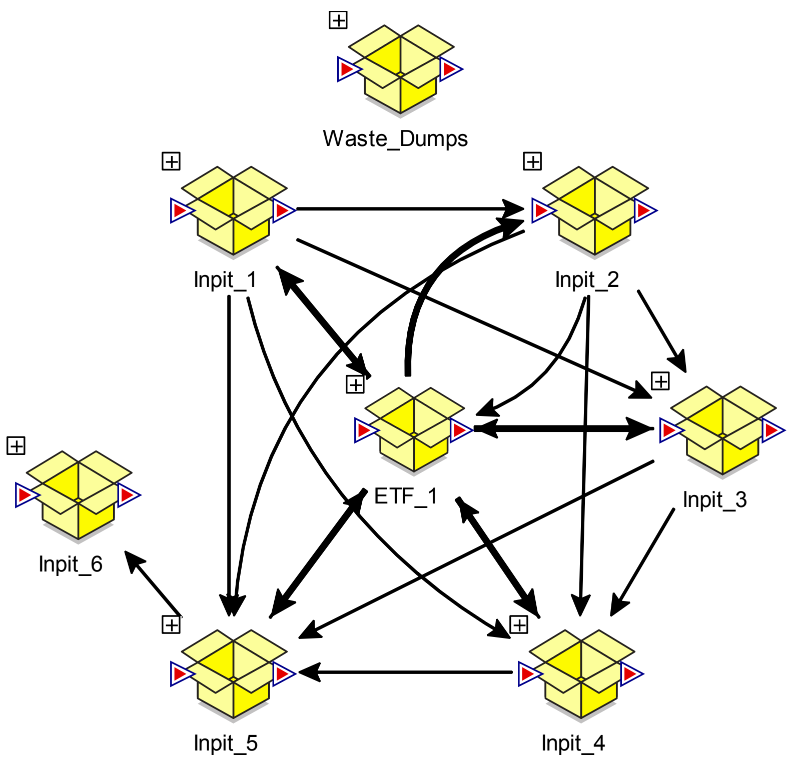


Figure A2- 7. Impoundment container model

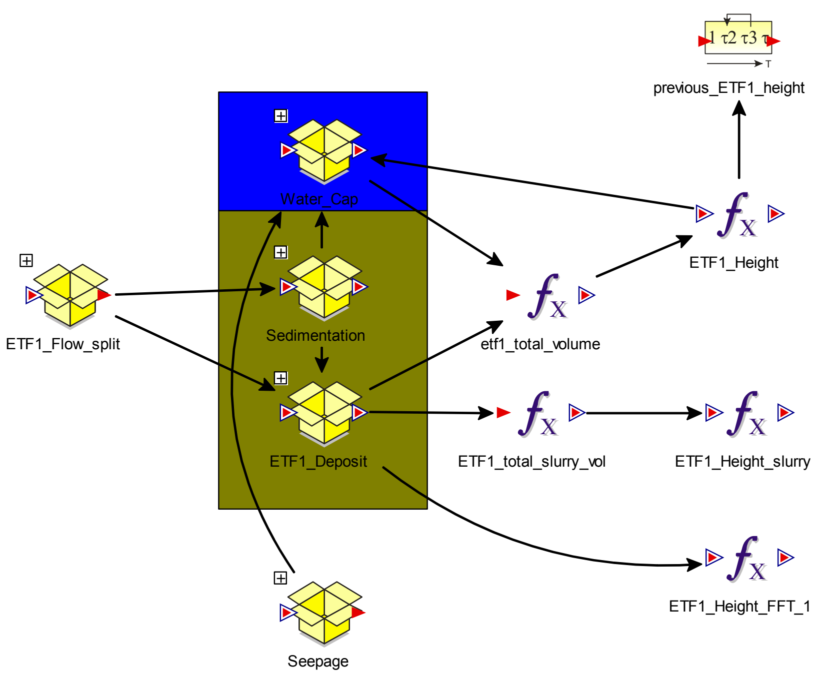


Figure A2- 8. ETF_1 container model

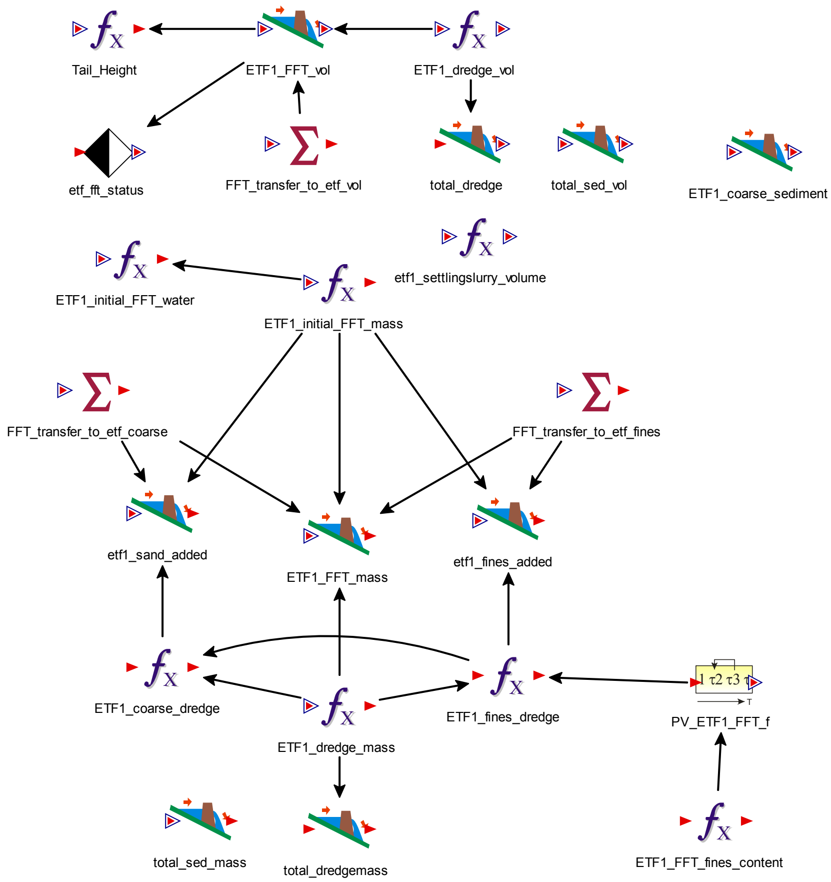


Figure A2- 9. ETF1_Deposit model

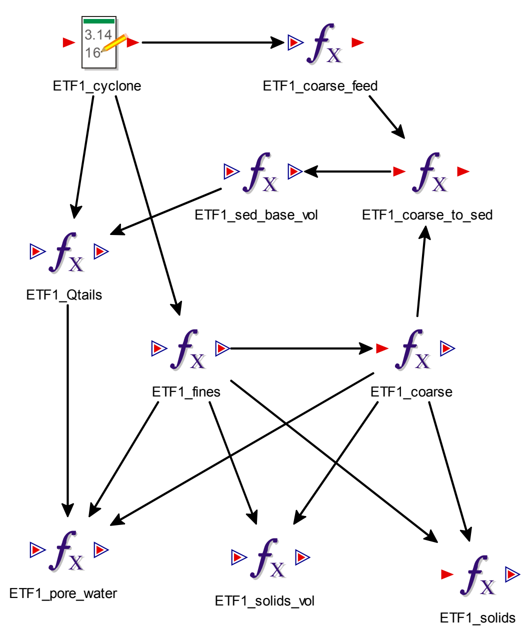


Figure A2- 10. ETF1_flow_split model

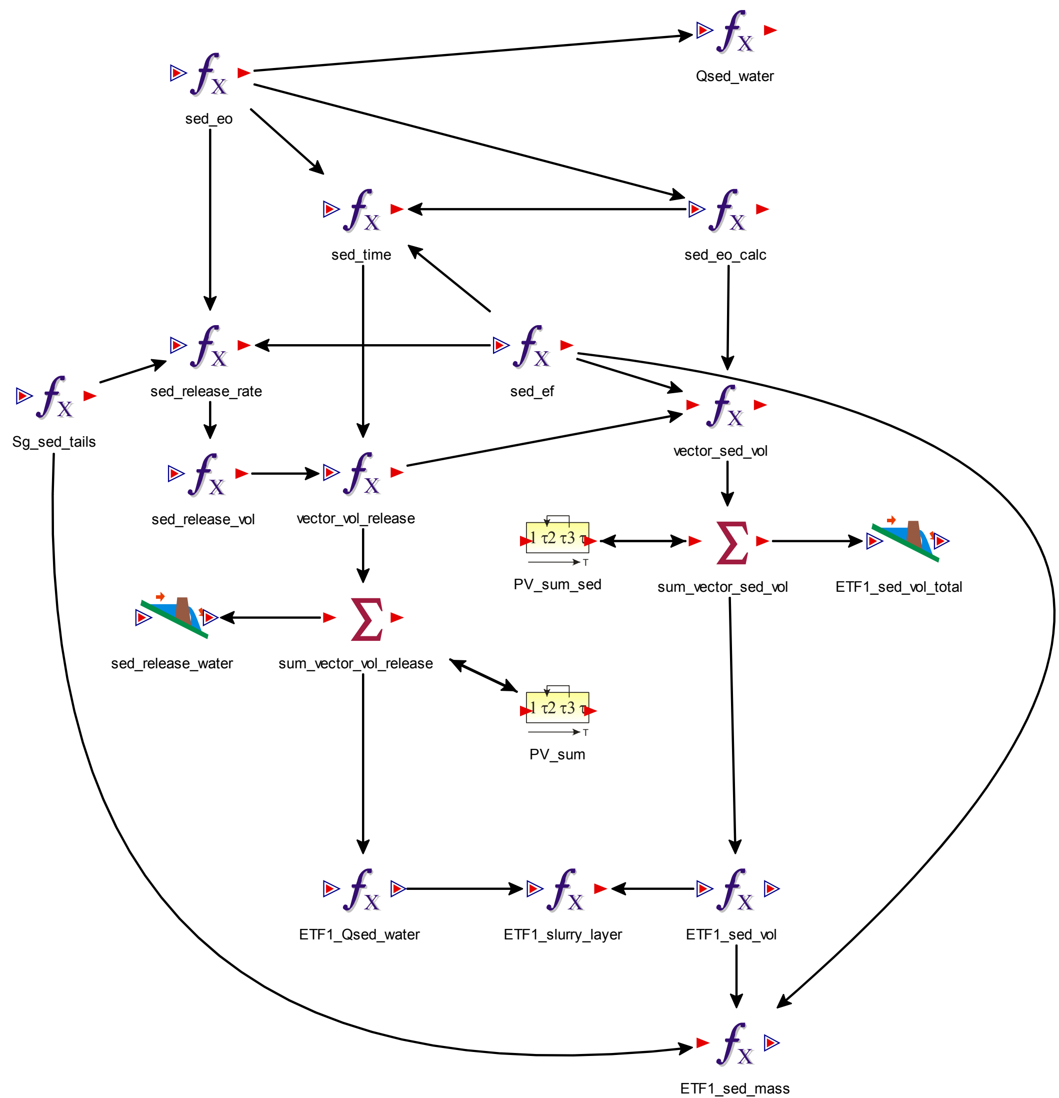
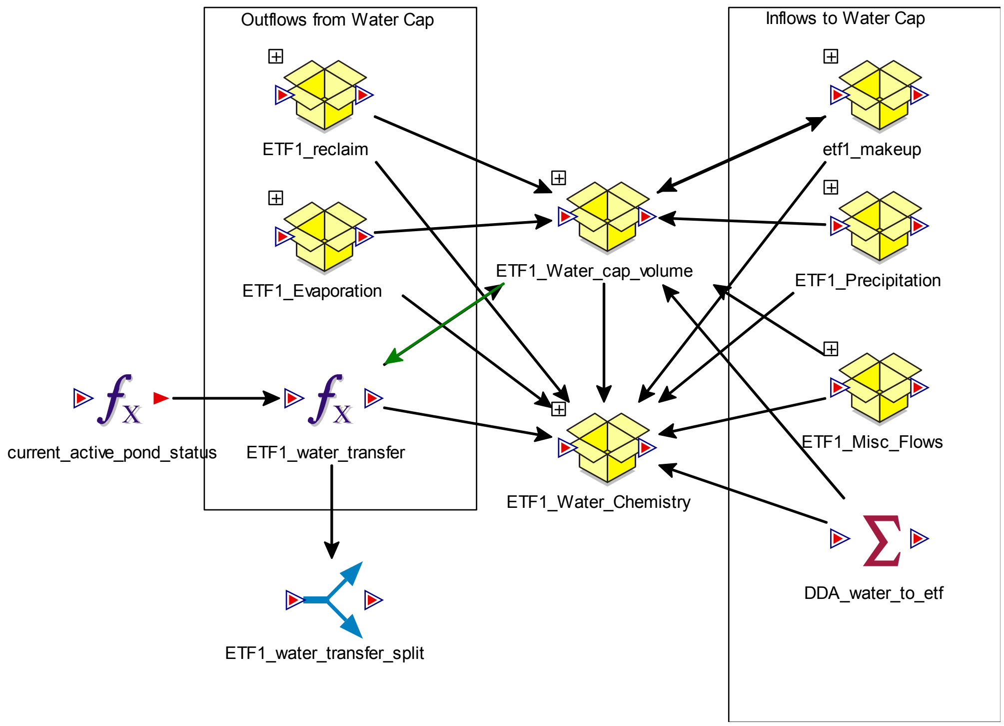


Figure A2- 11. Sedimentation model



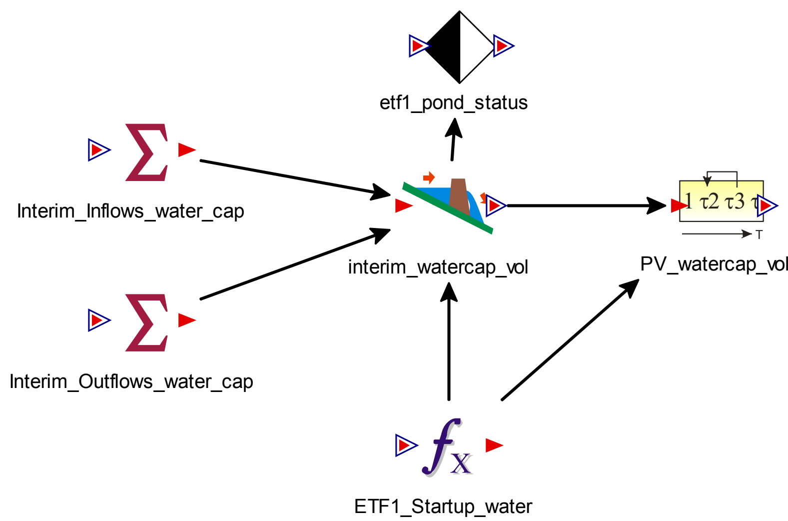


Figure A2- 12. Water_cap (top) and ETF1_water_cap_volume (bottom) models

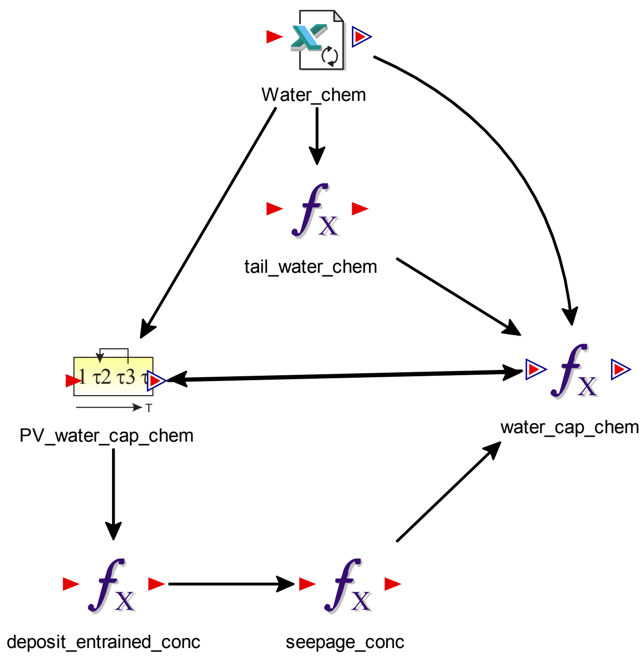


Figure A2- 13. ETF1_water_chemistry model

INPIT CONTAINER MODELS

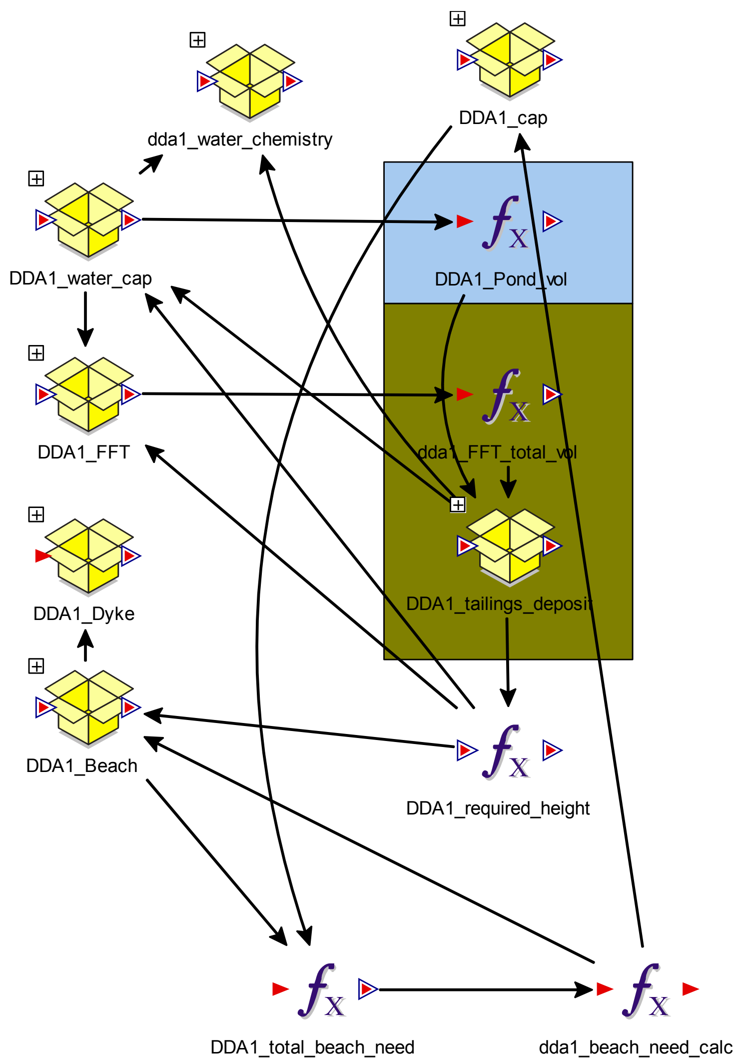


Figure A2- 14. Inpit_1 sub model (this structure is representative of all other DDAs).

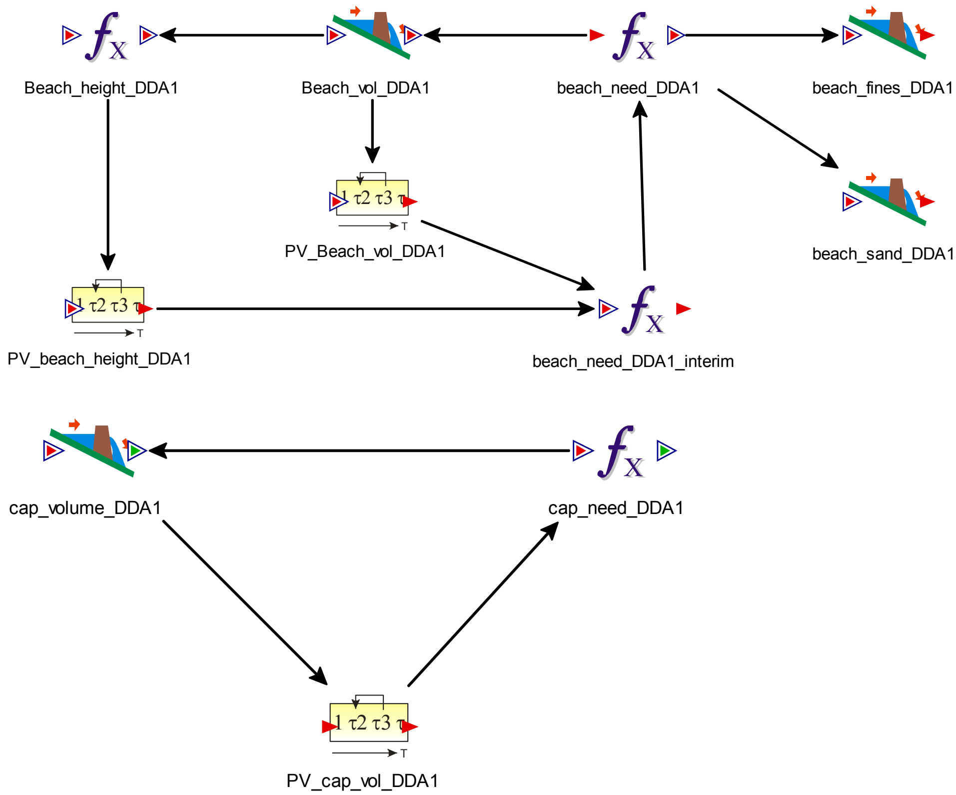


Figure A2- 15. DDA1_beach (top) and DDA1_cap (bottom) models

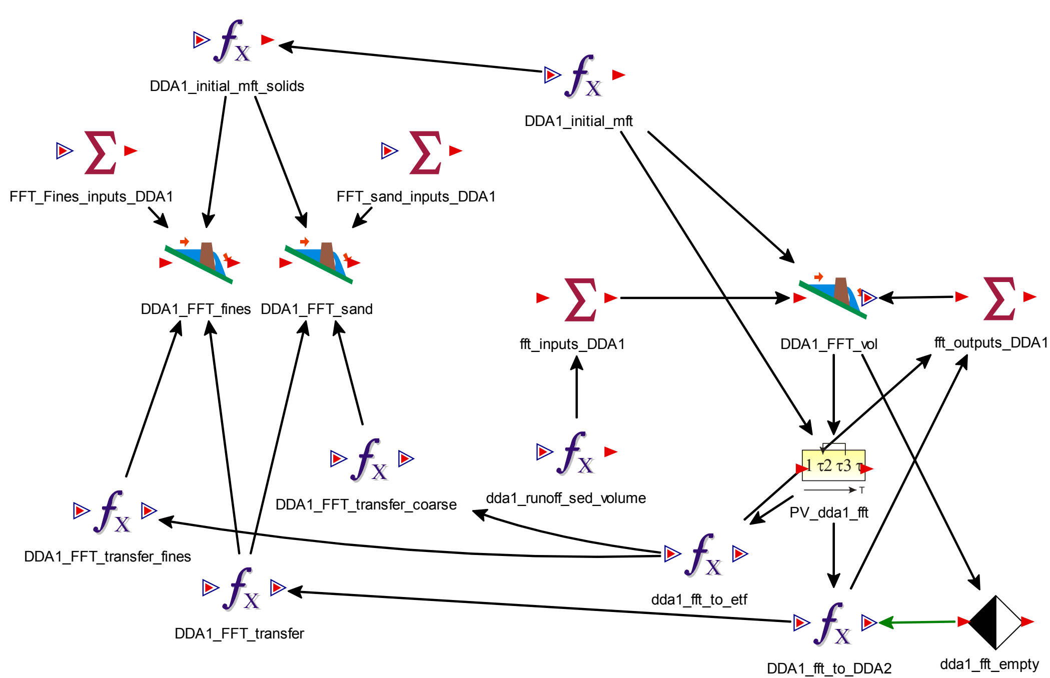


Figure A2- 16. DDA1_FFT model

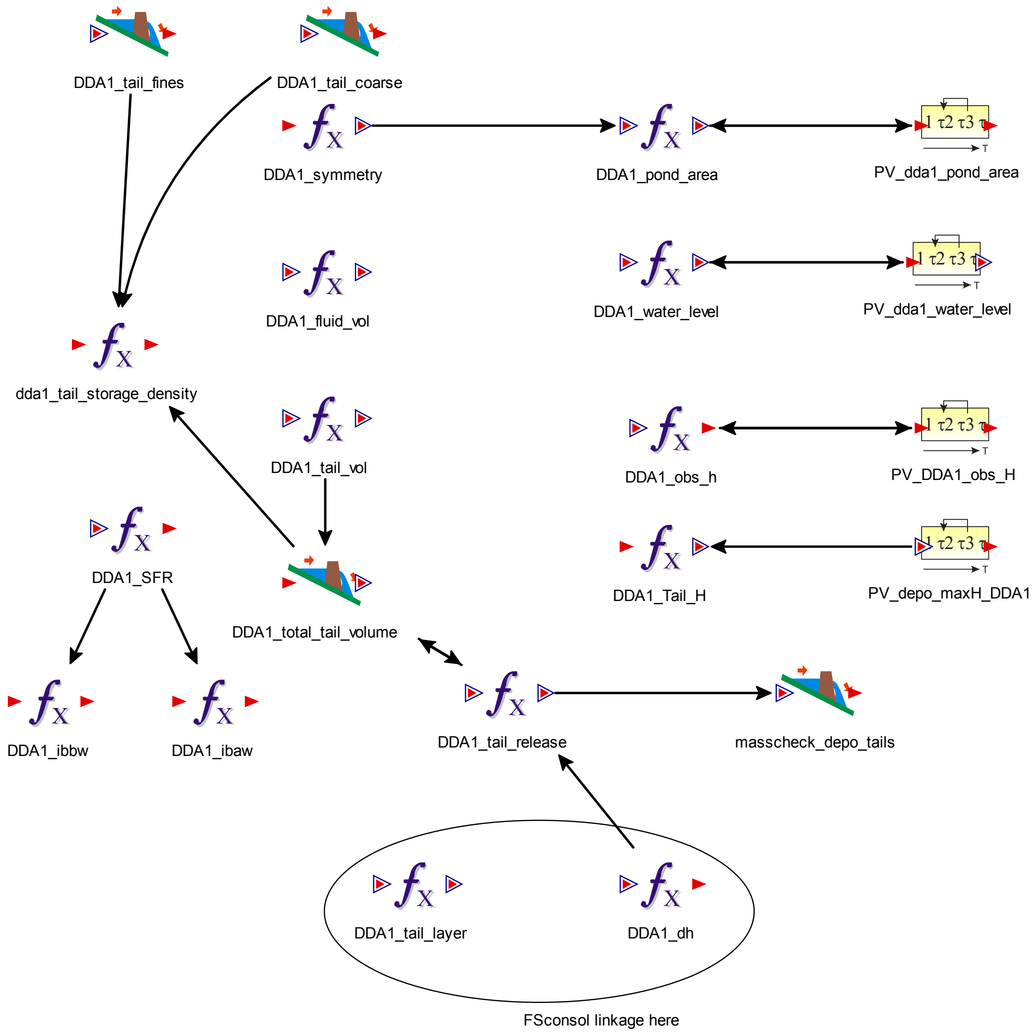


Figure A2- 17. DDA1_tailings deposit model

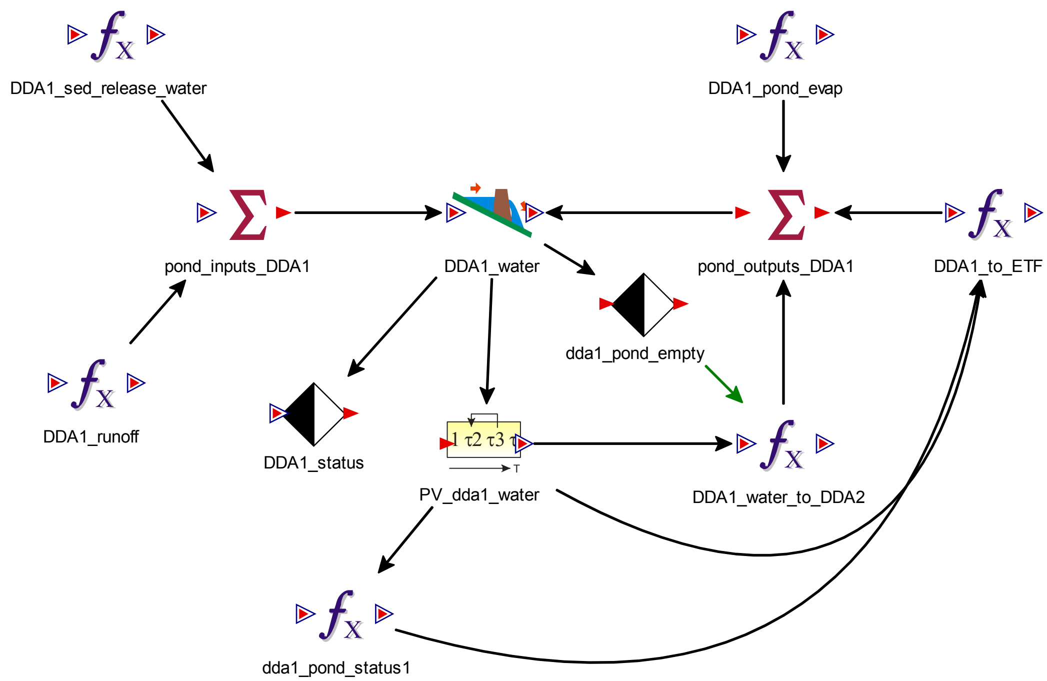


Figure A2- 18. DDA1_water_cap model.

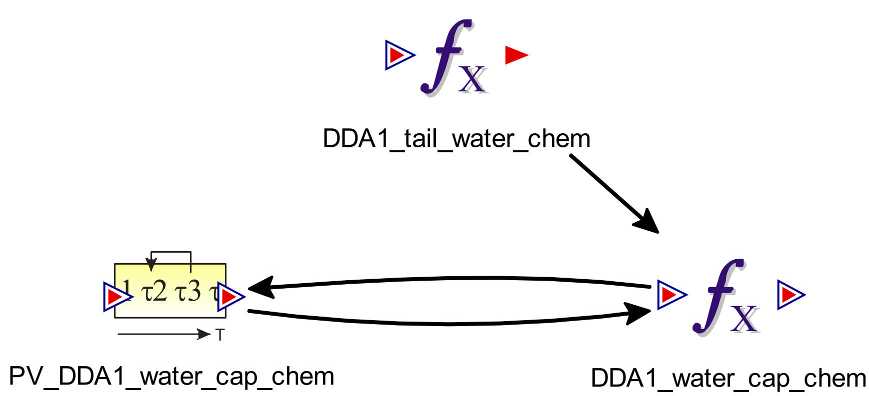


Figure A2- 19. DDA1_water_chemistry

Stage 1 Dewatering Models

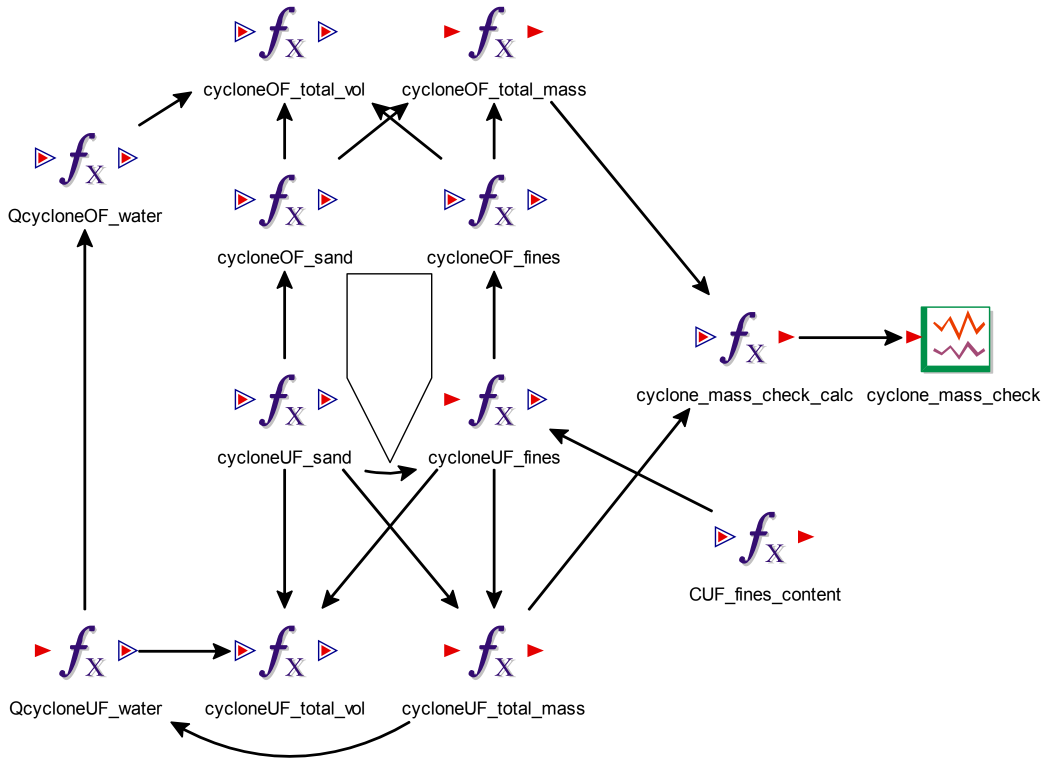


Figure A2- 20. Stage1_cyclone model

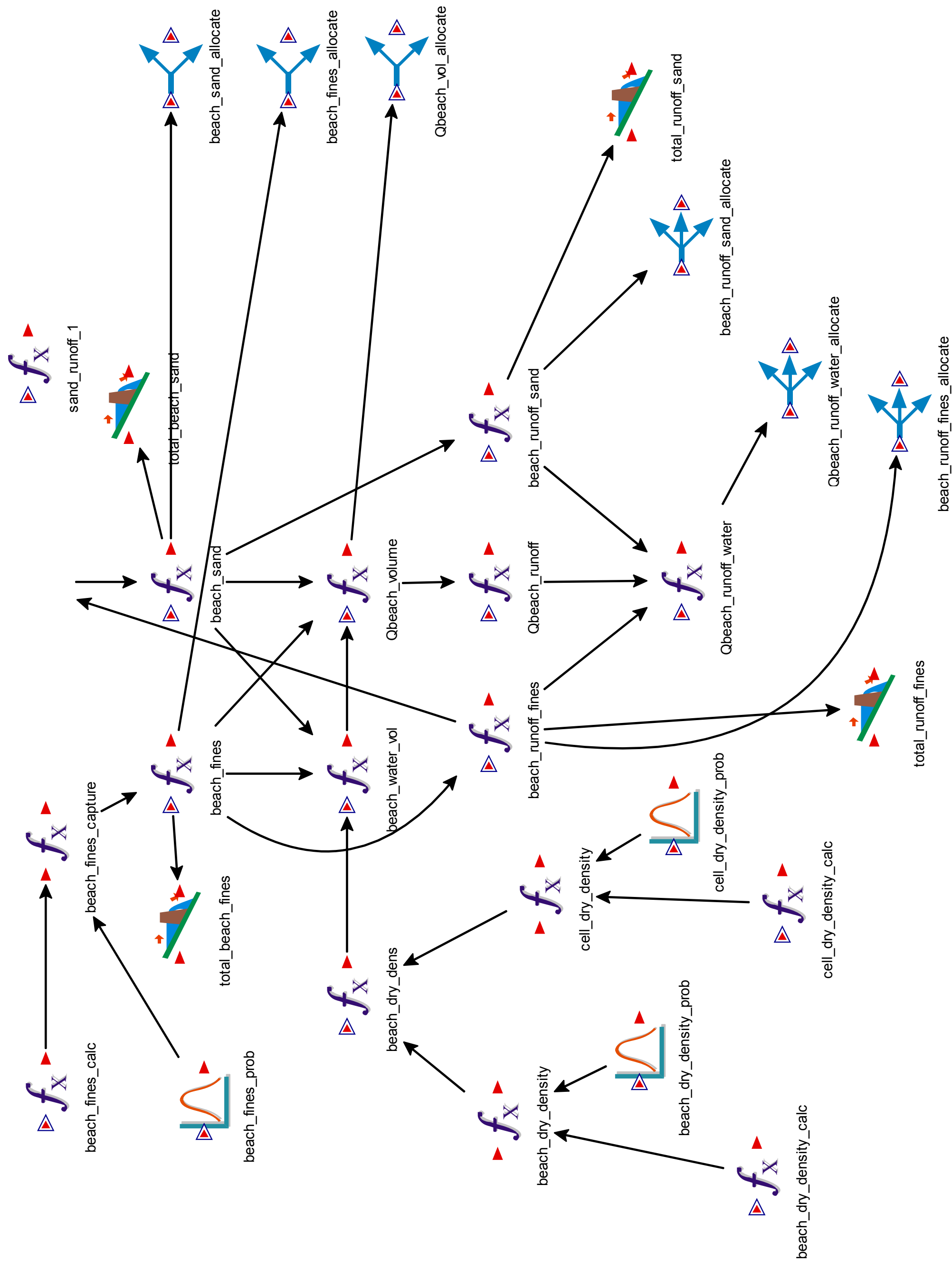


Figure A2- 21. Stage1_beaching_model.

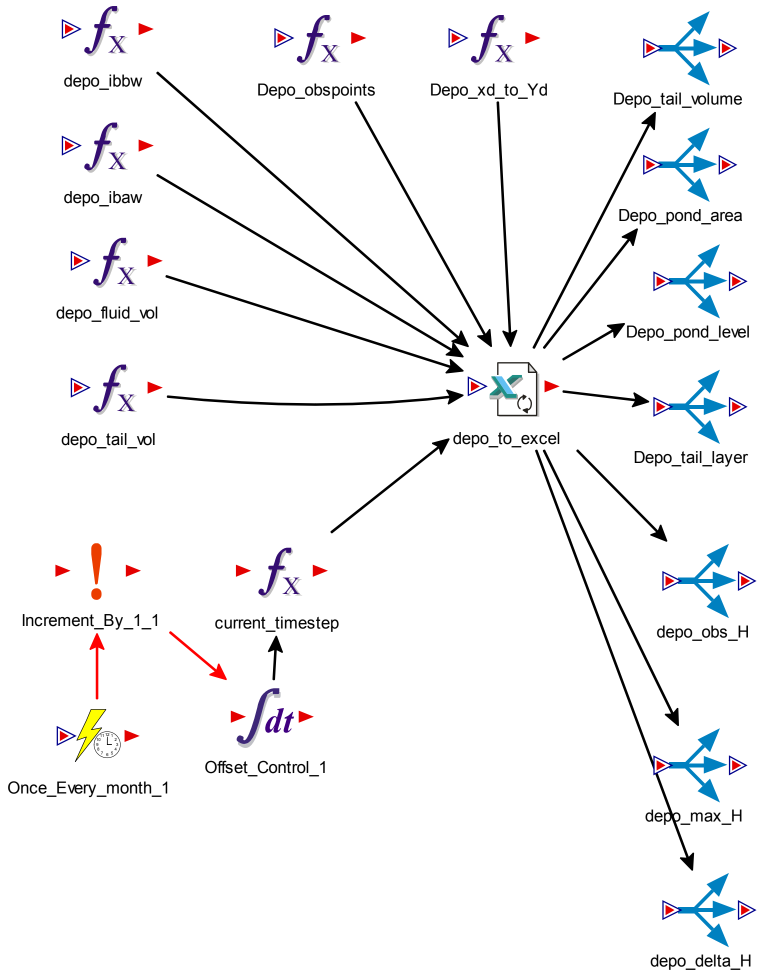


Figure A2- 22. Depostion_model structure

STAGE 2 DEWATERING MODELS

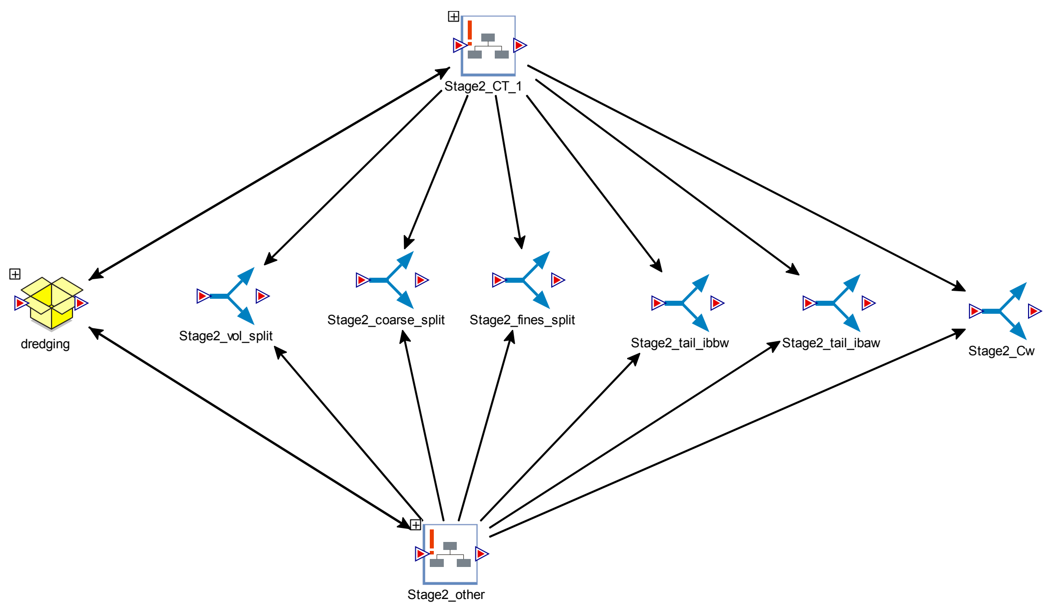


Figure A2- 23. Stage2_dewatering container model.

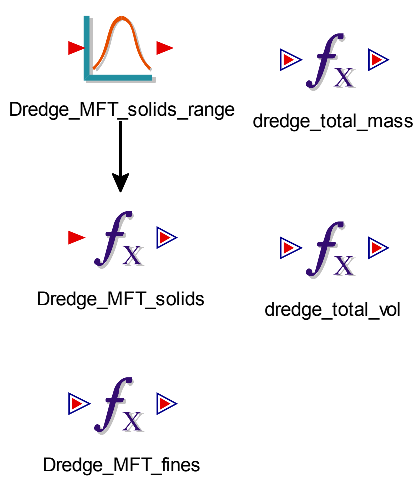


Figure A2- 24. Dredging model

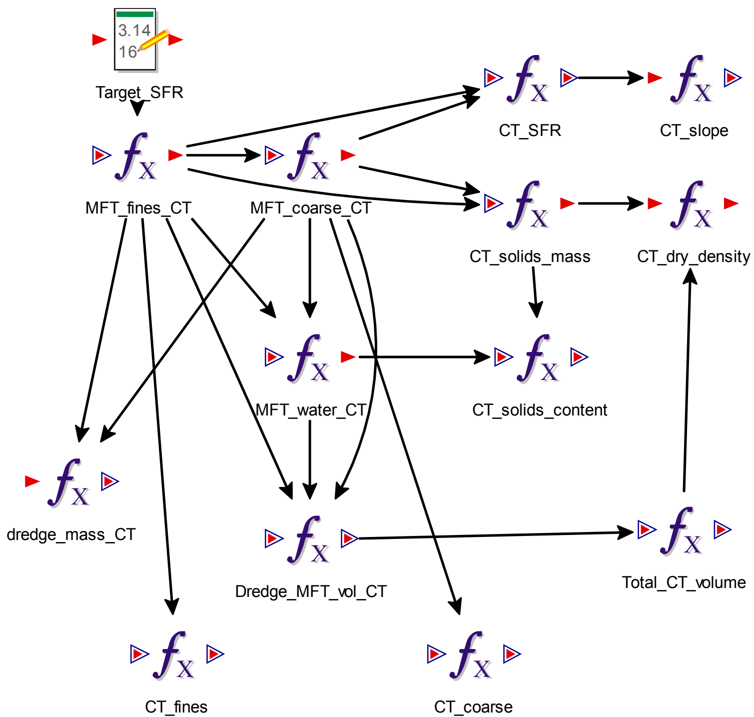


Figure A2- 25. Stage2_CT model.

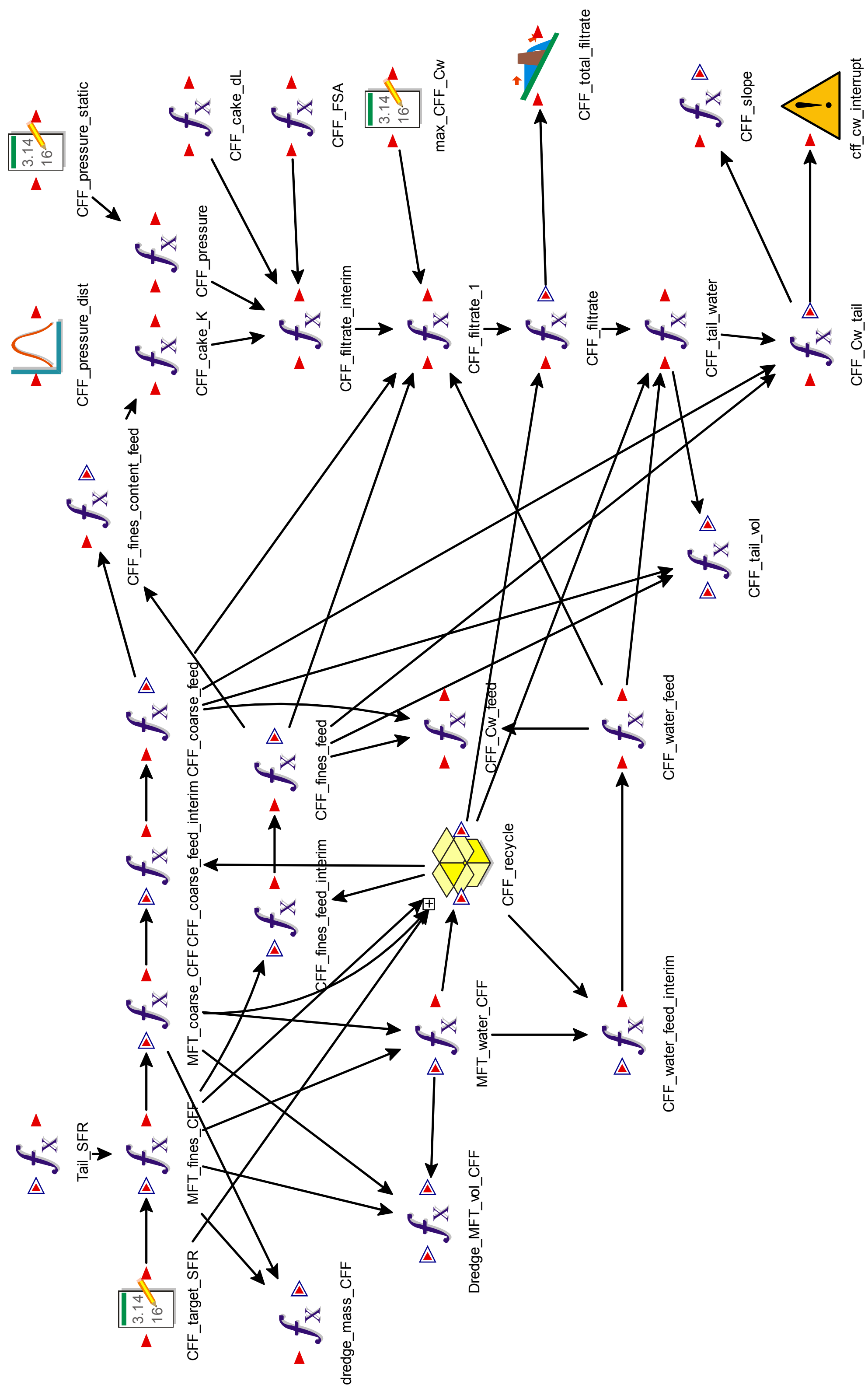


Figure A2- 26. Stage2_CFF model.

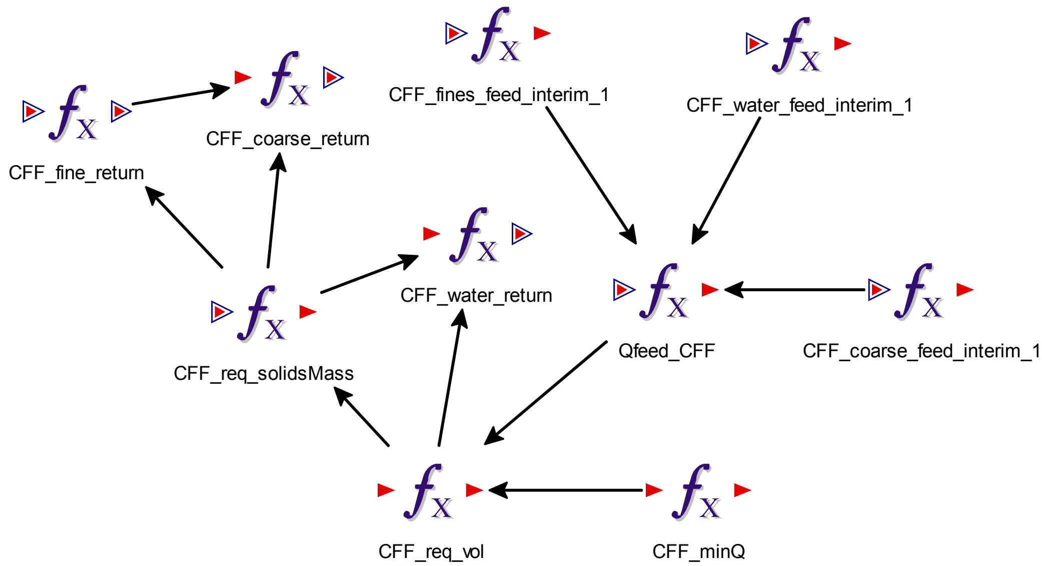
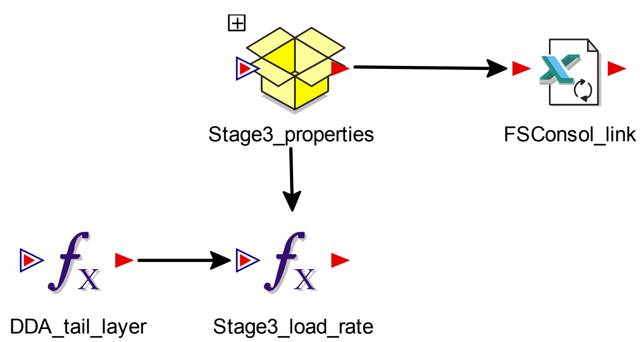


Figure A2- 27. CFF_recycle model.



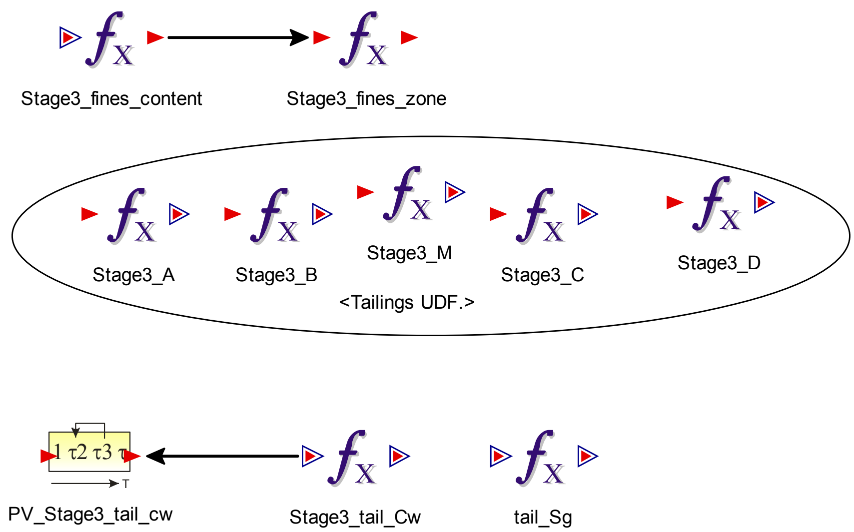


Figure A2- 28. Stage3_consolidation model and properties

Figure A2- 29. Input Container model. Provides linkage between input spreadsheet and GoldSim code.

APPENDIX 3 – OIL SANDS MODEL DATA SET ASSUMPTIONS

SYNCRUDE AURORA MINE AND TAILINGS PLAN DATA SET

The following information for the Syncrude Aurora mine and tailings management data were obtained from the Aurora North Environmental Impact Assessment report (Reeves 1996) the 2012 Annual Tailings Plan report (Syncrude 2012) and the 2010 Baseline Survey for Fluid Deposits (Syncrude 2010). A site plan of the Aurora mine is provided in Figure A3- 1. Aurora oil sand mine pit limits and DDA locations. The tailings plan incorporates three main dedicated disposal areas (DDAs) within the mine pit, Aurora East Pit (AEP), Aurora Centre Pit (ACP) and the Aurora West Pit (AWP). Each of these DDAs are sub-divided internally by 6 main dyke structures.

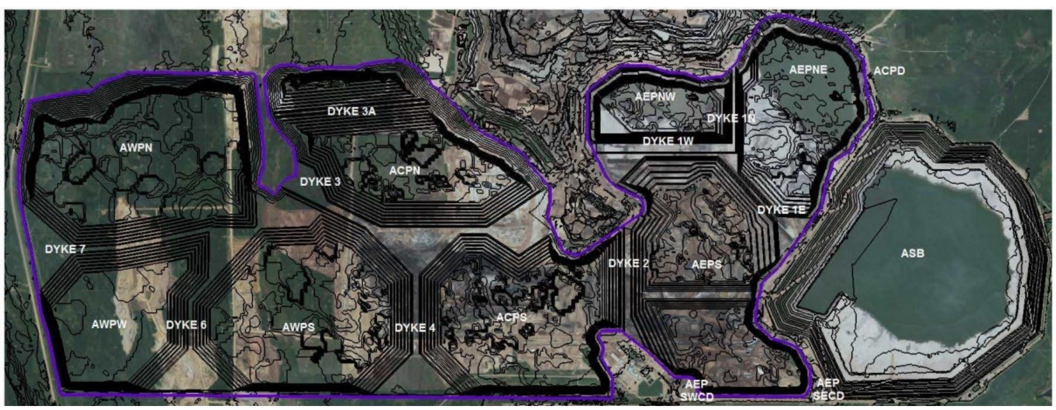


Figure A3- 1. Aurora oil sand mine pit limits and DDA locations.

The following Table A3- 1 to Table A3- 3 include the yearly mining, extraction and waste production data for the Aurora mine (Syncrude 2012). It includes the yearly ore, overburden, rejects, froth stream, and floatation tailings that are mined and processed at the Aurora mine. Rejects are incorporated into the overburden total waste deposits. The froth stream is sent to Mildred Lake and therefore is a loss of mass to the system. Floatation tailings are stored in the same manner as fluid fine tailings (FFT) or mature fine tailings (MFT). The floatation tailings dry density is 1.51 t/m^3.

Table A3- 1. Ore Summary Data

Year	Ore	Bitumen Grade	Bitumen	Water	Coarse Solids	Ore Fines	Fine Solids
	Mt/y	%	Mt/y	Mt/y	Mt/y	%	Mt/y
2013	104.6	11.3	11.82	3.45	71.5	17.1	17.9
2014	103.7	11.6	12.03	3.63	72.5	14.9	15.5
2015	114.9	11.3	12.98	4.02	78.9	16.5	19.0
2016	107.6	11.9	12.80	3.77	75.5	14.5	15.6
2017	122.2	11.2	13.69	4.28	87.5	13.7	16.8
2018	118.5	11.5	13.63	4.15	86.3	12.2	14.4
2019	123.0	10.9	13.41	4.31	85.9	15.8	19.4
2020	115.9	11.5	13.33	4.06	84.5	12.1	14.0
2021	108.9	11.4	12.41	3.81	78.3	13.2	14.4
2022	119.0	10.9	12.97	4.17	81.8	16.9	20.1
2023	124.0	10.7	13.27	4.34	84.4	17.8	22.0
2024	123.0	11	13.53	4.31	84.7	16.7	20.5
2025	119.0	11.4	13.57	4.17	82.6	15.7	18.6
2026	124.0	11.1	13.76	4.34	85.9	16.1	20.0
2027	123.0	10.6	13.04	4.31	85.5	16.4	20.2
2028	119.0	11	13.09	4.17	84.9	14.2	16.9
2029	124.0	11.7	14.51	4.34	89.1	13.0	16.1
2030	123.0	10.6	13.04	4.31	88.9	13.7	16.8
2031	119.0	11.1	13.21	4.17	85.7	13.4	16.0
2032	124.0	11.1	13.76	4.34	87.2	15.1	18.7
2033	123.0	11.4	14.02	4.31	86.4	14.9	18.3
2034	119.0	11.4	13.57	4.17	86.4	12.5	14.9
2035	124.0	10.5	13.02	4.34	87.7	15.3	19.0
2036	123.0	10.1	12.42	4.31	83.8	18.2	22.4
2037	119.0	10.7	12.73	4.17	83.0	16.0	19.1
2038	109.8	11.2	12.30	3.84	76.2	15.9	17.4
2039	109.7	11.2	12.29	3.84	76.0	16.0	17.6

Table A3- 2. Mine Waste Plan

Year	Ore Mt/y	Ore Mm^3/y	Total Waste M BCM	Total Mined M m^3/y	Suitable Waste %	Sand in Rejects Mt/y	Fines in Rejects Mt/y
2013	104.6	49.8	27.9	77.7	45.5	0.7	0.7
2014	103.7	49.4	22.7	72.0	24.8	0.7	0.7
2015	114.9	54.7	22.4	77.1	24.6	0.8	0.8
2016	107.6	51.2	26.1	77.3	42.5	0.8	0.8
2017	122.2	58.2	38.1	96.3	49.6	0.9	0.9
2018	118.5	56.4	45.0	101.4	24.9	0.8	0.8
2019	123.0	58.6	45.7	104.3	52.7	0.9	0.9
2020	115.9	55.2	50.5	105.6	43.5	0.8	0.8
2021	108.9	51.9	34.9	86.9	84.0	0.8	0.8
2022	119.0	56.7	41.5	98.2	82.4	0.8	0.8
2023	124.0	59.0	49.1	108.0	63.7	0.9	0.9
2024	123.0	58.6	40.6	99.2	71.9	0.9	0.9
2025	119.0	56.7	32.3	89.0	69.3	0.8	0.8
2026	124.0	59.0	23.5	82.5	23.0	0.9	0.9
2027	123.0	58.6	42.9	101.6	74.0	0.9	0.9
2028	119.0	56.7	46.5	103.2	75.3	0.8	0.8
2029	124.0	59.0	44.3	103.3	74.3	0.9	0.9
2030	123.0	58.6	50.1	108.8	47.8	0.9	0.9
2031	119.0	56.7	39.9	96.6	60.4	0.8	0.8
2032	124.0	59.0	23.1	82.0	52.6	0.9	0.9
2033	123.0	58.6	21.4	80.0	53.7	0.9	0.9
2034	119.0	56.7	24.0	80.8	72.6	0.8	0.8
2035	124.0	59.0	54.8	113.7	61.6	0.9	0.9
2036	123.0	58.6	71.4	130.0	34.5	0.9	0.9
2037	119.0	56.7	50.9	107.6	3.7	0.9	0.9
2038	109.8	52.3	37.0	89.3	0.0	0.8	0.8
2039	109.7	52.2	53.7	105.9	0.0	0.8	0.8

M BCM = million bulk cubic metres

Table A3- 3. Extraction Summary.

Year	Ore Mt/y	Sand in Froth Mt/y	Fines in Froth Mt/y	Sand in Floatation Tails Mt/y	Fines in Floatation Tails Mt/y	Floatation Tails Mm^3/y
2013	104.6	0.6	1.5	5.9	0.5	4.2
2014	103.7	0.6	1.4	5.8	0.4	4.1
2015	114.9	0.6	1.5	6.5	0.5	4.6
2016	107.6	0.6	1.5	6	0.5	4.2
2017	122.2	0.6	1.7	6.9	0.5	4.9
2018	118.5	0.6	1.7	6.7	0.5	4.7
2019	123.0	0.6	1.6	7	0.5	4.9
2020	115.9	0.6	1.6	6.5	0.5	4.6
2021	108.9	0.6	1.5	6.1	0.5	4.3
2022	119.0	0.6	1.5	6.7	0.5	4.8
2023	124.0	0.6	1.5	7	0.5	5
2024	123.0	0.6	1.6	7	0.5	4.9
2025	119.0	0.6	1.6	6.7	0.5	4.7
2026	124.0	0.6	1.6	7	0.5	4.9
2027	123.0	0.6	1.5	7	0.5	4.9
2028	119.0	0.6	1.5	5.7	0.5	4.8
2029	124.0	0.7	1.7	7	0.5	4.9
2030	123.0	0.6	1.5	7	0.5	4.9
2031	119.0	0.6	1.5	6.7	0.5	4.7
2032	124.0	0.6	1.6	7	0.5	4.9
2033	123.0	0.6	1.6	6.9	0.5	4.9
2034	119.0	0.6	1.6	6.7	0.5	4.7
2035	124.0	0.6	1.5	7.1	0.5	5
2036	123.0	0.6	1.4	7	0.5	5
2037	119.0	0.6	1.5	6.8	0.5	4.8
2038	109.8	0.6	1.4	6.2	0.5	4.4
2039	109.7	0.6	1.4	6.2	0.5	4.4

The cumulative mine excavation curve and cumulative deposit volumes for beached coarse tailings (referred to as SCT or straight coarse tailings by

Syncrude), composite tailings (CT) , floatation tailings, overburden dykes and in-pit overburden disposal is included as Figure A3- 2. The timing of the completion of in pit dykes separating each tailings disposal area is also included on Figure A3- 2. At the end of mining, FFT/MFT will be transferred from the external tailings facility to an end pit lake (EPL) within AWP.

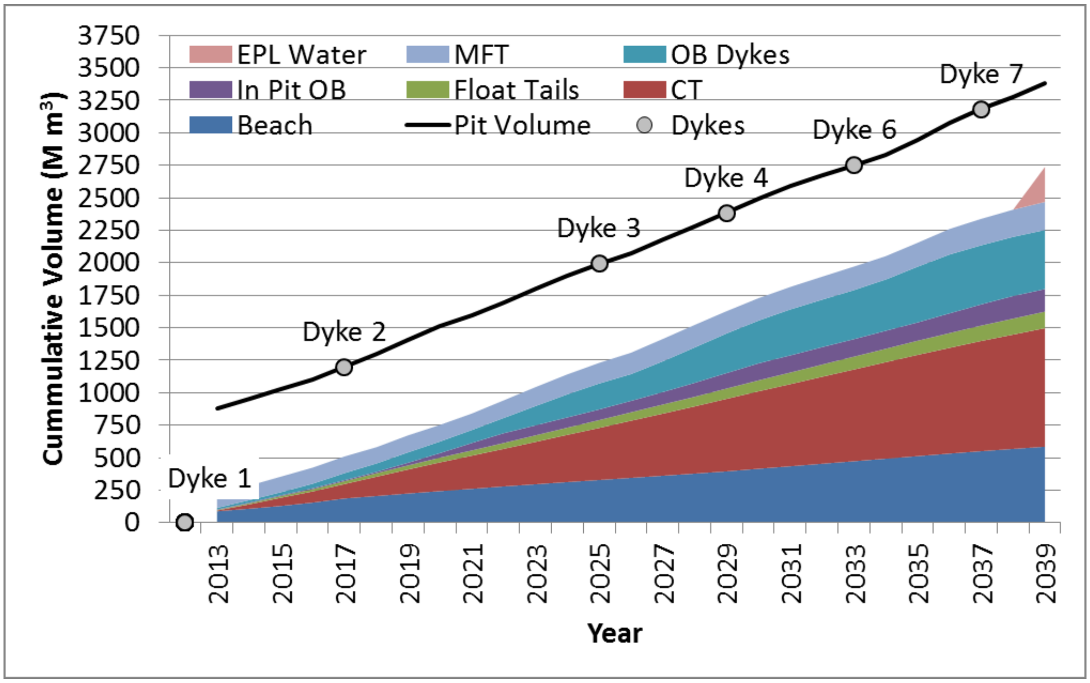


Figure A3- 2. Cumulative mined volume versus required in-pit storage.

The cumulative volume of deposits (required storage) for each DDA is summarized in the following Figure A3- 3 to Figure A3- 5. Deposits in each DDA will include beached coarse tailings, CT deposits floatation tailings, FFT, and process water. FFT and process water will be transferred among the DDAs over the life of the mine as space is available. At the end of mining, all FFT will be transferred to an EPL in the AWP DDA (Figure A3- 5).

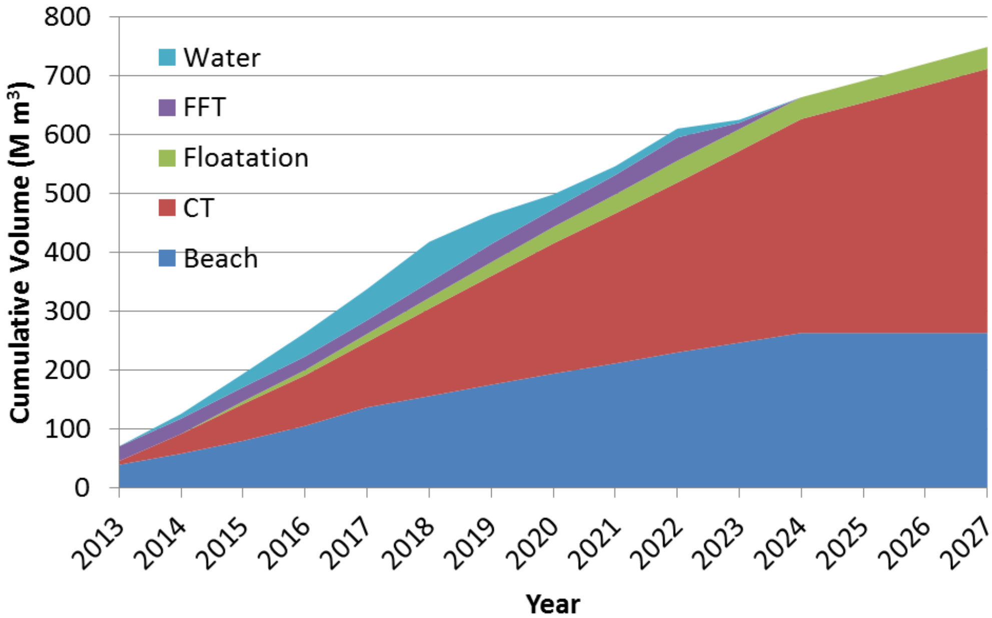


Figure A3- 3. AEP deposit volume curve.

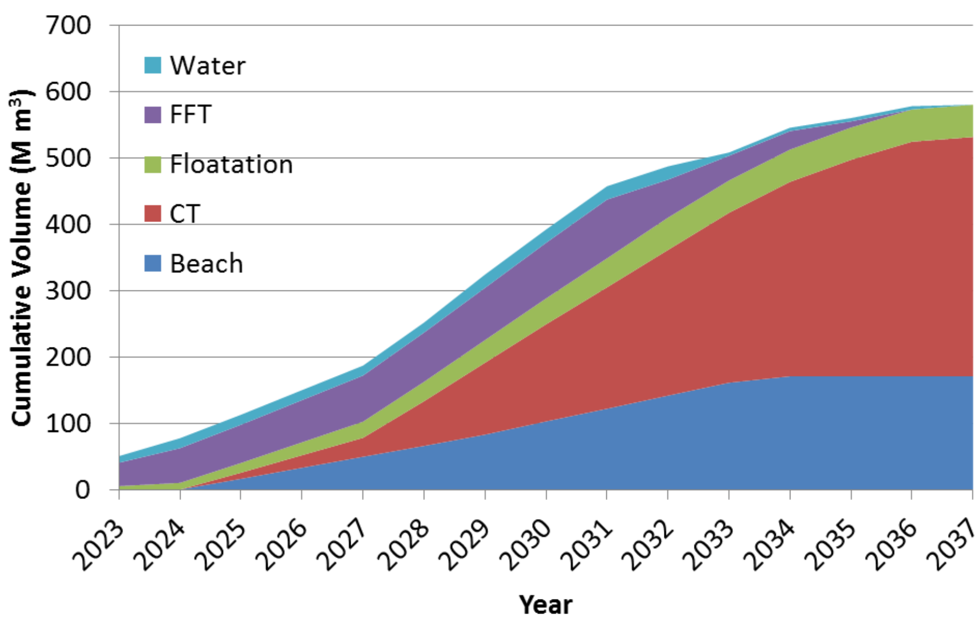


Figure A3- 4. ACP deposit volume curve.

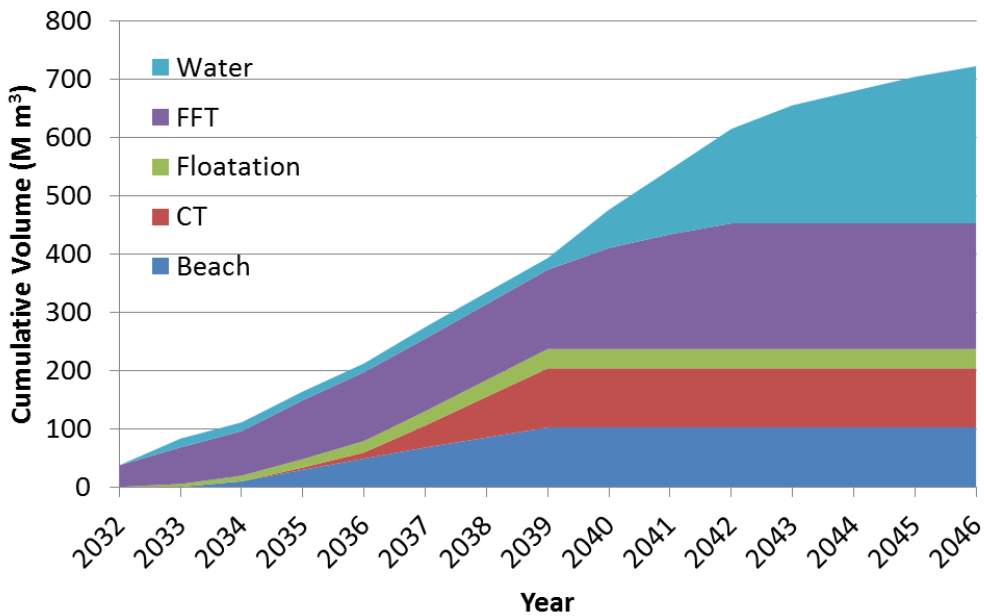


Figure A3- 5. AWP deposit volume curve (end of mining at 2039).

Model Oil Sands Mine Data Set for CT Technology

The model Aurora mine plan data set is based largely upon the Syncrude 2012 Tailings Plan report. Ore and overburden mining rates, ore body components and associated extraction rates/byproducts, outlined in Table A3- 1 to Table A3- 3, will be used for the model mine data set for the years 2013 to 2039. Historical data from start up to year 2013 are not included in the Syncrude (2012) report. To ensure the model mine pit limits at start of 2013 correspond to the Syncrude 2012 plan, the following assumptions were utilized. The historical mining rates will be based on a calculated ratio of MFT produced to bitumen produced. The Syncrude 2010 tailings plan included an historical MFT accumulation curve (Figure A3- 6) which will be used to correlate the amount of ore excavated during a given year.

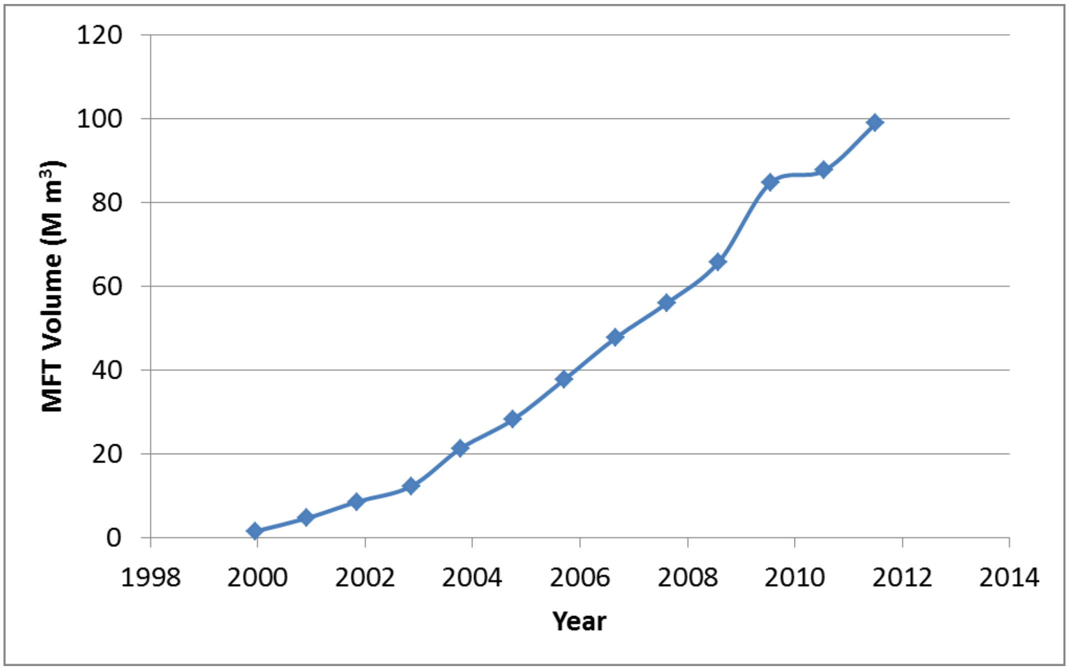


Figure A3- 6. Historical MFT Accumulation at Syncrude Aurora.

In the Syncrude 2010 report, only the first two years (2011 and 2012) utilized beaching only as a tailings technology (pre CT), therefore any MFT produced in these years would be similar to the years 2000 - 2010. The average MFT production was found to be 0.171 m^3 MFT/bbl bitumen for the years 2011-2012. The average bitumen recovery rate was 0.62 bbl bitumen/ tonne of ore mined. Using the historical MFT production rate and the bitumen and ore mining ratios, the historical rate of mining was determined and presented on Figure A3- 7.

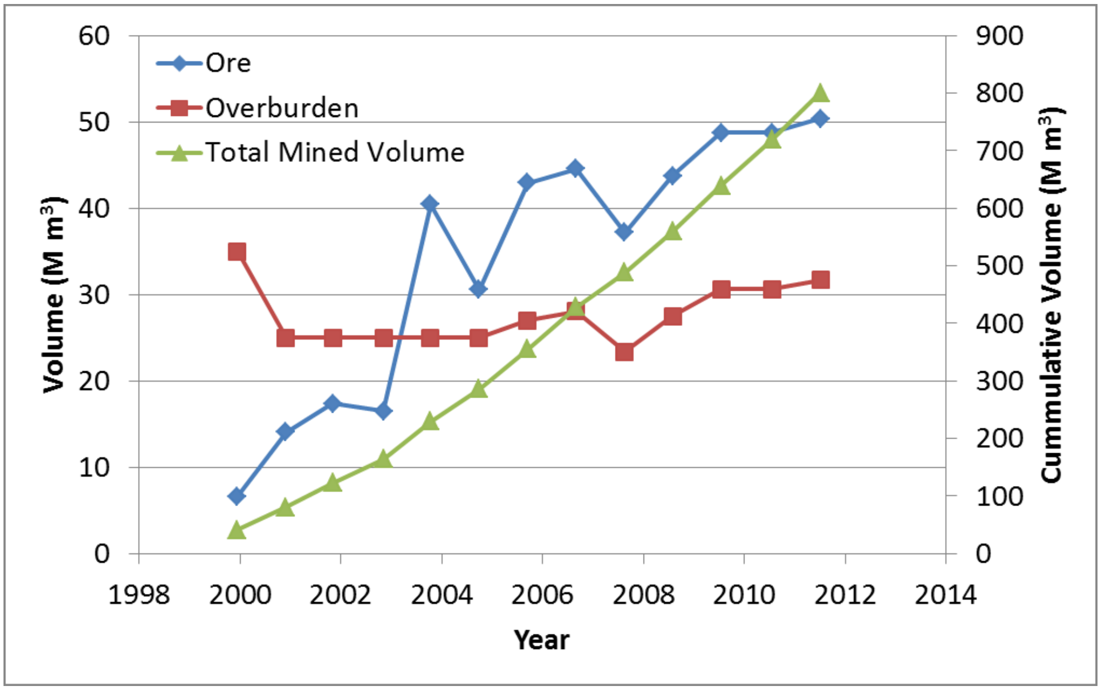


Figure A3- 7. Estimated historical mining rates for Aurora (2000-2012).

Based on a digitized site plan (Figure A3- 8, Syncrude 2012), the Aurora pit covers approximately 52.5 km^2 at the ground surface. Taking into account the pit wall slope of 2:1 to 3:1, the equivalent pit area is 48.8 km^2. Based on the total mined volume of ore and overburden from 2000 to 2012 estimated at 800 Mm^3 and 2578 Mm^3 calculated for 2013 to 2039, an average pit depth was determined to be 69.2 m. From the digitized site plan, there are seven DDAs with an average DDA area of 7.252 km^2. Assuming each DDA is of equal size, the total area is 50.764 km^2. This is very close to digitized pit limit area (52.5 km^2) and likely within the error of the digitizing process. The model oil sand mine data set will assume seven equally sized DDAs. The model DDAs will be square in shape with equal dimensions of 2.74 km by 2.74 km (52.5 km / 7 cells).

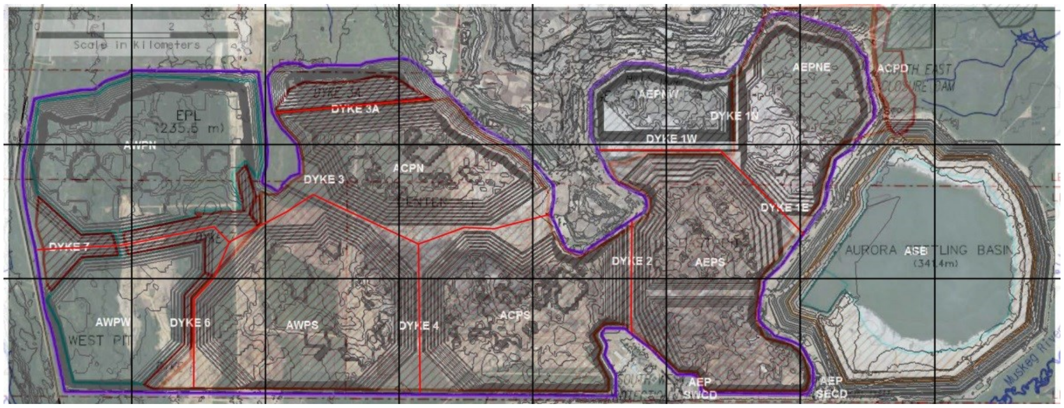


Figure A3- 8. Digitized Aurora site plan.

From the original Aurora mine plan, the average digitized dyke length is 2.75 km. Therefore, the assumption of model DDAs with a side dimension (dyke location) of 2.74 km is valid. The model mine pit limits and location of each of the seven model DDAs is included on Figure A3- 9. The model site data is overlain on the digitized actual Aurora mine plan for reference.

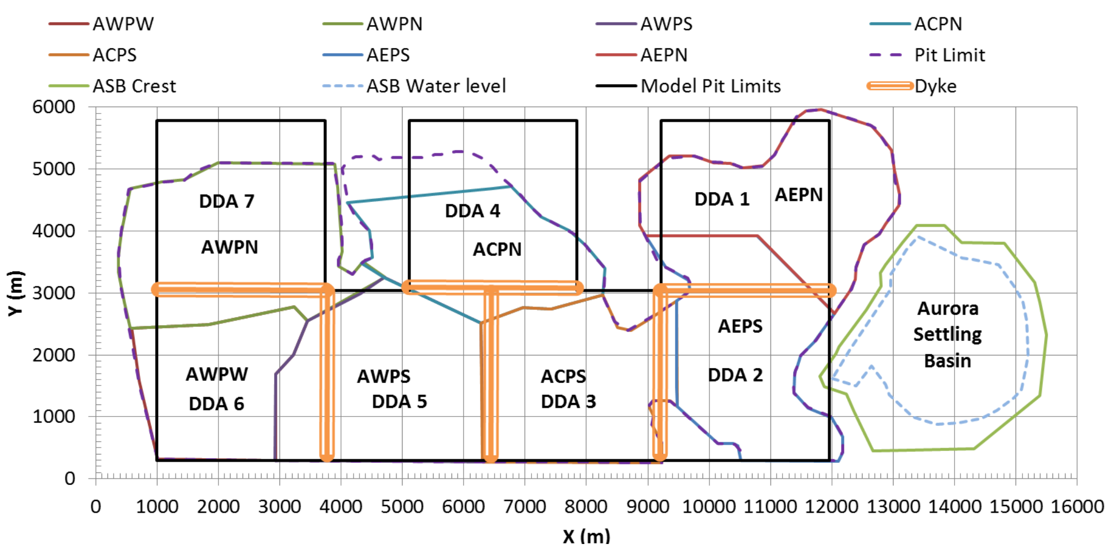


Figure A3- 9. Model mine pit limits and DDA locations.

Dykes separating DDAs will be constructed of overburden with the top 10 m of coarse sand tailings. The Aurora mine plan (Syncrude 2012) also assumes beach sand is deposited into each DDA in addition to CT. To match the beach sand deposition into each DDA, the model mine plan data set assumes a coarse tailings beach (~150 m wide) deposited on top of the dykes will provide a working surface for CT deposition. The model data set also assumes a sand cap of up to 4 m is placed on top of each CT deposit in the DDAs. A typical cross section of DDAs 2, 3, 5, and 6 depicting the inpit dyke configurations and CT tailings deposits is presented in Figure A3- 10.

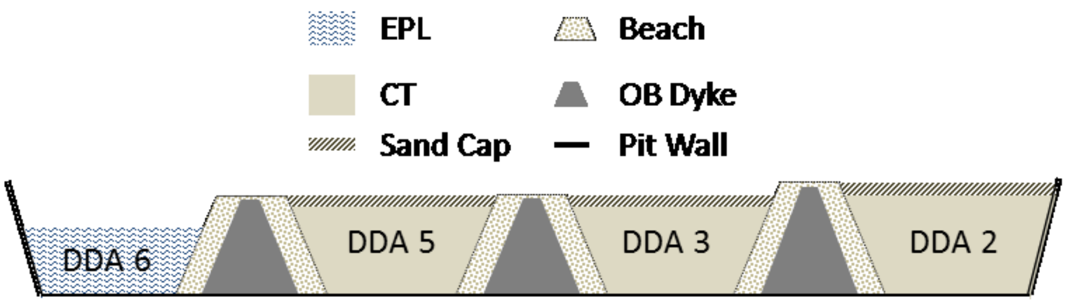


Figure A3- 10. Cross section of in pit dyke and CT tailings deposit configurations (not to scale). [EPL = end pit lake, CT = Composite/consolidated tailings, OB = overburden, DDA = dedicated disposal area].

Using the above assumptions, the cumulative beach volumes deposited over the life of the mine closely matches the deposition planned volumes of the Aurora plan (Figure A3- 11). The following sections detail the material demand for beaching, capping and dyke construction for each DDA which is summarized in Figure A3- 12.

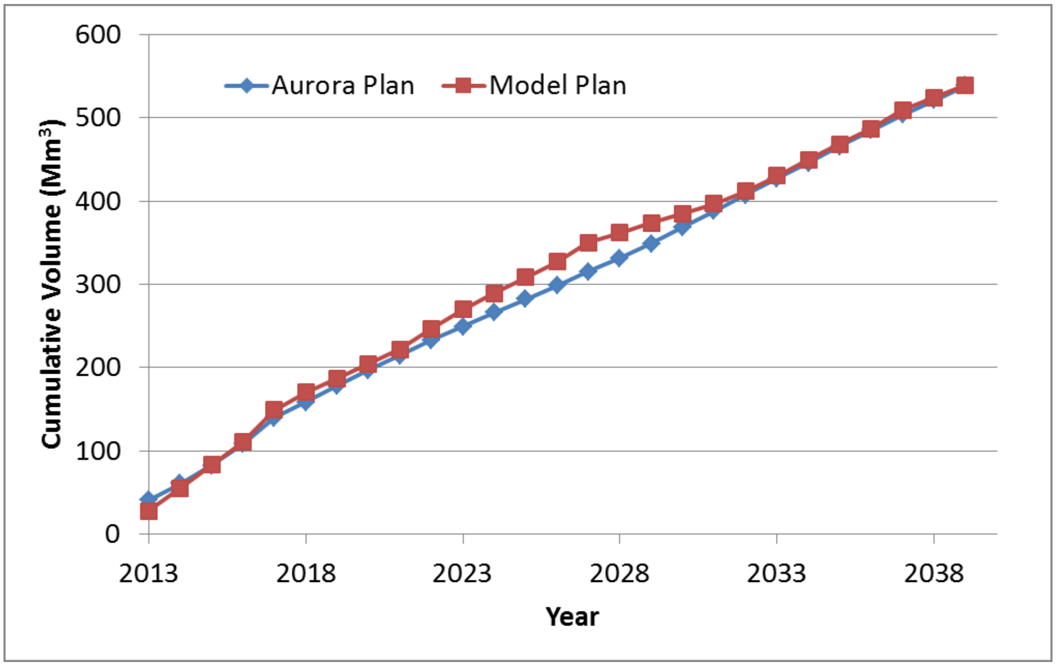


Figure A3- 11. Coarse sand demand for the Aurora mine plan versus the model data set.

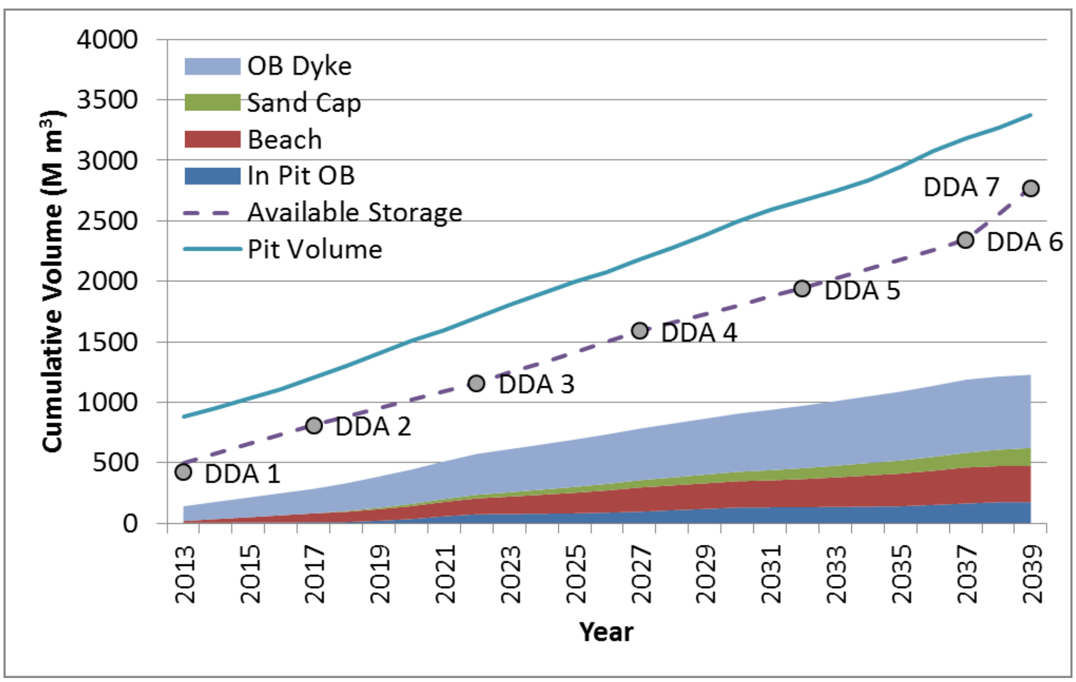


Figure A3- 12. Overburden and coarse sand demand.

DDA 1 (EAST PIT AREA)

The total mined volume of DDA 1 was calculated to be 474 Mm^3. From the Reeves (1997) report, the East pit area will be filled with tailings to 15 m above existing grade (including sand cap) or an elevation of 84 m above the pit base. Therefore, CT will be deposited to an elevation of 80 m above the pit base. DDA 1 is bounded by pit walls and dyke 1 to the south. The overburden dyke 1 will be constructed to an elevation of 80 m with the top 10 m constructed from beach sand (3.56 Mm^3) and finally capped with 4 m of sand. By year 14 (2013), all overburden for dyke 1 has been placed to an elevation (70 m). To provide a working surface, a 150-170 m wide coarse tailings beach will be deposited on top of each overburden dyke. The height-volume function for the starter beach is represented as Equation A3.1. To provide a buffer for an external waste dump, beaching of coarse tailings is required along the west wall of DDA 1. Therefore a beach of approximately 250 m wide, the full depth and length of the west wall is included in the material demand for DDA 1 (Equation A3.2). A closure dyke for DDA 1 is also required and will be constructed from coarse sand (3.56 Mm^3). The stage height-volume function is presented in Equation A3.3.

[A3.1] Volume = 437920*height

[A3.2] Volume = 609500*height

[A3.3] Volume = 4*(321950*height+428125)

A line of symmetry in DDA 1 occurs at 1250 m from the eastern wall, therefore only one observation point is required (Figure A3.12). The observation point will be at 410 m from the east wall (1/3 of distance to the line of symmetry) and 1190 m south of the northern wall. CT tailings will be deposited from a line along the east and west walls which will represent a series of spigots along the length of the wall.

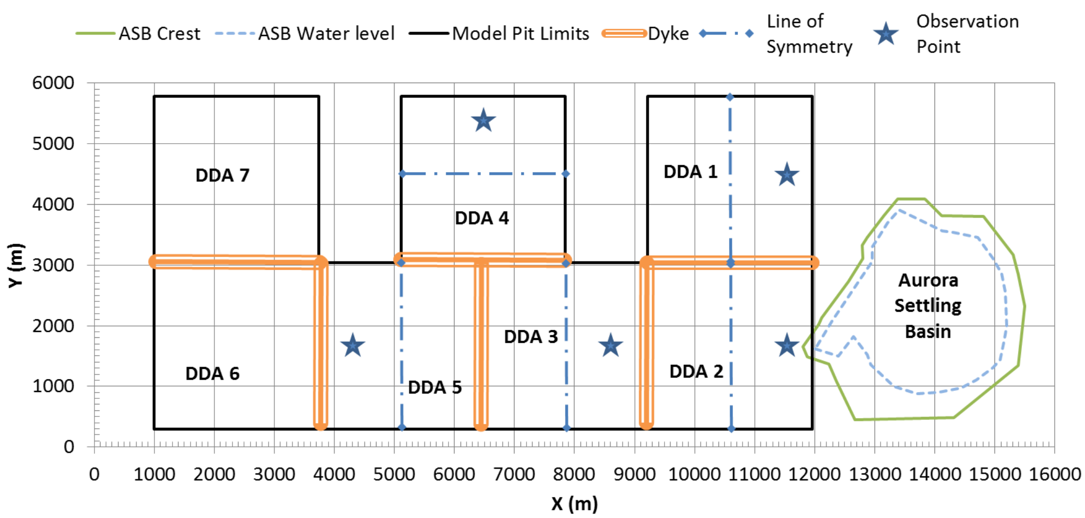


Figure A3.12. DDA observation points and lines of symmetry.

DDA 2 (East Pit Area)

The total mined volume in DDA 2 was calculated to be 489 Mm^3. DDA 2 will also be filled to 15 m above grade to account for settlement of the tailings. DDA 2 is bounded by pits walls, and dyke 1 and 2. Dyke 2 will be constructed to an elevation of 70 m with 35 Mm^3 constructed from overburden and the remaining 66.7 Mm^3 from coarse tailings. The final Dyke 2 elevation will be 80 m. Tailings will be capped with 4 m of coarse sand tailings. Coarse tailings will also be beached on all dyke surfaces in DDA 2. The volume-height relationship for beaching in DDA 2 is presented in Equation A3.4. A Closure dam is also required in the south end of DDA 2 constructed similar to DDA 1 (Equation A3.5).

[A3.4] Volume = 755820*height

[A3.5] Volume = 3*(321950*height+428125)

A line of symmetry in DDA 2 occurs at 1190 m from the eastern wall, therefore only one observation point is required (Figure A3.12). The observation point will be at 410 m from the east wall (1/3 of distance to the line of symmetry) and 1190 m north of the southern wall. CT tailings will be deposited from a line along the east and west walls which will represent a series of spigots along the length of the wall.

DDA 3 (CENTRE PIT AREA)

The total mined volume in DDA 3 is 488 Mm^3 and is bounded by Dyke 2 to the east, dyke 3 to the west and part of Dyke 4 to the north. According to Reeves (1996), tailings in DDA 3 will be deposited to 69.2 m (existing ground surface). Settlement of tailings in DDA 3 will promote surface runoff and drainage from the DDAs 1 and 2 to the planned EPL in DDAs 6 and 7. Dyke 3 will be constructed of overburden to an elevation of 64 m and to 74 m with coarse sand. The volume of overburden used in dyke 3 will be increased by 50% to account for flaring where it meets dyke 4. Coarse tailings will be beached on all dyke surfaces in DDA 3 (Equation A3.6) in addition to a 4 m cap on top of the CT deposit.

[A3.6] Volume = 994637*height

Given the complex shape of DDA 3, a representative symmetric deposition cell was constructed for simulation of tailings deposition in to DDA 3. The deposition cell has the same equivalent volume as DDA 3. A line of symmetry in the new DDA 3 occurs at 1370 m from the eastern boundary, therefore only one observation point is required (Figure A3.12). The observation point will be at 630 m from the east wall (1/3 of distance to the line of symmetry) and 1165 m north of the southern wall. CT tailings will be deposited from a line along the east and west boundaries which will represent a series of spigots along the length of the wall.

DDA 4 (CENTRE PIT AREA)

The total mined volume in DDA 4 is 474 Mm^3 and is bounded by dyke 4 to the south and pit walls. Dyke 4 will be constructed to the same elevation as dyke 3. DDA 4 will be filled in the same manner as DDA 3. In addition to CT deposition, overburden will also be deposited into DDA 4 including a separation dyke (57 Mm^3) required at the north pit wall to provide separation from a neighboring mine site. The stage relationship for the separation dyke is presented as Equation A3.7. Coarse tailings will be beached on all dyke surfaces in DDA 4 (Equation A3.8) in addition to a 4 m cap on top of the CT deposit.

[A3.7] $$\text{Volume} = -6440*\text{height}^2 + 1277696*\text{height}^2$$

[A3.8] $$\text{Volume} = 437920*\text{height}$$

A line of symmetry in DDA 4 occurs at 1320 m south of the northern wall, therefore only one observation point is required (Figure A3.12). The observation point will be at 440 m from the north wall (1/3 of distance to the line of symmetry) and 1370 m east of western wall. CT tailings will be deposited from a line along the north and south walls which will represent a series of spigots along the length of the wall.

DDA 5 (West Pit Area)

The total mined volume in DDA 5 is 488 Mm^3 and is bounded by dyke 3 to the east, dyke 4 to the north, dyke 5 to the west and a pit wall to the south. Dyke 5 will be constructed to the same elevation as dyke 3. DDA 5 will be filled in the same manner as DDA 3 and 4 until the end of mining. Coarse tailings will be beached on all dyke surfaces in DDA 5 (Equation A3.9) in addition to a 4 m cap on top of the CT deposit.

[A3.9] $$\text{Volume} = 994637*\text{height}$$

Given the complex shape of DDA 5, a representative symmetric deposition cell was constructed for simulation of tailings deposition in to DDA 5. The deposition cell has the same equivalent volume as DDA 5. A line of symmetry in the new DDA 5 occurs at 1370 m from the western boundary, therefore only one observation point is required (Figure A3.12). The observation point will be at 460 m from the west wall (1/3 of distance to the line of symmetry) and 1165 m north of the southern wall. CT tailings will be deposited from a line along the east and west walls which will represent a series of spigots along the length of the wall.

DDA 6 and 7 (West Pit Area)

Dedicated disposal areas 6 and 7 will be utilized as EPL and provide storage for any remaining FFT at the end of mining. At the end of mining, site runoff from DDAs 1-5 will be directed to DDA 6 and 7. Dyke 6 will be constructed to an elevation of 64 m to provide separation of the fluid containment in DDA 6 from mining operations in DDA 7.

External Tailings Facility

An existing tailings facility (ETF) contains 79.9 Mm^3 of fine tailings and 46.4 Mm^3 of process water and at a pond elevation of 44 m. The current pond surface area is 6.19 km^2. The ETF is constructed to its maximum extent providing a maximum storage capacity of 130 Mm^3 (pond elevation of 44.6 m). No further construction is permitted on the ETF. The stage volume-height capacity of the ETF was estimated as Equation A3.9.

[A3.9] $$\text{Volume} = 6.19e{+}6 * (\text{height}-23.63)$$

Model Decision Logic

For each time step, the model must determine where to direct the overburden material (dykes, in pit storage, or out of pit storage) and tailings (beach, dyke, cap, hydrocyclone, or other dewatering technology). The following is a chronology of decisions based on the model mine data set to determine where mine waste and tailings are deposited. The decisions assume the model will be starting at year 2013 and deposition will commence in DDA 1.

Overburden

At each time step, the model will check if the current DDA has sufficient freeboard (i.e. 3 m above the tailings and process water level). If sufficient freeboard is available, overburden will then be split between satisfying the in-pit and ex pit waste piles. If sufficient free board is not available, overburden will be used to increase height of containment dykes based on the dyke stage curve (height versus construction volume). Once a dyke has reached is maximum elevation, overburden will be directed to construct subsequent dykes.

EXTRACTION TAILINGS

For each time step, the model will first check if the current DDA has reached its maximum capacity (volume or elevation). If the DDA is not full, the model will then determine if the required coarse tailings beach demand has be met for the current tailings deposit elevation/volume. If beaching is required, the required fraction of the tailings stream will be diverted to satisfy the beaching demand. The model will then check if the coarse tailings capping demand has been met for the previous DDA. If not, a fraction of the tailings stream will also be diverted to satisfy the capping demand. The model will then ensure sufficient freeboard is available before depositing the remaining tailings into the DDA. If there is insufficient room, tailings will be beached in the external tailings facility. Tailings deposition will cease in the current DDA once a maximum height/volume has been achieved. Fine tailings runoff from beaching and capping will be directed to the currently active DDA. Fine tailings will be transferred to subsequent DDAs as in pit space become available and the dyke is constructed to provide sufficient freeboard.

TAILINGS CONSOLIDATION PROPERTIES

Chapter 7 demonstrated the relationship between fines content of the tailings streams and the large strain consolidation properties. Based on fluctuations within the extraction process and ore body changes, the fines content of the tailings streams will not be constant over the life of the mine, therefore the material properties will also change. To reduce the numerical complexity of having new tailings properties updated for every small change in fines content, a select sample of material properties will be chosen to represent a range of fines contents. For a typical CT operation, the target fines content of the deposited tailings may fluctuate from ~ 22% to 14% or sand to fines ratio, SFR, from 3.5:1 to 6:1. Using the equations developed in Chapter 7, large strain consolidation parameters (A, B, C, D) were calculated for the minimum, maximum and average fines content. For a 5 year deposition scenario, the difference in elevation between the minimum and maximum fines content tailings was approximately 2.5 m. Therefore a 4% difference in fines content resulted in a 1.5 m elevation difference upon deposition. If the difference in fines content was reduced to only 2% between calculated material parameters, the elevation change was only 0.5 to 0.9 m. Therefore, when determining properties of tailings deposits, a minimum 2% change in fines content from the previous time step is required before tailings properties will be updated. The following Table A3- 4 summarizes the ranges of fines contents and associated tailings properties for tailings.

Table A3- 4. Tailings properties as a function of fines content.

Fines Content	SFR	Fines Content range	A	B	C	D
14	6.1	13-15	0.81	-0.067	1.0E-04	5.47
16	5.3	15-17	0.85	-0.088	4.9E-05	5.47
18	4.6	17-19	0.88	-0.11	2.6E-05	5.47
20	4.0	19-21	0.92	-0.12	1.5E-05	5.47
22	3.5	21-23	0.94	-0.14	8.6E-05	5.47

REFERENCES

Reeves, B. 1996. Environmental Impact Assessment for the Syncrude Canada Limited Aurora Mine. BOVAR Environmental, 1996.

Syncrude Canada Ltd. (Syncrude). 2009. 2010 Annual tailings plan submission Syncrude Aurora North (Leases 10, 12 and 34). 30 September 2009.

Syncrude Canada Ltd. (Syncrude). 2012. 2012 Annual tailings plan submission Syncrude Aurora North (Leases 10, 12 and 34). 30 September 2012.

APPENDIX 4 – CROSS FLOW FILTRATION MODEL ASSUMPTIONS

The following information was used to for the simulation of the novel technology, cross flow filtration (CFF). The Aurora mine plan (Syncrude 2012) used in the CT model data set (Appendix 3) will also be utilized for the CFF simulations. Ore and overburden mining rates, ore body components and associated extraction rates/byproducts for the years 2013 to 2039 are outlined in Table A3- 1 to Table A3- 3. The CFF simulations will also use the digitized site plan (Figure A4- 1, Syncrude 2012) resulting in an equivalent pit area of 48.8 km^2. Based on the total mined volume of ore and overburden from 2000 to 2012 estimated at 800 Mm^3 and 2578 Mm^3 calculated for 2013 to 2039, the average pit depth for the CFF simulations will be 69.2 m. Using the digitized site plan as a guide, the CFF mine plan will require 5 DDAs. DDA1 and 4 are of equal dimensions (2.74 km by 2.74 km). DDA 2 and 3 will also be of equal dimensions of 4.1 km by 2.74 km. Tailings will be deposited into DDA 1, 2, and 3. Overburden will be deposited into DDA 4. At the end of mining, DDA 5 will be flooded with process water and form an end pit lake bounded by the tailings deposit in DDA 3.

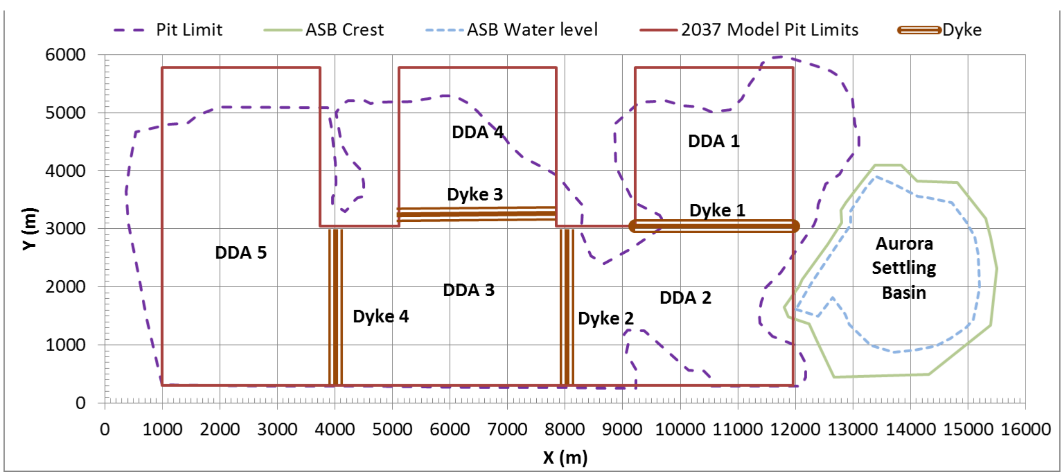


Figure A4- 1. Model mine pit limits and DDA locations.

DDA 1 (East Pit Area)

The total mined volume of DDA 1 was calculated to be 474 Mm^3 with a surface area of DDA 1 is 7.51 Mm^2. From the Reeves (1996) report, the East pit area will be filled with tailings to 12 m above existing grade (including overburden cap) or an elevation of 81 m above the pit base. Therefore, CFF tailings will be deposited to an elevation of 79 m above the pit base. DDA 1 is bounded by pit walls and dyke 1 to the south. The overburden dyke 1 will be constructed to an elevation of 79 m. By year 14 (2013), overburden for dyke 1 has been placed to an elevation (70 m). To provide a buffer for an external waste dump, beaching of coarse tailings was required along the west wall of DDA 1. By 2014, sufficient beach sand (47.3 Mm^3) has been placed in DDA 1 to develop the buffer. All existing FFT and water will be removed from DDA 1 and transferred to the ETF so it cab be spiked into the whole tailings. A 10 m high closure dyke for DDA 1 is also required and will be constructed from overburden (14.24 Mm^3). The stage height-volume function is presented in Equation A4.1.

[A4.1] Volume = 4*(321950*height+428125)

A line of symmetry in DDA 1 occurs at 1250 m from the eastern wall, therefore only one observation point is required (Figure A4.2). The observation point will be at 410 m from the east wall (1/3 of distance to the line of symmetry) and 1320 m south of the northern wall. CFF tailings will be deposited from a line along the east and west walls which will represent a series of spigots along the length of the wall.

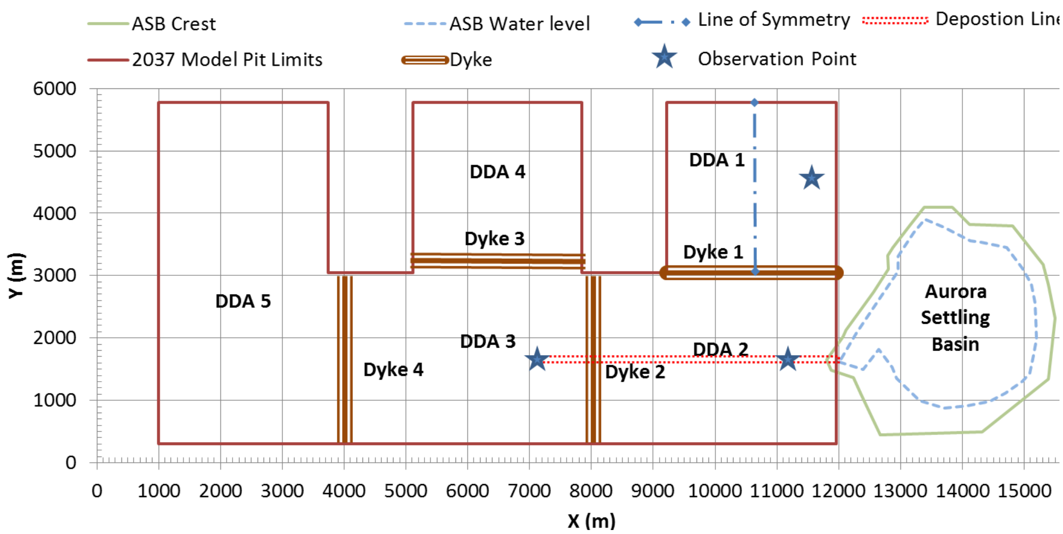


Figure A4.2. DDA observation points and lines of symmetry.

DDA 2 (East Pit Area)

The total mined volume in DDA 2 was calculated to be 717.7 Mm^3 with a surface area of 11.2 Mm^2. DDA 2 will be filled to 16 m above grade to account for settlement of the tailings. DDA 2 is bounded by pits walls, dyke 1 and 2. Dyke 2 will be constructed to an elevation of 20 m with overburden. CFF-Tailings will be capped with 2 m of overburden. CFF-tailings will be deposited from a central series of spigots (line discharge) along the centre of the DDA, forming a triangular prism shaped stack. The line of deposition points start at the east wall and extends 850 m west and is 1370 m north of the southern wall (Figure A4.2). An observation point will coincide with the western end of the deposition line. An overburden dyke above grade is required to bound the tailings stack along the eastern wall.

DDA 3 (Centre Pit Area)

The total mined volume in DDA 3 is 732 Mm^3 with a surface area of 11.2 Mm^2. DDA 3 is bounded by the tailings deposit in DDA 2 to the east, dyke 3 to the north and Dyke 4 to the west. Deposition of CFF-tailings will continue the line deposition DDA 2 to infill the existing deposit in DDA 2 and extend into DDA 3. DDA 3 will be filled to a 26 m above grade. Dyke 4 will be constructed to 40 m using the same design as Dyke 2 to contain the tailings stack along the western edge. Tailings will also be capped with 2 m of overburden. The line of deposition points will extend 1350 m to 4950 m from the eastern wall and 1370 m north of the southern wall (Figure A4.2). An observation point will coincide with the western end of the deposition line.

DDA 4 (North Central Pit Area)

The DDA 4 are will be used for overburden storage, emergency process water storage, and emergency tailings storage. The total mined volume in DDA 4 is 474 Mm^3 and is bounded by dyke 3 to the south and pit walls. Dyke 3 will be constructed to the same elevation as dyke 1. Overburden deposited into DDA 4 will also provide separation from a neighboring mine site.

Model Decision Logic

For each time step, the model must determine where to direct the overburden material (dykes, in pit storage, or out of pit storage) and tailings (beach or CFF dewatering). The following is a chronology of decisions based on the model mine data set to determine where mine waste and tailings are deposited. The decisions assume the model will be starting at year 2013 and deposition will commence in DDA 1.

Overburden

At each time step, the model will check if the current DDA has sufficient freeboard (i.e. 3 m above the tailings and process water level). If sufficient freeboard is available, overburden will then be split between satisfying the in-pit and ex pit waste piles. If sufficient free board is not available, overburden will be used to increase height of containment dykes based on the dyke stage curve (height versus construction volume). Once a dyke has reached is maximum elevation, overburden will be directed to construct subsequent dykes.

EXTRACTION TAILINGS

For each time step, the model will first check if the current DDA has reached its maximum capacity (volume or elevation). If the DDA is not full, the model will then ensure sufficient freeboard is available before depositing the remaining tailings into the DDA. If there is insufficient room, tailings will be beached into the external tailings facility or DDA 4. Tailings deposition will cease in the current DDA once a maximum height has been achieved. Fine tailings runoff from beaching will be directed to the ETF. Fine tailings in the DDAs will be transferred to the ETF. All FFT accumulated in the ETF will be spiked into the whole tailings stream.

TAILINGS CONSOLIDATION PROPERTIES

Chapter 8 demonstrated the relationship between fines content of the tailings streams and the large strain consolidation properties. Based on fluctuations within the extraction process, ore body changes, and fines spiking, the fines content of the tailings streams will not be constant over the life of the mine, therefore the material properties will also change. To reduce the numerical complexity of having new tailings properties updated for every small change in fines content, a select sample of material properties will be chosen to represent a range of fines contents. For a typical CFF operation, the target fines content of the deposited tailings may fluctuate from ~ 21% to 14% or sand to fines ratio, SFR, from 3.8:1 to 6.4:1. Using the equations developed in Chapter 8, large strain consolidation parameters (A, B, C, D) were calculated for the minimum, maximum and average fines content. For a 5 year deposition scenario, the difference in elevation between the minimum and maximum fines content tailings was approximately 2.5 m. Therefore a 4% difference in fines content resulted in a 1.5 m elevation difference upon deposition. If the difference in fines content was reduced to only 2% between calculated material parameters, the elevation change was only 0.5 to 0.9 m. Therefore, when determining properties of tailings deposits, a minimum 2% change in fines content from the previous time step is required before tailings properties will be updated. The following table A4-XX summarizes the ranges of fines contents and associated tailings properties for tailings.

Table A4- 1. Tailings properties as a function of fines content.

Fines Content	SFR	Fines Content range	A	B	C	D
14	6.1	13-15	0.81	-0.067	4.6E-06	3.77
16	5.3	15-17	0.85	-0.088	3.3E-06	3.77
18	4.6	17-19	0.88	-0.11	2.6E-06	3.77
20	4.0	19-21	0.92	-0.12	2.2E-06	3.77
22	3.5	21-23	0.94	-0.14	1.9E-06	3.77

REFERENCES

Reeves, B. 1996. Environmental Impact Assessment for the Syncrude Canada Limited Aurora Mine. BOVAR Environmental, 1996.

Syncrude Canada Ltd. (Syncrude). 2012. 2012 Annual tailings plan submission Syncrude Aurora North (Leases 10, 12 and 34). 30 September 2012.