Fluid Fine Tailings Management Methods

An Analysis of Impacts on Mine Planning, Land, GHGs, Costs, Site Water Balances and Recycle Water Chloride Concentrations

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1. EXECUTIVE SUMMARY

1.1. Overview

Oil sands operators are currently developing and employing technologies to manage fluid fine tailings (FFT). Tailings planning is a fully integrated process involving mine planning, containment design, water management, progressive reclamation, and closure planning. Different FFT technologies have different implications to the overall site planning process. To understand the benefits and impacts of the different methods, COSIA requested Norwest to develop generic models to examine the relative impacts of different FFT management processes. The models were based on two ‘virtual mine’ scenarios selected to represent a range of conditions typical of the surface mine sites of the Athabasca oil sands region. Figure 1-1 shows a comparison of the selected virtual mine conditions with actual mine site conditions in the region.

The virtual mine conditions were:

- VM 1 – High grade, low fines, low waste to ore (waste:ore) ratio
- VM 3 – Lower grade, high fines, high waste:ore ratio

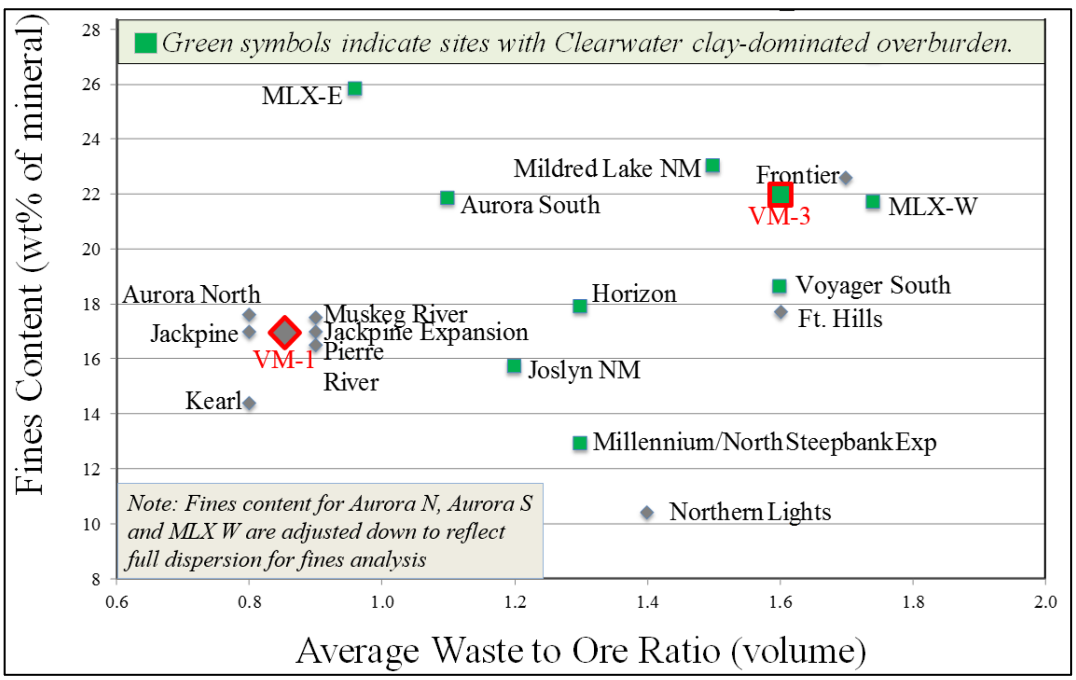

Figure 1-1. Mine Site Resource Properties

For virtual mine scenarios VM 1 and VM 3 different FFT management methods were individually applied to develop life-cycle costs and predict key environmental impacts. Table 1-1 shows the cases that were developed.

Table 1.1
Case Matrix Considered for the Study

Fines Management Technology	Site Conditions	
	VM 1	VM 3
Conventional Tailings	✓	✓
Composite Tailings (CT)	✓	
Centrifuged Fluid Tailings Stacked (CFTS)	✓	✓
In-Line Flocculation (ILF)	✓	✓
Fluid Tailings Thermal Drying (FTTD)	✓	

Further site condition details are shown in Table 1.2. The impacts examined for both cases include: net and total disturbed area, GHG generation, chloride concentration buildup in recycle water, and incremental costs over the conventional cases. The relative attributes of each are compared in the report body.

Table 1.2.
VM1 and VM3 Design Bases

Parameter	VM1	VM3
Production Schedule	40-year mine life ramping up to 200,000 bbls per day bitumen production after 4 years	
Ore Grade (wt% bitumen)	11.3%	10.7%
Fines Content (wt% mineral)	17%	22%
Ore Chloride content	Low case: 50 wt PPM of ore High case: 300 wt PPM of ore	Low case: 50 wt PPM of ore High case: 300 wt PPM of ore
Waste:Ore Ratio	0.85 with 24 m overburden thickness	1.6 with 54 m overburden thickness
Out of Pit Area	Highly constrained	Not constrained
Overburden Type and Ex-Pit Foundation	- No Clearwater - 6H:1V downstream slopes for tailings - 5H:1V slopes for dumps	- Dominant Clearwater - 20H:1V downstream slopes for tailings - 15H:1V slopes for dumps
In-Pit Foundation	- Poor (Basal clays) - 8H:1V downstream slopes for tailings - 10H:1V slopes for overburden	

1.2. Study Limitations

The virtual mine plan method is not a substitute for site-specific planning. It identifies relative impacts and trends that may be useful to mine planners in planning for their site-specific conditions. Simplified mine parameters were assumed to conduct the virtual mine case studies. Oil sand grade, fines content, overburden type, waste:ore ratio, ore chloride content, and base of ore foundation conditions remained constant through the life of mine. Under real site conditions, these parameters may vary significantly over the operating life of mine. Moreover, both cases employed a single fluid fine tailings (FFT) process method to best illustrate its impacts. The plans were developed assuming a new mine opening without legacy tailings or legacy pit development. Actual site conditions could favour some methods over others and the use of two or more processes over the mine life.

Given the differences between virtual mine and actual mine conditions, only site-specific plans can evaluate the constraints and opportunities against the actual process risk factors inherent in the different methods. The value of the study lies in the comparison of the relative attributes of the different methods. It also provides a method for comparing and contrasting new technologies.

1.3. Trends

Plans were developed for each case which included mass balance, volumes, water and salt balances, and planning parameters (slopes, setbacks, rates of rise, deposit densities) from pit opening through completion of closure activities.

Figure 1-2 shows a typical selection of plan figures for the VM 1 Centrifuge Case. Each method has its own set of figures provided in the main body of the report.

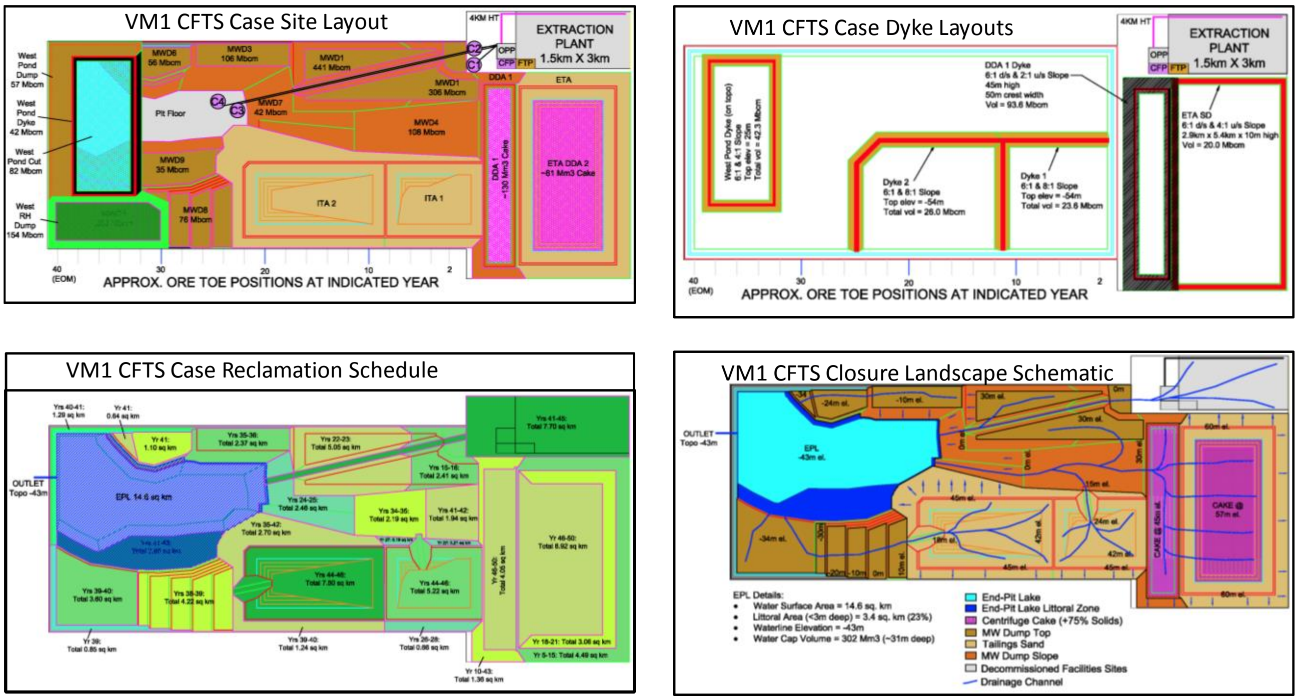

Figure 1-2. Typical Case Figures

Several important trends emerge from the analysis of each of the methods, summarized as follows.

Constrained sites with little resource-free area for placement of tailings, overburden and interburden are impacted in two ways:

- The out-of-pit tailings pond or overburden may need to be placed on mineable resource and subsequently relocated for mining access, incurring significant re-handling cost.
- Constrained space drives a need for early in-pit access to limit out-of-pit disposal. This results in in-pit deposits created at high capacity in short time windows. For in-pit treated deposits, this drives up CAPEX for FFT treatment and placement.

A pit shape that would facilitate early in-pit disposal, or availability of an adjacent previously mined-out pit, could counteract these cost impacts. This may be advantageous for the Conventional Case, CT Case and, particularly, the In-Line Flocculant Case.

Depending on site-specific conditions, **limiting FFT volume buildup could reduce costs for mining** haul distances, re-handle, and containment dyke construction. This is best illustrated by the lower mining costs for the thermal drying case which requires only a working volume of FFT and no large atmospheric drying area. However, thermal drying energy costs offset the mining cost savings. Even with an assumed 80% energy recovered for reuse, dryer GHGs overwhelm those from mining mobile equipment. Thermal drying could be an attractive component of FFT management if high energy recovery and reuse can be achieved without negating other energy efficiencies. Otherwise, the method is unattractive from an energy cost and GHG perspective. (See discussion in the following section.)

Chloride concentration buildup in the recycle water is a function of the chloride salts contained in the ore processed through bitumen extraction and the site water balance. The most intensive dewatering methods coupled with minimized water import and no water release during active mining, could more than double the concentration buildup in the recycle water system. With high chloride ore, the concentration could be problematic to ore processing and could also prolong the period for the end-pit lake water to attain low salt characteristics.

The CT process at 4:1 target SFR requires the largest in-pit containment volume. In the VM-1 case this incurs greater re-handling and dyke building costs. Unlike the other treatment cases, the CT Case did not process all FFT and relied on placement of untreated FFT in the end-pit lake. The VM-3 CT Case was not run. At 22% fines, a 4:1 SFR recipe, and 70% process utilization, an additional large capacity method would be needed to manage the fines. Cases with multiple treatment methods were not run.

Pit lake design is an important aspect of mine closure. Apart from habitat design features, two important characteristics must be managed in a site-specific context:

- The ratio of contributing drainage area to lake open water surface area is important to ensure lake level and fresh water are both maintained. Regional and site-specific drainage were beyond the scope of this study.
- Lake depth in relation to lake surface area are important considerations in avoiding the potential for meromictic behaviour in the lake, which is generally considered to be undesirable.

Reduction of the lake outlet elevation could be used to address or partially address both factors. Availability of sufficient backfill is also a factor in governing lake depth. How these factors can be employed to achieve desired lake characteristics, can only be addressed on a site-specific basis.

While the technologies have different containment requirements and ex-pit disposal volumes that affect the site terrain profiles and life-cycle water balance, operators have the planning flexibility and choice of FFT methods to meet the design requirements necessary for sustainable hydrology in the closure plan.

1.4. Cost Comparison

A comparison and breakdown of incremental costs relative to the Conventional Case is shown in Figure 1-3. This includes a sensitivity where only 50% of the thermal dryer heat is recovered for reuse in extraction.

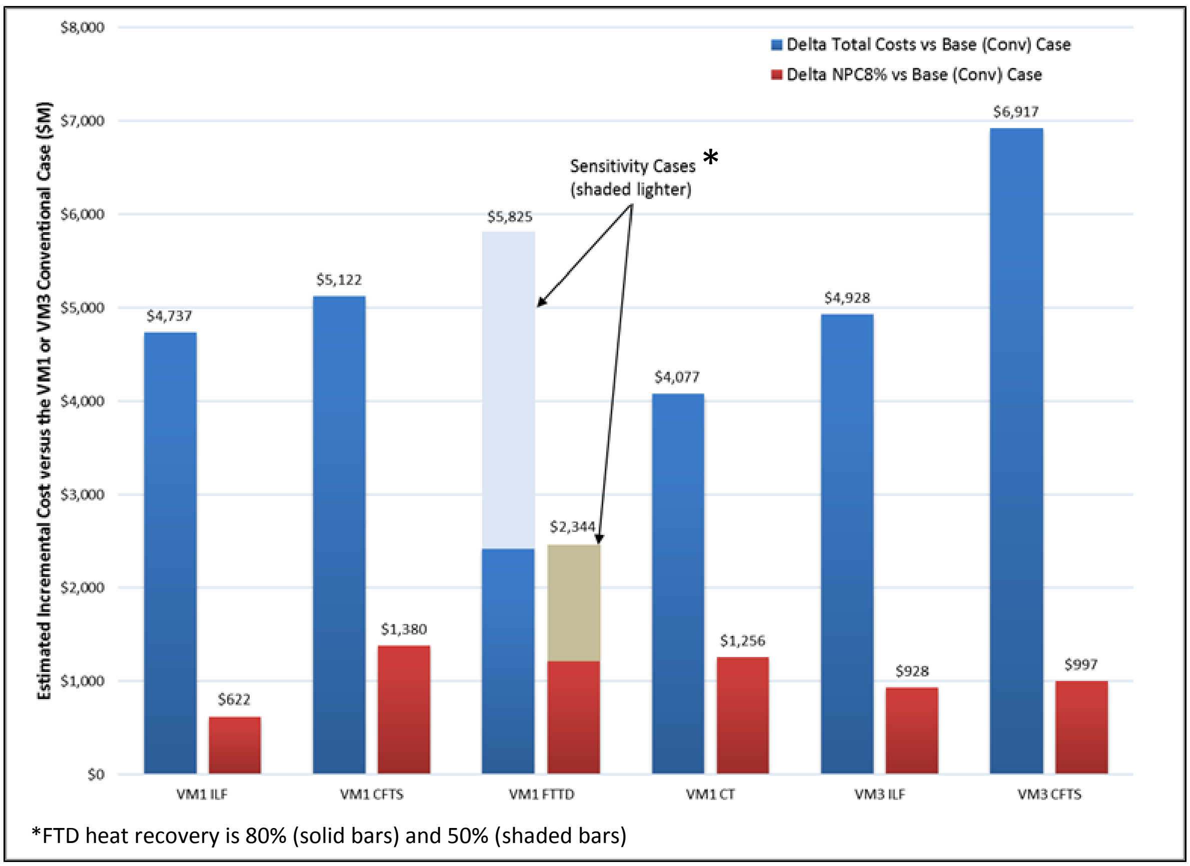

Figure 1-3. Cost Increment Relative to the Base (Conventional) case

All of the different methods are higher cost relative to the Base (Conventional Case). The In-Line Flocculation Case (ILF) has the lowest net present cost (NPC) increment over the Conventional Case for both VM1 and VM3 site conditions. It incurred high CAPEX due to a high capacity process operating over two short time periods and would be more competitive with early access to an in-pit disposal area. CT showed the lowest undiscounted cost increment over the Conventional Case. However, as noted, the CT Case did not treat all FFT and relied on placement of the balance in the end-pit lake. Thermal drying (FTTD) is competitive only if 80% of heat can be re-used without offsetting other integration efficiencies.

1.5. Risks and Limitations

All FFT management methods have related process risks and limitations. While an exhaustive listing of these is beyond the scope of this report, the following are highlights of the more important issues.

Sequestration of untreated FFT in pit, under a water cap is being demonstrated through the Syncrude Base Mine Lake demonstration project. The project will provide useful information on key issues:

- Naphthenic acid toxicity to fish is present in the recycle process water used for bitumen extraction and therefore in tailings water. The Base Mine Lake will provide information on the time required for natural biodegradation to reduce the concentration to acceptable levels. This information will be useful in assessing the progression of water quality where process water is used for capping and there is any upward seepage from the underlying FFT deposit. (The information will also be useful to all tailings management scenarios.)
- It will determine whether natural processes, such as the formation of a detrital layer, will provide a sufficiently robust interface at the 'mudline' to preclude re-suspension of fines in the overlying water during dimictic thermal turnovers of the water cover (behaviour common to temperate and sub-arctic lakes).
- It will provide insights on the management of bitumen (and diluted bitumen from froth treatment tailings) during deposition to avoid prolonged floating hydrocarbon on the lake surface.

The use of in-line treatment (flocculation) to dewater and increase density of the deposit, as in the In-Line Flocculation Case, is designed to both allow for more efficient use of in-pit void space and to reduce the above risks.

For the Conventional and In-Line Flocculation cases, long-term containment risks were addressed by placing all FFT, or geotechnically weak treated FFT, below grade in pit, to facilitate timely dam decommissioning and allow for mine closure. CT deposits were also contained in pit.

The dry stacking case assumed that centrifuge cake could be placed in maximum lift thickness of 2 metres, averaging 1.4 metres, and drying over a 17-month period to a geotechnical strength sufficient for free-standing deposits. It is recognized that this rate of drying is reliant upon attaining consistent cake density, a relatively uniform lift depth and having good surface drainage from the deposit. Moreover, freeze-thaw cycles and atmospheric drying effects are weather-dependent. Unfavourable weather conditions could impede productivity. These risks might be managed by providing for added time and area or by employing additional methods to manage all FFT.

The thermal drying case presents a different limitation. The process has been piloted for COSIA by BEPEX but unlike the other tailings treatment technologies it has not been implemented at commercial scale. Economics and energy efficiency for method feasibility requires recovery of 80% of the thermal-drying heat. Even if engineering solutions can achieve energy recovery, energy reuse is not a net saving if it offsets other energy efficiencies. Currently, all sites practice energy recovery from on-site processing and employ co-generation to supply both site electrical load and process heat. In this context, it is difficult to see how thermal drying can represent a useful base-load process. It could have some use in niche applications (e.g., to meet winter peak energy demand).

The use of CT has the benefit of providing a basis for terrestrial surface reclamation over the treated tailings deposit. Hydraulically placed sand capping is a proven method in the oil sands. CT requires careful attention to treatment protocols, slurry composition and deposition to avoid segregation of fines from the deposit. The sand required to produce the target SFR of 4:1 represents a planning constraint due to the high-volume containment requirement for deposition and the competing use of sand needed for construction. These factors have limited the use of CT on some sites.

1.6. Fines, Clays, and Fluid Tailings

This study basis assumed that fluid tailings and fluid tailings volumes are associated with the mineral fines component of the oil sands ore processed - defined as particles with diameter less than 44 micrometres. (The former AER Directive 74 also focussed on the mass of fines processed.) However, it is the clay component that generates the volume of FFT with some clay minerals disproportionately due to their high water retention properties.

Clay particles represent a fraction of the total fines mass in oil sand. While the total fines content is indicative of clay content in oil sand ore; sand, fines, and clay distribute differently in the various types of tailings deposits. Higher solids contents in the more-aged, deeper zones of tailings settling ponds are partially a function of sand and silt particles sinking into the MFT. As industry knowledge deepens in this area, more focus can be applied to the clay content of the source ore and its distribution in settling pond FFT and treated tailings deposits.

1.7. Conclusions

1. All cases involving active treatment incur more costs during the operating phase than the Conventional Case where all FFT is sequestered, untreated, in pit lakes. With the Base Mine Lake demonstration underway, questions remain as to how long the lake will take to stabilize to an acceptable state and whether this duration is suitable for end-pit solutions and desired closure timing.

2. The use of In-Line Flocculation (ILF) Case treatment to densify the FFT prior to placement in pit with water capping makes better use of pit void space and reduces some of the risks associated with untreated FFT water capping.

3. Limited ex-pit space and lack of available early in-pit void, exacerbate costs for the conventional, ILF and CT cases. Site-specific conditions, or the availability of legacy pit space, could mitigate re-handling costs and reduce unit CAPEX for the treatment cases.

4. Thermal drying of FFT illustrates the benefits of reduced FFT inventory to mining and containment construction costs. However, the high thermal fuel requirement suggests that its use may only have application as a small-scale component of FFT management, such that the energy can be recovered and re-used without negating other energy integration efficiencies.

5. Basing FFT volumes on < 44µm fines can distort some predictions and measures of performance. Incorporation of clay content into deposit planning and measurement is a desirable improvement to the practice.

6. Efforts by industry continue toward improving process performance, reducing uncertainties, and improving economics of all methods. A technology breakthrough could improve the relative attractiveness of any of the methods.

7. Long term containment of process water would be aided by the ability to release water during the active mine life. Adoption of criteria for release would allow operators to minimize containment and achieve functional pit lakes in the closure landscape in less time.

8. Overall, the study emphasizes the necessity of life-cycle planning to determine the costs, opportunities, and risks for different tailings methods. The generic modelling process can assist but is not a substitute for site-specific planning.

2. INTRODUCTION

Fluid fine tailings (FFT) management technologies currently used, or being considered for use, in the oil sands mining industry have been developed to varying stages of readiness for commercial implementation over the past two decades. The relatively rapid deployment of FFT management technologies during this period is a result of ongoing industry research and increased regulatory focus on tailings management.

Tailings management processes have a wide range of direct costs which tend to be highly visible as they relate to coarse tailings transport and placement, and fluid tailings containment, treatment, and placement in disposal deposits. However, tailings management practices can also result in significant indirect costs for mining operations, reclamation, and closure. Those indirect costs are less visible and more difficult to quantify as they relate to mining and transportation costs impacted by the site layout and realized over the life-cycle of the mine, including closure activities after the cessation of mining. Additionally, indirect costs attributable to tailings management practices vary significantly, from site to site, due to differences in such factors as pit geometry, availability of ex-pit space for tailings and mine waste, timing of in-pit space availability, distribution of geotechnically weak overburden and pit basal materials and the mine development area setting within the regional drainage system.

Different tailings management approaches can also result in differences in land disturbance, reclamation schedules, water use, energy efficiency, air emissions, and closure landscape features. A tailings management approach that results in good performance on operating or capital cost, or a single environmental effect, may result in poorer performance than alternative approaches when all environmental impacts and closure costs are considered. The magnitude of the differences between outcomes achieved using different tailings management approaches can also vary significantly for different site-specific conditions. Therefore, at each mine site the trade-offs between economics, environmental impacts and risk profile must be considered to determine the most desirable approach. This fact is recognized by the Government of Alberta (GOA) in its Tailings Management Framework (TMF) that was released in March 2015.

The purpose of this Study is to estimate, at a high level, how selected Fluid Fine Tailings (FFT) management technologies could affect mining and tailings capital and operating costs (CapEx and OpEx), reclamation and closure costs and environmental impacts. Environmental impacts are quantified in terms of total land disturbance, reclamation schedules, mine fleet greenhouse gas (GHG) emissions, surface water and groundwater stored in the closure landscape, and closure landscape design including the characteristics of pit lakes required. High-level annual water balances and chloride tracking are provided for each of the tailings management cases, to quantify the volume of water, mass of chloride and chloride concentrations that remain in tailings pore spaces and the "free" water during operations and at closure. Chloride from the salt in ore remains in solution and therefore concentrates proportionately with reduced water import to the process recycle water.

3. STUDY METHODOLOGY

This section outlines the methodology for the case plans, economic analysis, and water/chloride balance work for the various tailings management approaches.

3.1. Cost Estimate Approach

The Study was comprised of conceptual life of mine (LOM) mine and tailings plans developed for two virtual mine sites (VM1 and VM3) employing five tailings technologies. To limit the number of options to a manageable level, all five tailings technologies were applied to the VM1 site but only three to the VM3 site for a total of eight cases as shown in Table 3.1. CT would not handle all FFT in the high fines VM3 context. Cases with more than one treatment were not run. The thermal drying impacts on mining costs are best illustrated in a space-constrained context. Therefore, these methods were excluded from the VM3 cases.

Table 3.1.
Case Matrix Considered for the Study

Fines Management Technology	Site Conditions	
	VM 1	VM 3
Conventional Tailings	✓	✓
Composite Tailings (CT)	✓	-
Centrifuged Fluid Tailings Stacked (CFTS)	✓	✓
In-Line Flocculation (ILF)	✓	✓
Fluid Tailings Thermal Drying (FTTD)	✓	-

Cost estimates for mining and tailings operations as practiced in oil sands mines were developed on an annual basis to estimate as-spent and net present costs (NPC). The plans, and therefore the cost estimates based on the plans, were completed at a conceptual level that Norwest would describe as 'pre-scoping' level. While the costs for each case are comparable to each other for identification of trends, cost estimates developed for this Study may not be directly comparable to cost estimates for site-specific conditions based on more detailed engineering work such as prefeasibility studies (often done to provide a basis for regulatory applications), or feasibility studies (often done to provide a basis for project sanction decisions). The conceptual mine and tailings plans for each case include:

- Selection of locations for the extraction plant, ore preparation plants (OPPs), ore crushers, and ore conveyors;

- Approximate mine face advance schedule based on plan view areas (i.e., no detailed mine scheduling);
- Ore, mine waste, and tailings production profiles;
- FFT inventory projections based on a common settlement/consolidation schedule;
- Conceptual designs for dykes, dumps, and tailings disposal areas based on 3-dimensional (3D) designs in AutoCAD with volumes estimated using AutoCad/Quicksurf;
- Annual mine waste material balances and tailings;
- Conceptual closure drainage plans;
- Approximate ore and waste haul distances;
- Mine fleet net operating hours for shovels, trucks, and support equipment;
- Conceptual tailings line layouts;
- Disturbance and reclamation schedules;
- Approximate mass transport requirements through tailings lines; and
- High-level cost estimates for design features and activities that differ between the cases.

The focus of the plans developed for each case was to achieve the following objectives in order of priority:

1. Create workable mine waste and tailings disposal plans
2. Create closure landscapes with acceptable levels of geotechnical risk
3. Minimize operating and capital costs
4. Limit the size of end-pit lakes (EPLs).

The total EPL surface area and water depths shown for some cases may not be sustainable due to the ratio of EPL surface areas to the limited drainage area within the project development areas. However, it was not possible to assume the site conditions with respect to the setting within the drainage basin beyond the development area. Contributing drainage areas vary significantly for actual mine sites in the region. Therefore, no constraints were established with respect to EPL surface area or depth to limit planning options. In several cases the closure plan would require greater drainage area than shown within the project development area to sustain the EPL systems. Surface drainage input to pit lakes is a site-specific design consideration.

3.2. Virtual Mine Site Descriptions

Three important site-specific considerations that affect the total cost and environmental impacts of an FFT management technology are waste:ore ratio, ore fines content, and distribution of geotechnically weak materials (mainly Clearwater clay overburden and in-pit basal clays) across the site. These conditions vary significantly between oil sands mine sites and the selection of a single virtual mine site condition that adequately represents all site conditions for the purposes of the Study is not possible.

Figure 3-1 shows the approximate average waste:ore ratio, average ore fines content and presence of Clearwater overburden for the operating and future mine sites as stated in publicly available documents. The two virtual mine sites used for this Study, VM1 and VM3, are also shown on Figure 3-1 to illustrate their relationships to actual mine sites. The most significant difference between sites, in terms of distribution of geotechnically weak materials, is the presence or absence of widespread Clearwater Clay Formation (Kc), a weak foundation unit that necessitates more shallow slopes for geotechnical designs for pit walls, mine waste dumps, and containment dykes. Sites that have widespread Kc are shown with green square symbols on the figure while those without widespread Kc are shown as grey diamond symbols. Note that for Syncrude's Aurora North, Aurora South, and MLX-W pits, the fines analysis method is not directly comparable to those used for other sites. Therefore, an adjusted fines value, assumed to be 80% of the published value, is shown for those Syncrude sites. The VM1 site conditions were selected to represent the sites clustered relatively tightly in the lower left portion of the figure. Similarly, VM3 site conditions were selected to represent the looser cluster of sites shown in the upper right portion of the figure.

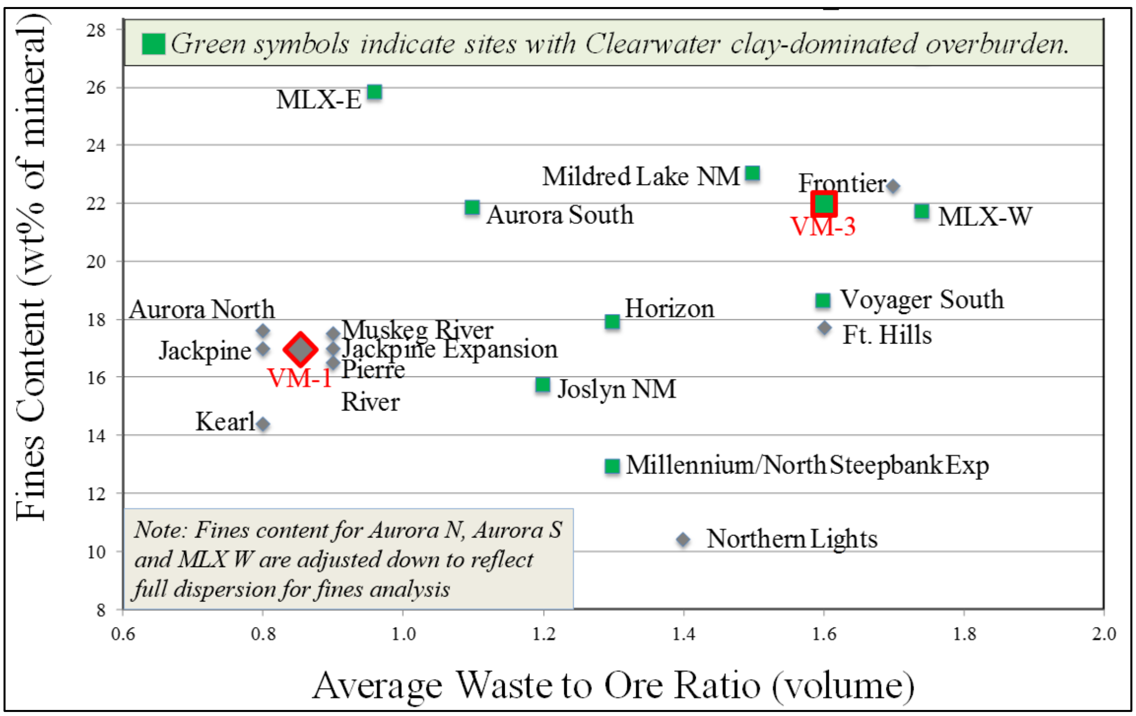

Figure 3-1. Comparison of Virtual Mine Site Characteristics to Oil Sands Mining Projects

To determine appropriate site conditions for VM1 and VM3, Norwest reviewed the generalized characteristics of the oil sands surface mine sites. The characteristics of the VM1 and VM3 sites shown on Figure 3-1 were then selected to represent a range of site conditions for the Study. A simple rectangular pit shape with constant ore, overburden, and interburden thicknesses across the pit was chosen to simplify the planning work and to enable better identification of effects caused by changes to tailings technology. Actual mine sites can have quite irregular pit shapes and highly variable overburden and interburden to ore ratios, with overburden usually trending higher to the end of mine life.

The VM1 pit area covers 60 km^2 and the available ex-pit area for facilities covers an additional 30 km^2. The pit is approximately 11 km from east to west and 5.4 km from north to south. Due to lower ore grade, increased overburden thickness and shallower overburden slopes, the VM3 pit area increased to 72 km^2 and the available ex-pit area for facilities covers an additional 92 km^2. The VM3 pit is approximately 12.1 km from east to west and 6 km from north to south. (Note that the figures are of different scale.) Approximate ore toe locations are shown on a scale below the pits on Figure 3-2 and Figure 3-3 to represent the approximate location of the ore toe if the mines were advanced from east to west across the full north-south width of the pit. Pit advances proceed in a more complex manner to enable timely development of in-pit mine waste and tailings disposal areas. Those refinements were incorporated in the VM1 and VM3 cases that were evaluated. However, the scale shown on the figures provides a useful reference to assist the reader in understanding scheduling issues for the VM1 and VM3 cases.

The ex-pit area dimensions for VM1 are 4.3 km and 7 km in the east-west and north-south directions, respectively, and the size of the ex-pit area was chosen to reflect the highly space-constrained nature of most VM1-type mine sites (with an ex-pit to pit area ratio of 0.5). The ex-pit area dimensions for VM3 are 11.5 km in both the east-west and north-south directions. The size of the ex-pit area was chosen to reflect the space-abundant nature of most of the VM3-type mine sites, with an ex-pit to pit area ratio of 1.3 compared to 0.5 for VM1. The ex-pit area is L-shaped as shown on Figure 3-3 with a smaller portion of the area located east of the pit and a larger portion located to the north of the pit. Table 3.2 provides the mine design basis for the VM1 and VM3 cases. The design basis for tailings streams and deposits vary with the technology selected.

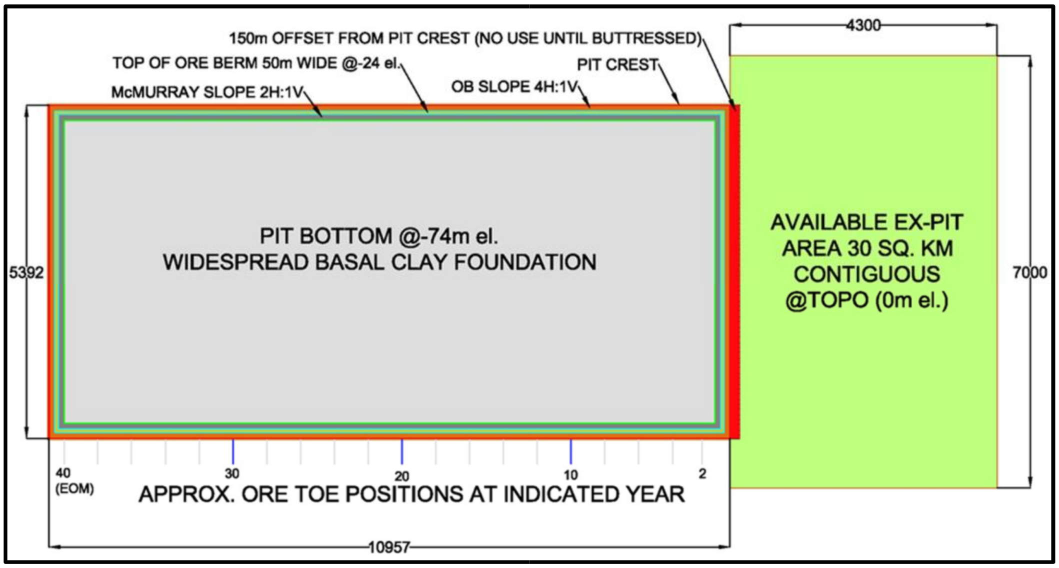

Figure 3-2. VM1 Site

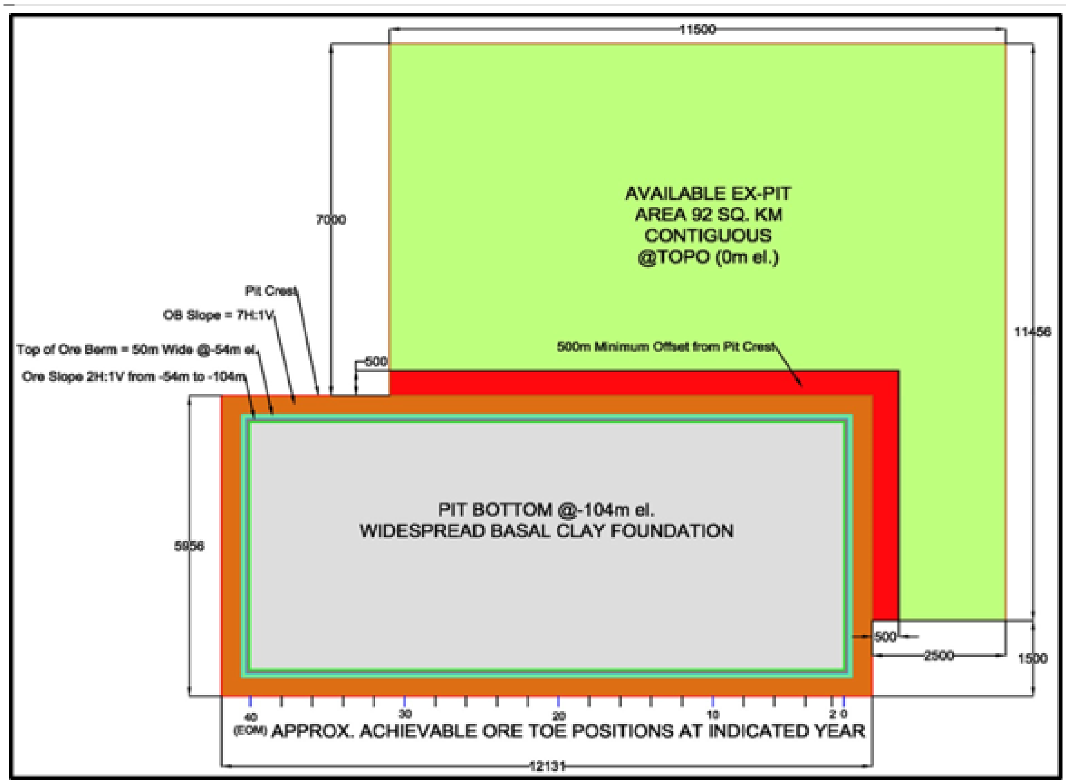

Figure 3-3. VM3 Site

Table 3.2.
VM1 and VM3 Design Bases

Parameter	VM1	VM3
Production Schedule	40-year mine life ramping up to 200,000 bbls per day of bitumen after 4 years	
Ore Grade (wt% bitumen)	11.3%	10.7%
Fines Content (wt% mineral)	17%	22%
Ore Chloride Content	Low case: 50 wt PPM of ore High case: 300 wt PPM of ore	Low case: 50 wt PPM of ore High case: 300 wt PPM of ore
Waste:Ore Ratio	0.85 with 24 m overburden thickness	1.6 with 54 m overburden thickness
Out of Pit Area	Highly constrained	Not constrained
Overburden Type and Ex-Pit Foundation	- No Clearwater - 6H:1V downstream slopes for tailings - 5H:1V slopes for dumps	- Dominant Clearwater - 20H:1V downstream slopes for tailings - 15H:1V slopes for dumps
In-Pit Foundation	- Poor (Basal clays) - 8H:1V downstream slopes for tailings - 10H:1V slopes for overburden	

3.3. Virtual Mine Compared with Actual Site Conditions

Table 3.3 provides a comparison of the virtual mine site conditions compared with real conditions in the region. It is important to remember that the virtual mine plans do not reflect actual mine plans but have been defined to illustrate the differences between fines treatment methods.

Table 3.3.
Virtual Mine assumptions versus real site conditions

	VM-1	Actual Mines	VM 3	Actual Mines
Waste:Ore[1]	Constant waste:ore ratio of 0.85	Similar average waste:ore ratios, but ratios vary through mine life, generally trending up.	Constant waste:ore ratio of 1.6	Similar average waste:ore ratios, but ratios vary through mine life, generally trending up.
Grade	Constant grade of 11.3 wt% bitumen	Similar average, but grades vary through mining sequence.	Constant grade of 10.7 wt% bitumen	Similar average, but grades vary through mining sequence.
Fines content	17 wt% of mineral (Constant through life of mine)	Similar average fines, but contents vary through mining sequence.	22 wt% of mineral (Constant through life of mine)	Similar average fines, but contents vary through mining sequence.
Base of ore foundation conditions	Consistently poor requiring shallow-sloped in-pit dykes	May have good foundation areas close to base of mine.	Consistently poor requiring shallow-sloped in-pit dykes	May have good foundation areas close to base of mine.
Clearwater clay overburden	Not dominant	Not dominant	Dominant - limits slope of out-of-pit structures.	May or may not be dominant
Proximate area devoid of mineable resource for out-of-pit disposal.	Highly constrained. –re-handling of tailings or waste generally required. ExPit:Pit area = 0.5	Generally, highly constrained, but prior mine pits may be available for in-pit deposits.	Unconstrained -Large ex-pit area available. ExPit:Pit area = 1.27	Large ex-pit areas generally available.
Mineable ore pit outline	Rectangular 5,392 m x 10,957 m	Pit shape may be more favourable to early in-pit disposal.	Rectangular 5,956 m x 12,131 m	Pit shape may be more favourable to early in-pit disposal.
Chloride content in ore	High case: 300 wt ppm Low case: 50 wt ppm (Constant through life of mine)	Chloride content varies regionally and within leases.	High case: 300 wt ppm Low case: 50 wt ppm (Constant through life of mine)	Chloride content varies regionally and within leases.
FFT Methods Employed	Each case employs a single process.	Operators may employ more than one FFT process.	Each case employs a single process.	Operators may employ more than one FFT process.
Legacy conditions	No legacy	Legacy FFT volumes and pit space	No legacy	Legacy FFT volumes and pit space

[1] Waste represents the total of overburden and interburden (sometimes called centre reject), consisting of low-grade oil sand, siltstone, or clay-rich strata. Ore is the oil sand delivered to extraction.

3.4. Fluid Fine Tailings Composition and Density Prediction

The tailings technologies evaluated for the Study are differentiated primarily by how they affect management of FFT inventories. Therefore, the prediction of FFT inventories is critical to unbiased evaluation of the technologies. In Norwest's experience, no common method is used by the oil sands industry to predict FFT inventories and there is significant variation between the approaches used by the operators. For the purposes of this Study, Norwest developed an FFT inventory prediction model based on the following assumptions:

- 45% of ore fines were released to form FFT annually (except CT Case for which the assumed fines release rate was modified to account for on-line tailings processing).
- 55% of fines were captured in sand beach-above-water/beach-below-water deposits.
- FFT volume was predicted based on fines dry density (i.e., the mass of dry fines per cubic metre of FFT).
- The FFT began to accumulate sand after four years and continued to accumulate sand for the LOM due to a "sand-raining" effect. To be valid this requires that coarse tailings be actively deposited in the FFT storage area so that suspended fine sand particles are available to settle into aged FFT.
- FFT stratifies by age (and therefore density). The inventory of FFT and contained fines and sand was tracked for FFT generated in each year of operation. FFT consumption was manually scheduled for each case by selecting the age of FFT to be fed to engineered tailings processes.

Figure 3-4 shows the profile of total solids percentage and the dry density of the fines and sand components used as a planning basis for the different FFT management techniques. This model does not consider the clay:fines and clay:water ratios, which are more fundamental parameters for MFT accumulation, composition, and volume projections.

In all cases, tailings streams from the extraction and froth treatment plants including primary separation vessel (or cell) underflow (PSV u/f), flotation tailings, and froth treatment tailings (paraffinic froth treatment assumed) are beach deposited to an ex-pit tailings pond. Beach runoff during hydraulic tailings deposition settles to form mature fine tailings (MFT) reaching a concentration of 30%wt solids (predominantly fines) during the fourth year of deposition and 45%wt solids after 20 years with inclusion of sand as shown in Figure 3-4.

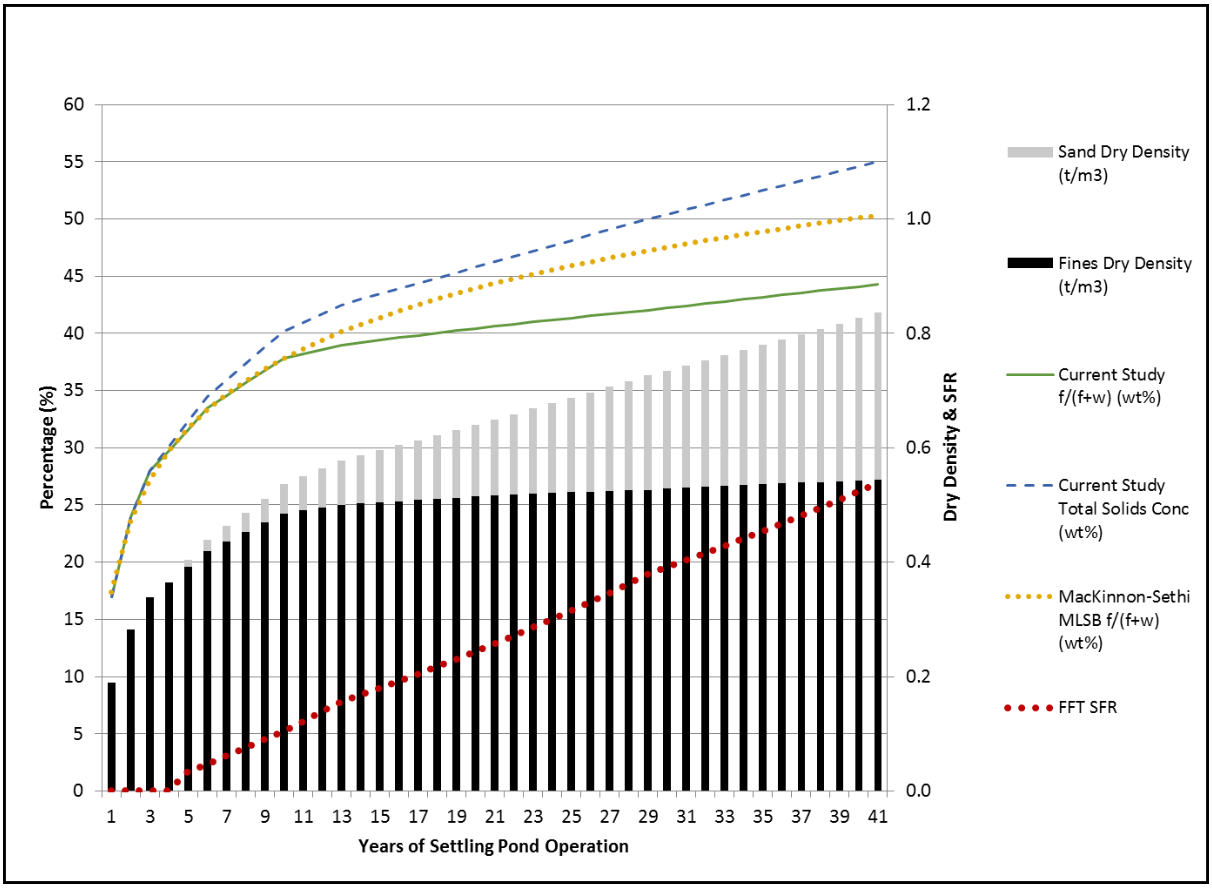

Figure 3-4. Fluid Fine Tailings Density and Composition with Time

3.5. Tailings Production Schedule

Mine production schedules for the Study were based on recovery of 200,000 barrels of whole bitumen per calendar day. Based on that production rate and the orebody quality parameters shown in Table 3.2, a mine production schedule was developed for the cases. Using the mine production schedules as a basis, tailings production schedules were developed using the tailings production modeling assumptions shown in Table 3.4.

Table 3.4.
Tailings Planning Design Basis

Parameter / Characteristic	Value / Condition
Froth treatment tails deposit (FTT) production	0.02 m^3 per tonne of ore
FTT deposit solids concentration	60 wt%
FTT deposit SFR	0.58
Flotation tails deposit production	0.04 m^3 per tonne of ore
Mass of sand in flotation tails deposits	8.04% of sand in ore
Mass of fines in flotation tails deposits	2.78% of fines in ore
Primary Separation Vessel underflow (PSV u/f) deposit sand and fines mass	All sand and fines not captured in ore rejects, FTT deposits, flotation tails deposits, or FFT (Note: Solids in bitumen product were considered insignificant and were ignored.)
PSV u/f deposit dry density	1.53 t/m^3
Ore rejects mass	3% of ore
Ore rejects mineral content	88 wt%
Ore rejects SFR	3:01
Ore rejects deposit density	2.1 t/m^3
Target peak FFT volume	No limit
Recycle water inventory management strategy	Recycle water volume controlled to the minimum required to provide a 3-m depth of clarified water over the FFT “mudline” in any active recycle water ponds.
ILF deposit initial solids concentration	50 wt%
ILF deposit near term consolidation	Negligible within the planning timeframe
ILF deposit long-term consolidation	54% reduction in volume (to 80% solids) over centuries
CFTS cake deposit initial solids concentration	55 wt%
CFTS deposit consolidation	Deposit solids concentrations: Year 1 = 55%, Year 2 = 65%, Year 3 = 70%, Year 4 = 75%; no further consolidation until after closure (100% saturation assumed). Post-closure: 11.7% reduction in volume (to 80% solids) over decades/centuries.
Percentage of PSV u/f to cyclones during CT production	70% (35% in start-up and shutdown years)
Percent of cyclone feed sand to CT	94.30%
CT SFR	4:01
CT deposit dry density	1.59
Solids concentration of FTTD fines product	85 wt% (fully saturated)

3.6. Costing Assumptions

Estimation of costs was completed for components that differ between the cases and would contribute significantly to total costs. Cost estimates were completed for mine fleets (shovels, trucks, and support equipment) using Norwest's in-house Xeras-based costing model. Costs estimated for other activities were based on unit rates compiled from discussions with COSIA mining members and Norwest's experience. The unit rates used are shown in Table 3.5, Table 3.6, and Table 3.7 for Mining, Tailings, and Reclamation and other cost categories respectively. Each industry member may have its own perspective on unit rates that they may wish to apply as part of sensitivity analysis when considering the results of the Study.

Additionally, the plans were completed at a very high level and the unit rates selected were based on an amalgam of past estimates and reported costs for a variety of mine sites with varying site conditions. Therefore, the accuracy of the cost estimates is not considered to be adequate to enable direct comparison to cost estimates that may have been developed for specific mine sites based on more detailed engineering studies, using site-specific factors. The costs are considered adequate to identify trends indicating relative performance of the technologies evaluated for the assumed mine-site conditions.

Table 3.5.
Mining Cost Assumptions

Cost Item	Conv Case	ILF Case	CFTS Case	FTTD Case	CT Case
Contract mining	All pre-production mine waste movement was assumed to be by contract mining. Costs were determined in Norwest's XERAS costing model.				
Shovels	100% electric cable shovels (e.g., P&H 4100 C). CapEx and OpEx determined using XERAS model.				50% electric cable shovels (e.g., P&H 4100 C) & 50% hydraulic shovels (e.g., Cat 6090 FS) to reflect greater selectivity requirements for increase mine waste utilization for dyke construction. CapEx and OpEx determined using XERAS model.
Trucks	CapEx and OpEx determined using XERAS model based on estimated total haulage requirements (Mbcm*kms).				
Support equipment	CapEx and OpEx determined using XERAS model based on shovel and truck net operating hours.				
Crushers and conveyors	Crusher installation CapEx = $250M each Conveyor CapEx = 18.5 $M/km Conveyor OpEx = $0.04/t*km				
Mine waste dyke construction	Sand filter volume = 3% of dyke volume Sand filter cost = $30/bcm Average dyke fill placement = $0.50/bcm				
Dump trafficability	$0.05/bcm				$0.20/bcm (higher due to high allocation of mine waste to dyke construction)

Table 3.6.
Tailings Cost Assumptions

Cost Item	Conv Case	ILF Case	CFTS Case	FTTD Case	CT Case
PSV u/f	Pipe installation CapEx = $4M/km Pump CapEx = $5M/km OpEx = $0.06/dry tonne*km (includes allowance for pipe turning and replacement)				
Flotation tails	Pipe installation CapEx = $2M/km Pump CapEx = $2M/km OpEx = $0.10/dry tonne* km (includes allowance for pipe turning and replacement)				
FTT	Pipe installation CapEx = $2M/km Pump CapEx = $2M/km OpEx = $0.10/dry tonne* km (includes allowance for pipe turning and replacement)				
Cyclone o/f	n/a				Pipe installation CapEx = $2M/km Pump CapEx = $2M/km OpEx = $0.10/dry tonne* km (includes allowance for pipe turning and replacement)
FFT reclaim and transfers	FFT cutter head suction dredge (from ETA to Plant) CapEx = $50M each FFT cutter head suction dredge OpEx + Sustaining CapEx = 3% of initial CapEx annually) FFT transfer barge (for thin fine tails & water) CapEx = $10M each initially + 3% sustaining CapEx annually FFT transfer line + pump CapEx = $3M/km initial + 3% of installed pipe length replaced annually. FFT transfer line OpEx = $20k/Year fixed per installed line				

Table 3.6 (cont'd) - Tailings Cost Assumptions

Cost Item	Conv Case	ILF Case	CFTS Case	FTTD Case	CT Case
FFT treatment process	n.a.	Initial CapEx = $68/dry tonne of fines in FFT feed/Year (40% of CFT Plant cost) Sustaining CapEx = 8% of installed CapEx per year. OpEx: Floc dosage 1,000 g/t of fines in FFT feed Floc cost = $4.50/kg Gypsum cost = $0.5/t of fines Labor = 48 workers x 4 shifts x 273 days/Year (213 days operating) x $160k annually) (set for 28.4 Mt fines/Year and adjusted to reflect annual fines processing rates)	Initial CapEx = $170/dry tonne of fines in FFT feed/Year Sustaining CapEx = 3% of installed CapEx per year. OpEx: Floc dosage 1,000g/t of fines in FFT feed Floc cost = $4.50/kg Gypsum cost = $0.5/t of fines Labor = 16 workers x 4 shifts x 365 days/Year x $160K/Year each (set for 9.5 Mt/Year of fines/Year and scaled to annual rates for the VM1 Case; set for up to 18 Mt/Year of fines for the VM3 Case with no scaling)	Initial CapEx = $180/dry tonne of fines in FFT feed/Year Sustaining CapEx = 3% of installed CapEx per year. OpEx: Floc dosage 1,000g/t of fines in FFT feed Floc cost = $4.50/kg Gypsum cost = $0.5/t of fines Labor = 20 workers x 4 shifts x 365 days/Year x $160K/Year each Natural gas consumed = 43.15 MBTU/dry tonne of fines in cake Natural gas cost = $3/MBTU Heat recovery efficiency = 82% Cake haulage load factor = 70% (accounted for in mining costs) Labor = 20 workers x 4 shifts x 273 days/Year (213 days operating) x $160k annually) (set for 6.7 Mt/Year of fines and scaled to reflect annual fines processing rates)	CT Plant initial CapEx = $900M CT OpEx + Sustaining CapEx = $4.5M per tonne of fines in CT
Sand dyke construction	$0.40/m^3 of sand cell construction.				

Table 3.7.
Reclamation and Other Cost Assumptions

Cost Item	Conv Case	ILF Case	CFTS Case	FTTD Case	CT Case
Placement of reclamation soils	$5/bcm (includes final landform grading and soil placement as an incremental cost to normal material movement)				
Stockpiling reclamation soils	$1/bcm (incremental cost to normal material movement)				
Use or re-handle of reclamation stockpile	$4/bcm				
Mine waste or tailings sand re-handle post mining	$3.50/bcm (primarily for dyke breaches)				
Shoreline armoring	$300/m3 of rock placed				
Greenhouse gas offsets	Diesel fuel consumption estimated using Norwest's Xeras costing model from estimated equipment hours. GHG offset costs = $30/t CO2e				

3.7. Water and Chloride Balance Approach

To develop high level annual water and chloride balances, Norwest developed an approach to accurately quantify the volume of water and mass of chloride that remain in tailings pore spaces and "free" (surface) water during operations and at closure. The six major components of the water balance are connate water, river water, precipitation, evaporation, groundwater removed for mining, and the mine site water inventory. The mine site water inventory is a combination of the "free" surface water in the system and the water locked up in tailings pore spaces. The six components are related in the following equation:

Δ ("Free" [Surface] Water + Tailings Pore Water) = Connate Water + River Water + Precipitation + Groundwater – Evaporation

For each scenario, the volumes of tailings (PSV u/f, flotation tailings, and froth treatment tailings) and FFT generated each year were taken from the tailings work described in the previous section. Water contents for each tailings stream were used to calculate the water locked up in tailings pore spaces every year. The water content for coarse tailings was assumed to be constant throughout the years, but the water content for FFT was a dynamically changing value based on consolidation and sand inclusion. The amount of surface water required each year to maintain a 3-m cap on fluid tailings settling areas was derived based on the calculated inputs and outputs of precipitation, evaporation, connate water, and groundwater. The groundwater calculations were based on a specified saturated thickness of overburden that required dewatering, and a specified saturated thickness of basal watersands that required depressurization. Once these inputs and outputs were calculated for each year, the amount of river water required, if any, was calculated to complete the water balance.

Once the conceptual water balance was constructed, the chloride component was added. The chloride balance assumed a certain mass percentage of chloride in the ore (50wtppm for the low chloride cases and 300 wt ppm for the high chloride cases). Precipitation, river water, and overburden water were assumed to contain a negligible chloride concentration, and the basal water sands chloride concentration was assigned based on Norwest's experience in the Athabasca Oil Sands.

Ground water volumes and basal salt concentrations vary across the region, but their relative contribution is small in comparison to the other water and chloride contributions – particularly the ore chloride content.

Table 3.8 provides the main assumptions used to generate the water and chloride balances.

Table 3.8.
Assumptions Used in the Water and Chloride Balance

Parameters	Units	Value
Precipitation	mm/year	418
Evaporation	mm/year	1,000
Runoff/Infiltration Ratio	NA	0.89
Saturated Overburden Thickness	m	15
Overburden Specific Yield	%	17.5
Saturated Basal Sands Thickness	m	14
Basal Sands Available Head	m	20
Basal Sands Specific Storage	m^{-1}	3.30E-06
Connate Water Content	% wt.	5.5
Low Range Ore Chloride Content	ppm	50
High Range Ore Chloride Content	ppm	300
Coarse Tailings Water Content	% wt.	25
FFT Water Content	% wt.	Variable
Clean Water Cap	m	3

4. FLUID FINE TAILINGS (FFT) MANAGEMENT TECHNOLOGY CASES

As noted in Section 3, the tailings technologies evaluated for the Study are differentiated primarily by how they affect management of FFT inventories. Each of the FFT management cases use the profile of FFT composition and densification profile illustrated in Section 3. For each of the cases, MFT accumulates for different periods prior to transfer and processing, depending on the available timing of the disposal areas.

Each technology was assumed to perform as described in the respective design basis descriptions without regard to varying perspectives on their expected performance or technical risk. The fines management technology cases evaluated are described below.

Figures that illustrate site progression to closure for each case are presented at the end of this section.

4.1. Conventional Tailings

Accumulated MFT are transferred to two mined-out pit areas (a mid-pit lake and an end-pit lake) and water capped to form lakes below original topography for closure. Figure 4-1 shows the site layout, in-pit dykes, reclamation schedule and closure landscape for the VM1 Conventional Case.

Figure 4-2 shows the site layout, in-pit dykes, reclamation schedule and closure landscape for the VM3 Conventional Case.

4.2. Composite Tailings (CT)

The PSV u/f stream is cycloned and MFT and gypsum are added to the cyclone underflow to form non-segregating slurry at an average sand-to-fines ratio (SFR) of 4:1. The CT slurry is deposited into dyked-off areas in pit. The model assumes homogeneous CT deposit reaching 80 wt% solids shortly after deposition. In reality CT deposits exhibit some non-homogeneity. Figure 4-3 shows the site layout, in-pit dykes, reclamation schedule and closure landscape for the VM1 Composite Tailings Case.

4.3. Centrifuged Fluid Tailings Stacked (CFTS)

MFT are recovered and treated by adding flocculent and gypsum and the slurry is processed in solid-bowl scroll centrifuges to produce centrifuge cake at a nominal 55% solids concentration. The cake is trucked to the disposal area and stacked in thin lifts (<2 m). Each lift is allowed to dewater from freeze-thaw cycles and evaporation effects over a 17-month period until the next lift is placed. Through those atmospheric effects, the cake layers reach approximately 75% solids concentration after 4 years. After additional long-term consolidation, the solids concentration is assumed to reach approximately 80%. Figure 4-4 shows the site layout, in-pit dykes, reclamation schedule and closure landscape for the VM1 Centrifuge Case.

Figure 4-5 shows the site layout, in-pit dykes, reclamation schedule and closure landscape for the VM3 Centrifuge Case.

4.4. In-Line Flocculation (ILF)

MFT are recovered and treated by adding flocculent using in-line mixing and discharging the flocculated slurry into mined-out areas of the pit to form a deep, fines-dominated deposit (using COSIA terminology). The action of the flocculent promotes rapid dewatering of the slurry so an average solids concentration of 50 wt% is reached during the deposition period. Following initial dewatering, self-weight consolidation occurs very slowly due to the long seepage path through the deep deposit and low hydraulic conductivity of the material. It is assumed for the plan that the deposit remains at 50 wt% solids until mining and closure landform creation are complete. Long-term consolidation of the deposit is expected over the ensuing decades and centuries, ending when the deposit volume reaches approximately half of the original volume (at approximately 80 wt% solids). To mitigate the low strength and long-term subsidence of the deposit surface, the deposit is placed in-pit, below grade and water-capped for closure. Figure 4-6 shows the site layout, in-pit dykes, reclamation schedule and closure landscape for the VM1 In-Line Flocculation Case.

Figure 4-7 shows the site layout, in-pit dykes, reclamation schedule and closure landscape for the VM3 In-Line Flocculation Case.

4.5. Fluid Tailings Thermal Drying (FTTD)[2]

MFT is allowed to accumulate to a minimum working inventory and then processed using the same technology as the centrifuge case. Most of the centrifuge cake, at 55 wt% solids, is fed to thermal dryers where the remaining water is evaporated. The fully dried cake is then mixed with additional centrifuge cake at 55 wt% to achieve an average solids concentration of 85 wt% solids, then placed in a waste disposal area as with overburden. Figure 4-8 shows the site layout, in-pit dykes, reclamation schedule and closure landscape for the VM1 Thermal Drying Case.

[2] Thermal dryer heat loads and heat recovery provided by Joe Bonem of Bepex.

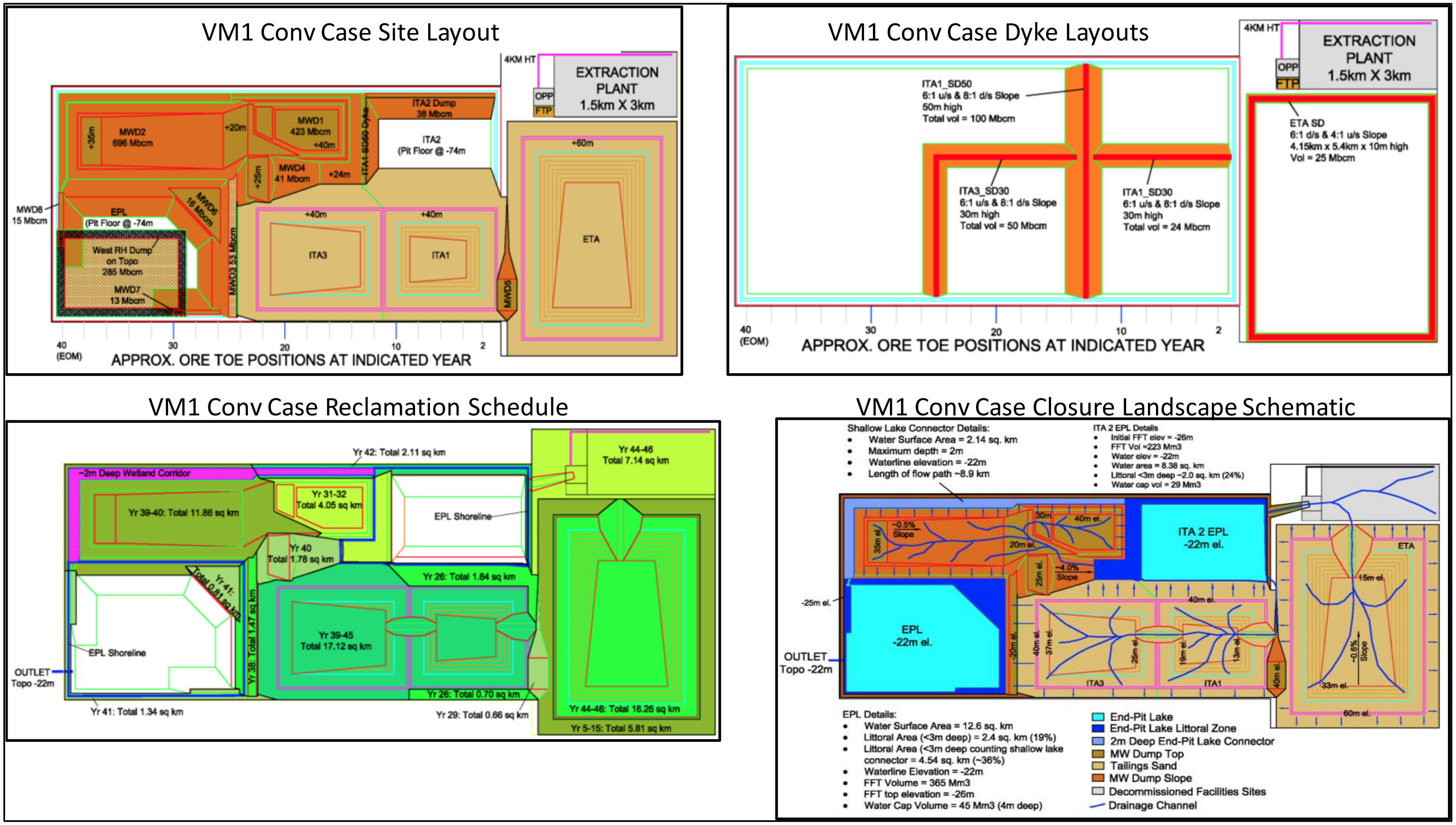

Figure 4-1. Site Layout and Progression to Closure for VM1 Conventional Case

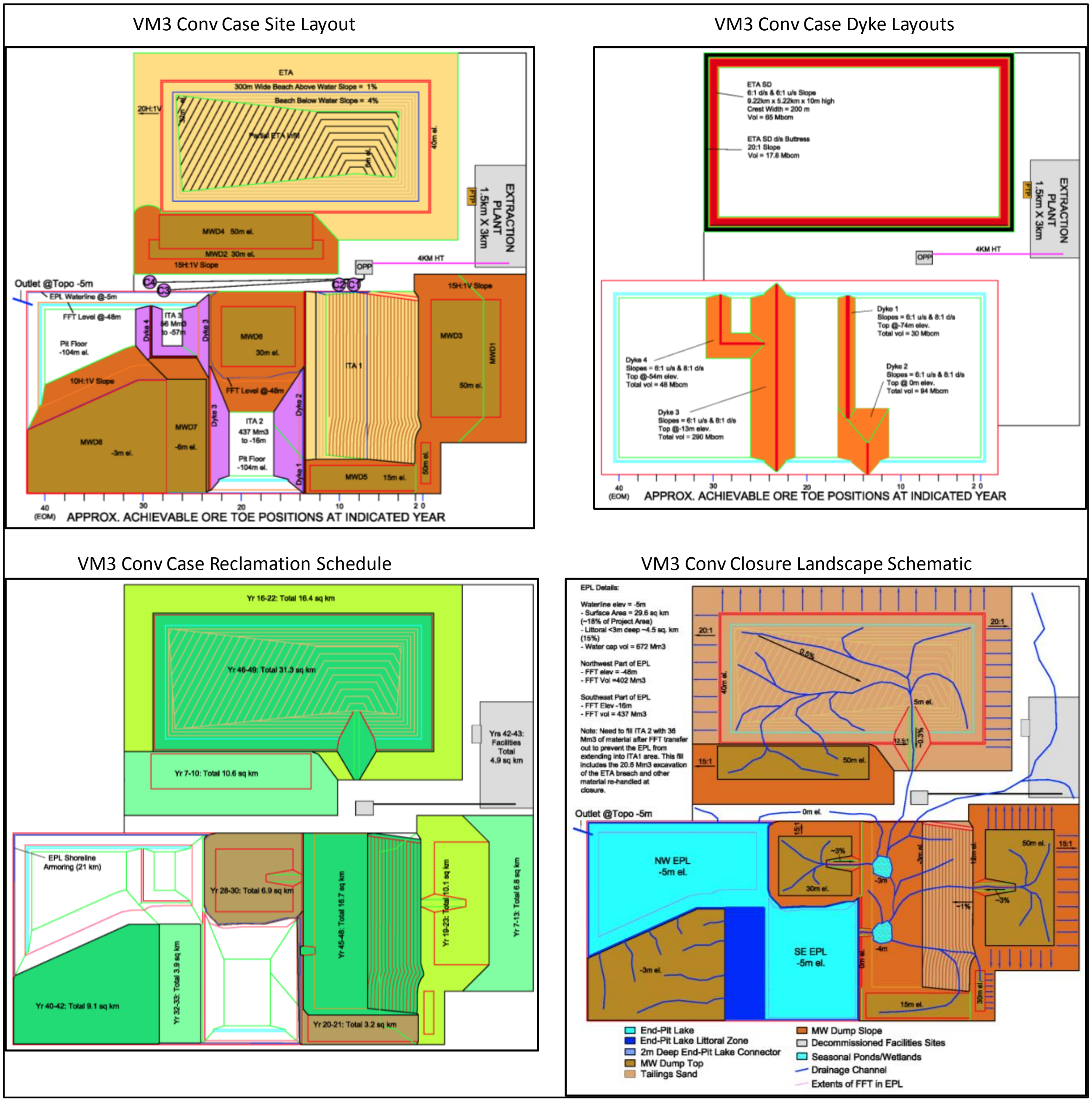

Figure 4-2. Site Layout and Progression to Closure for VM3 Conventional Case

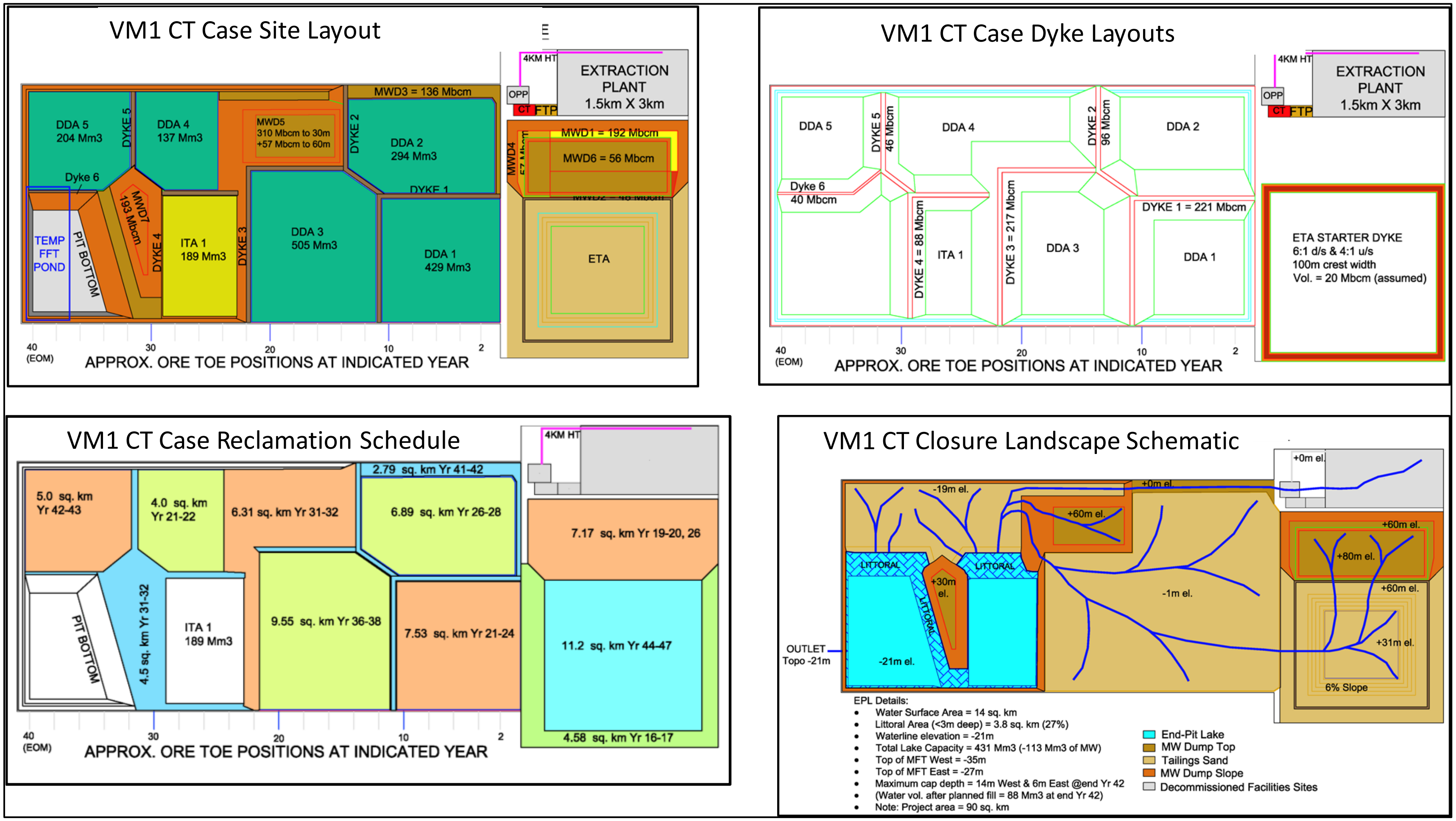

Figure 4-3. Site Layout and Progression to Closure for VM1 CT Case

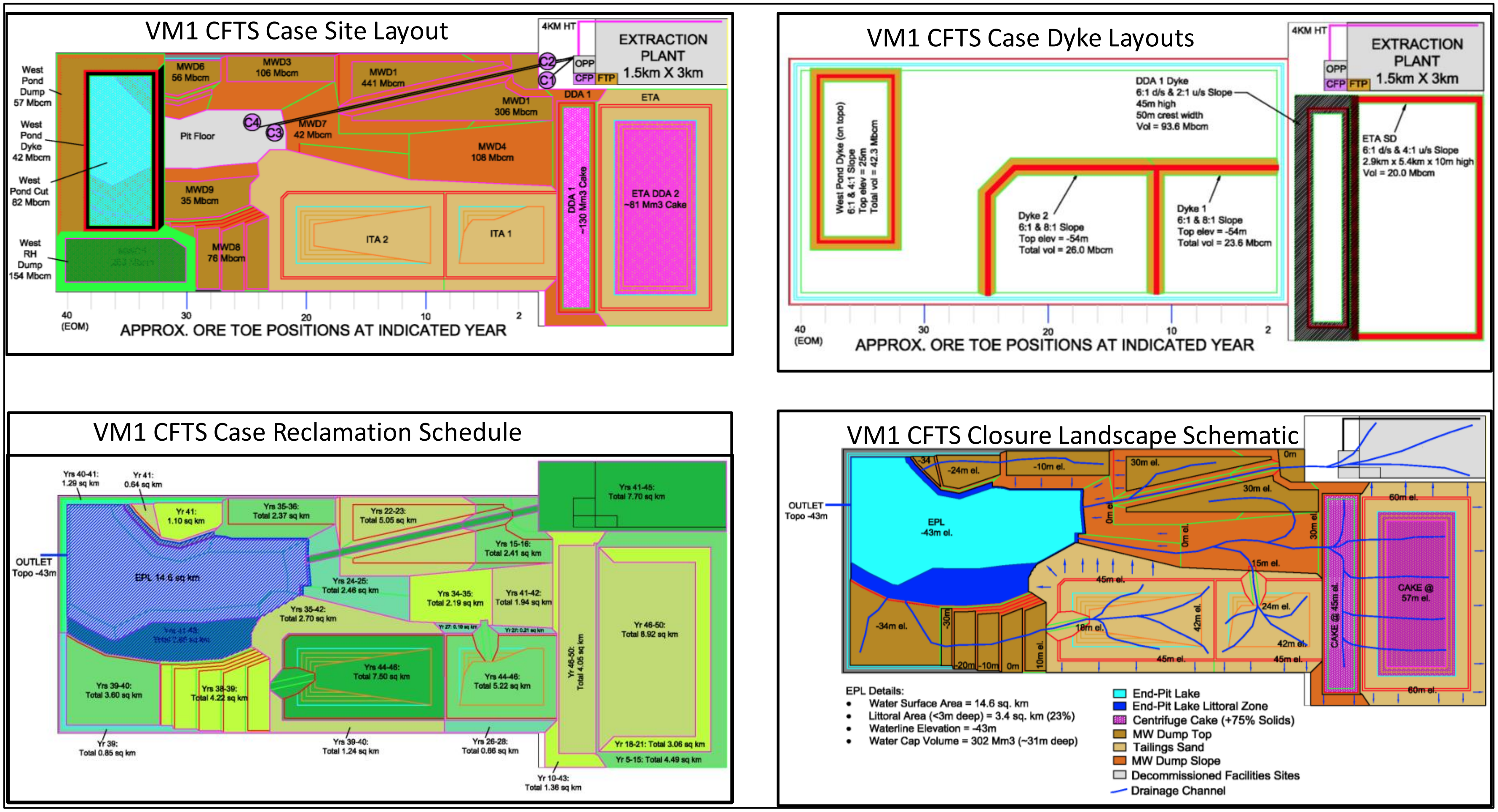

Figure 4-4. Site Layout and Progression to Closure for VM1 CFTS Case

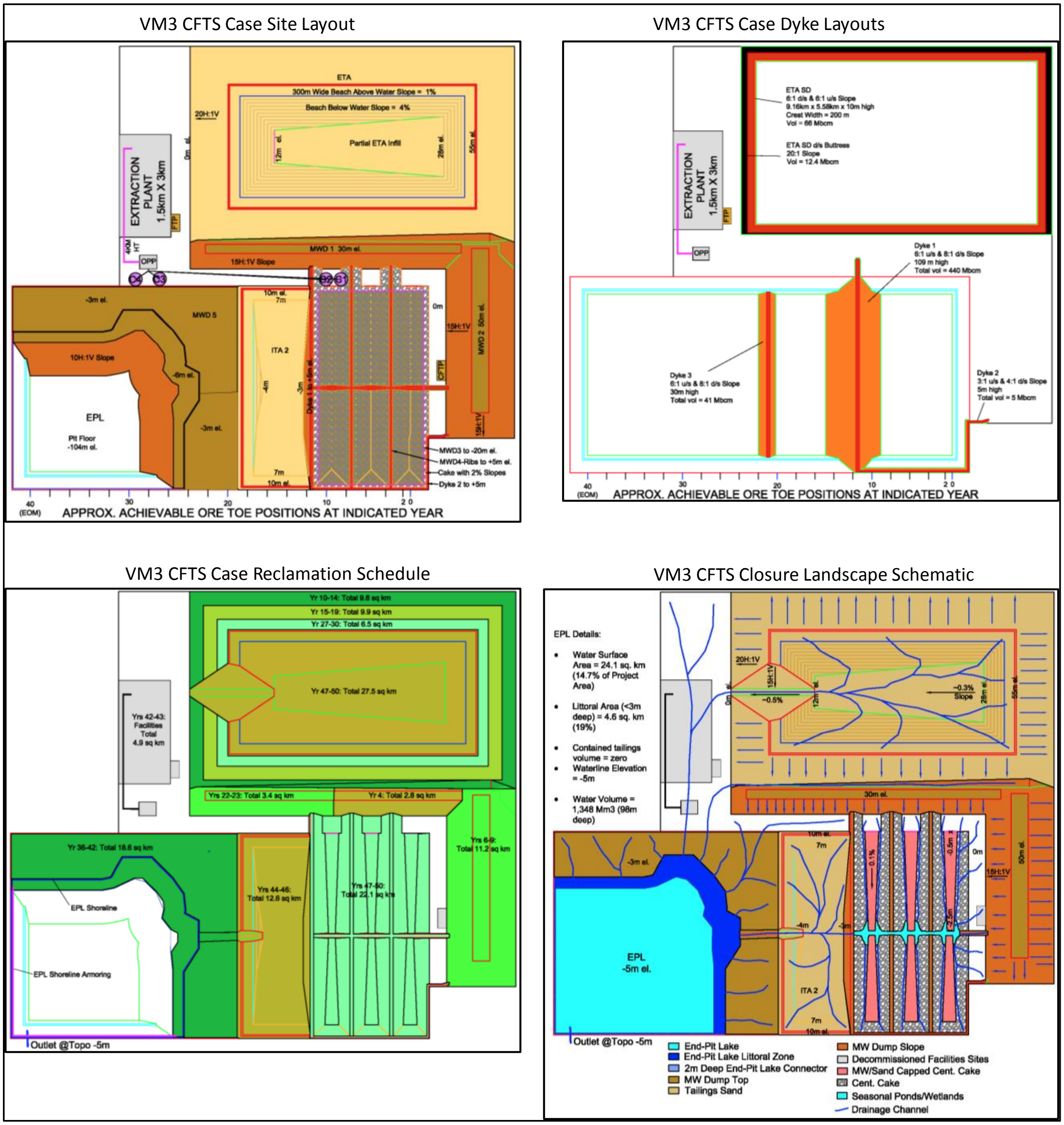

Figure 4-5. Site Layout and Progression to Closure for VM3 CFTS Case

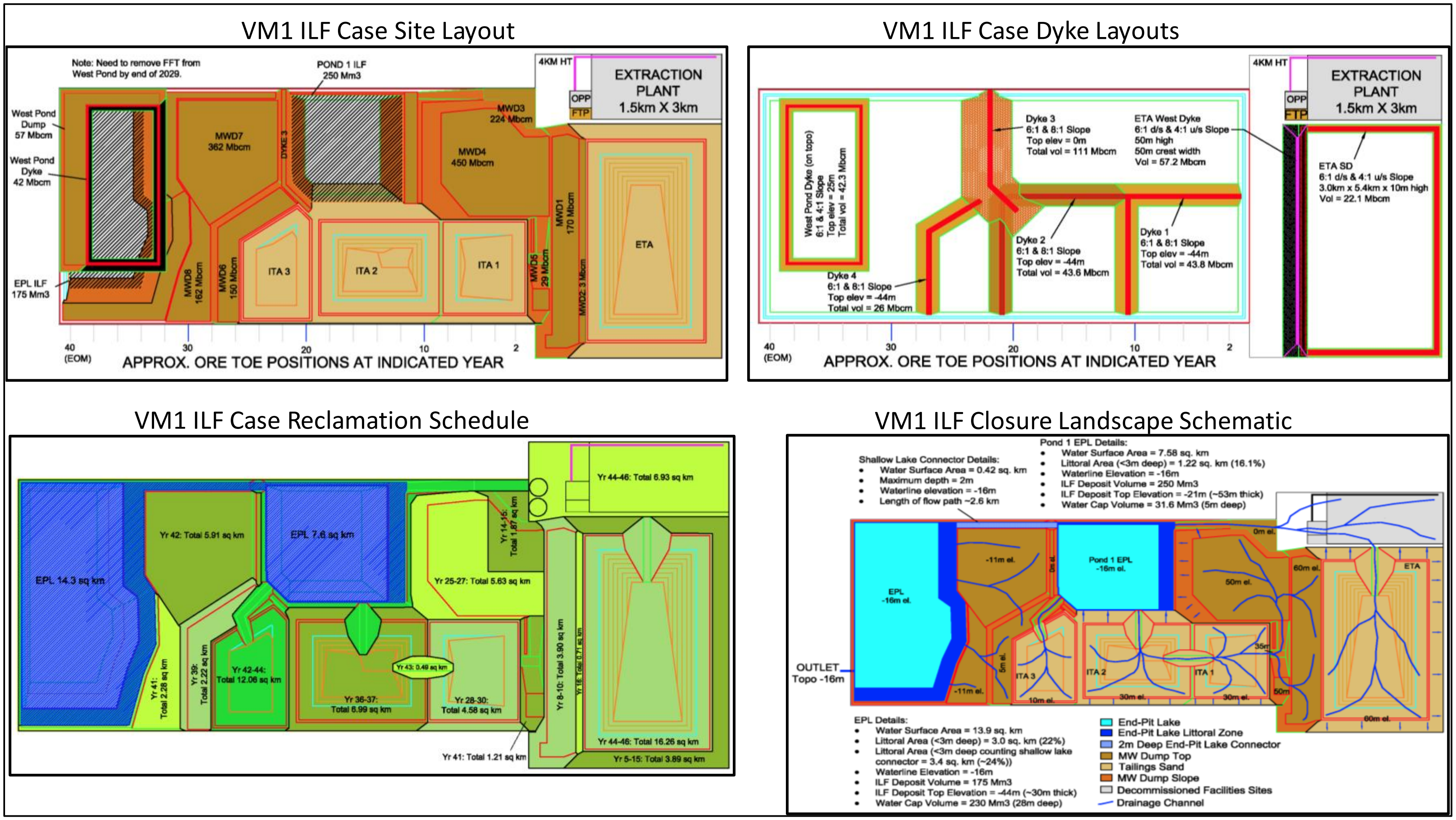

Figure 4-6. Site Layout and Progression to Closure for VM1 ILF Case

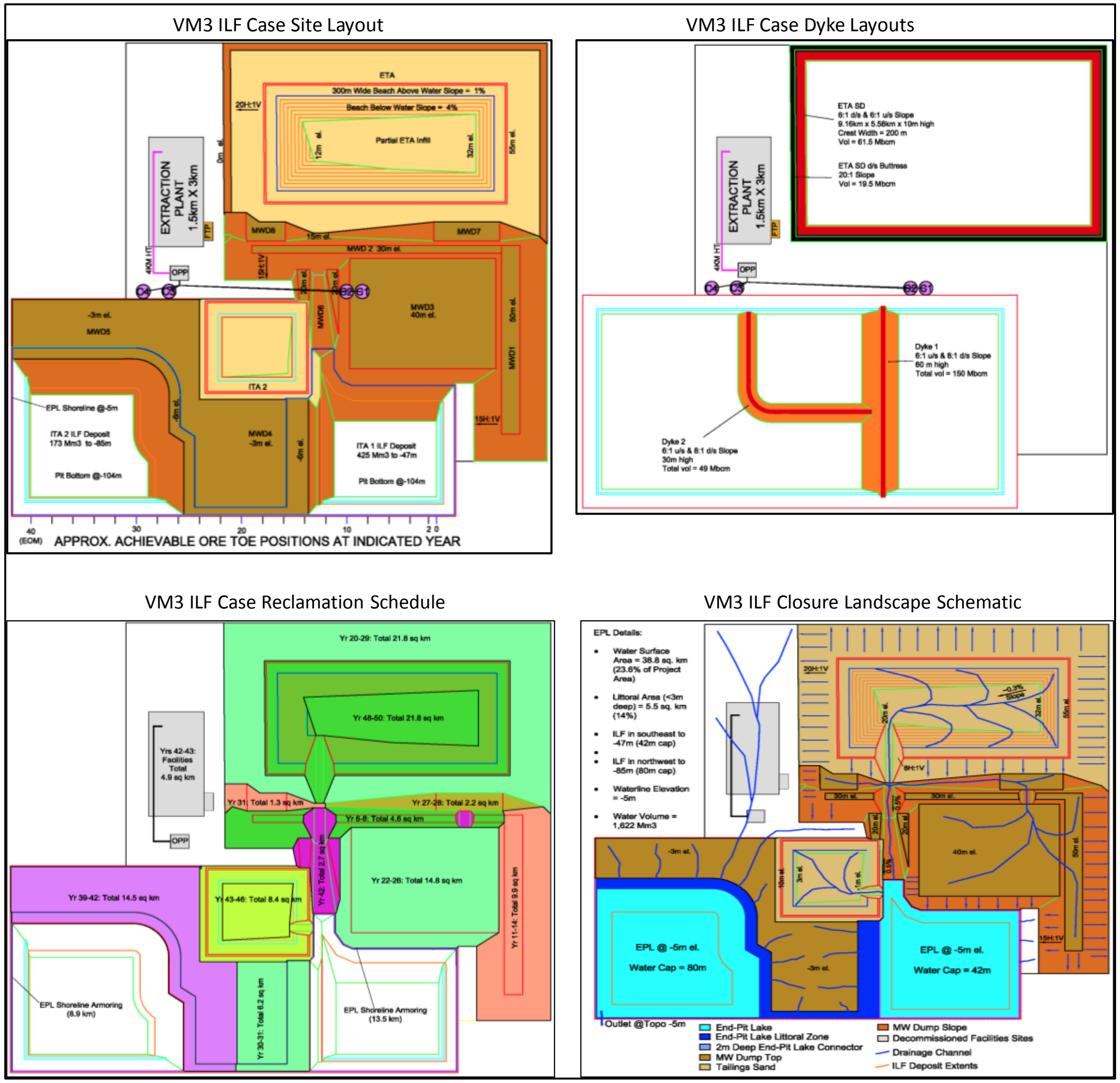

Figure 4-7. Site Layout and Progression to Closure for VM3 ILF Case

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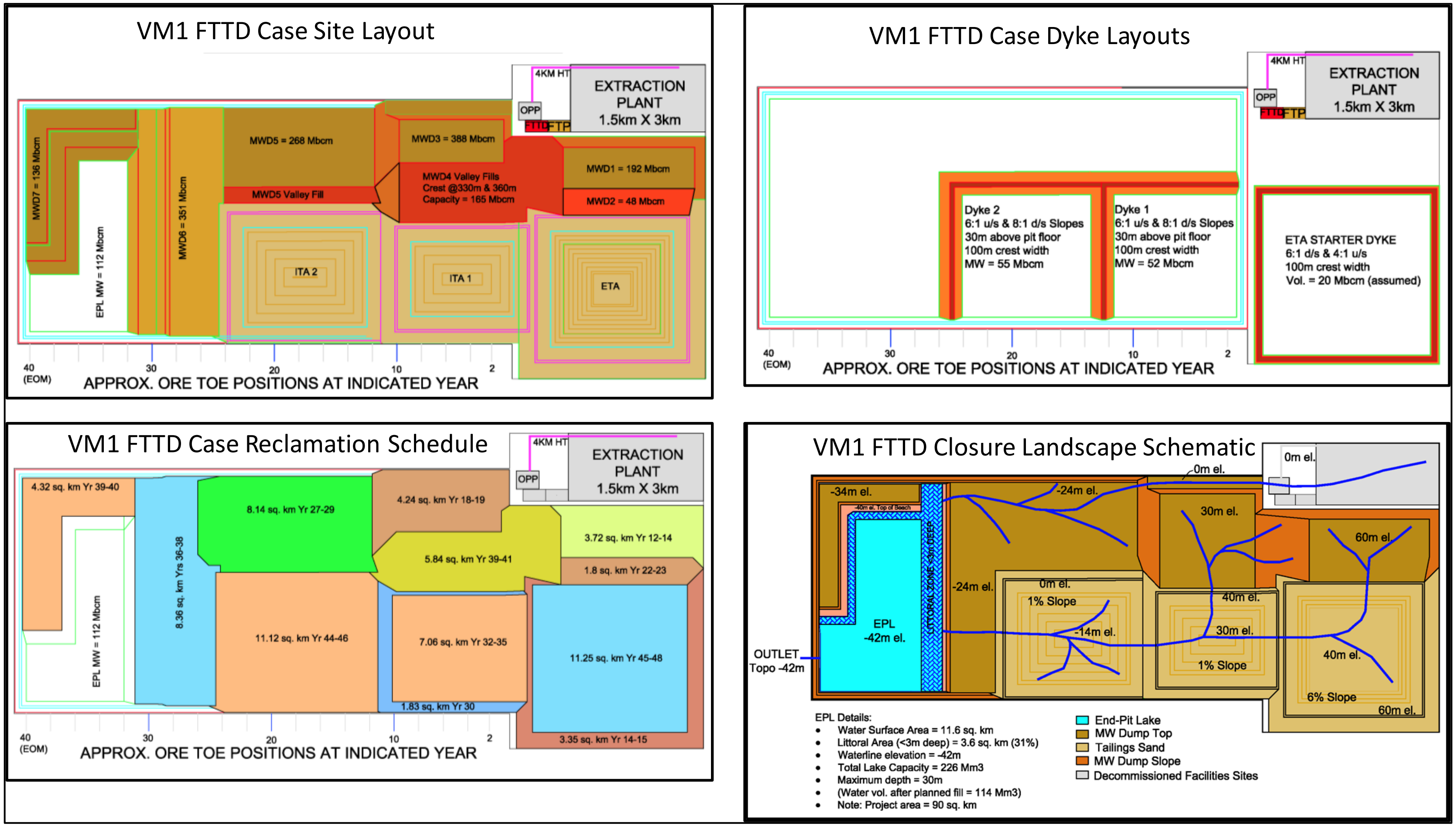


Figure 4-8. Site Layout and Progression to Closure for VM1 Thermal Drying Case

5. COMPARISON OF CASES

5.1. Comparison of Costs

Table 5.1 provides a summary of estimated costs for each case by category. In the bottom row of the table, the cases are ranked in order of lowest to highest net present cost (NPC) using a nominal 8% discount rate.

Table 5.1.
Summary of Estimated Costs for all Cases

Category	VM1 Conventional ($M)	VM1 In-line Flocculent ($M)	VM1 Centrifuge Freeze Thaw ($M)	VM1 Fluid Tails Thermal Drying ($M)	VM1 Composite Tailings ($M)	VM3 Conventional ($M)	VM3 In-line Flocculent ($M)	VM3 Centrifuge Freeze Thaw ($M)
Sub-Total Mining Costs	**18,842**	**18,289**	**19,032**	**16,686**	**20,385**	**25,575**	**25,587**	**25,532**
Contract Mining	854	755	687	512	512	1,103	1,050	1,127
Shovels	2,942	2,784	3,025	2,721	3,583	3,989	3,992	3,987
Trucks	11,289	10,811	11,394	9,876	10,534	15,125	15,694	15,170
Suppport Equipment	1,922	1,841	1,931	1,748	1,790	2,604	2,659	2,617
Crushers/Conveyors	1,463	1,593	1,608	1,558	2,727	1,871	1,663	1,714
MW Dyke Constr.	276	421	285	176	1,008	731	362	764
Dump Trafficability	97	84	102	94	230	152	166	151
Sub-Total Tailings Costs	**2,956**	**8,197**	**7,994**	**7,783**	**5,515**	**3,585**	**8,515**	**10,601**
PSV U/F & CT	2,205	1,989	2,245	2,199	2,308	2,818	2,173	2,870
Flotation Tails	241	243	241	179	289	242	207	223
Froth Treatment Tails	107	116	112	54	54	107	93	100
Cyclone O/F	-	-	-	-	55	-	-	-
FFT Reclaim & Transfers	328	787	535	312	459	374	608	282
FFT Treatment Process	-	5,009	4,834	4,980	2,332	-	5,369	7,060
Sand Dyke Constr.	74	54	29	59	19	44	66	67
Sub-Total Reclamation & Closure Costs	**676**	**735**	**566**	**478**	**611**	**867**	**839**	**810**
Reclamation	435	437	464	323	319	507	448	520
Re-handle & armoring	241	298	103	155	292	360	391	290
Sub-Total Other Costs	**300**	**289**	**303**	**263**	**341**	**401**	**415**	**402**
Mine Fleet GHG Offsets	300	289	303	263	341	401	415	402
Total Costs	**22,774**	**27,510**	**27,896**	**25,210**	**26,851**	**30,428**	**35,356**	**37,345**
Total Costs per Bbl of Rec. Bitumen ($/bbl)[1]	8.10	9.79	9.93	8.97	9.55	10.83	12.58	13.29
Total NPC @8% Nominal Discount Rate	***7,120***	***7,742***	***8,500***	***8,358***	***8,376***	***9,585***	***10,513***	***10,583***
Rank by Lowest $NPC_{8\%}$	**1**	**2**	**5**	**3**	**4**	**1**	**2**	**3**

Note 1: Total recovered barrels of bitumen for each case was set at 2,810.5 Mmbbls.
Note 2: Thermal drying costs assume that 80% of drying energy can be recovered and used as a net energy saving. As noted in the text, this is an unlikely scenario.
Note 3: For the CT Case, high indirect mining costs (crusher relocations and mine waste dyke construction) caused a low rank despite relatively low FFT treatment costs.

5.1.1. Mining Costs

A summary of the non-discounted mining costs is shown on Figure 5-1. The VM1 FTTD Case had significantly lower mining costs than any of the other cases even though dried cake haulage costs were included in the mining costs and did not occur in some of the other cases. The VM3 CFTS Case non-cake mining costs were sufficiently low to enable the cake haulage costs to be absorbed without seeing an overall escalation of mining costs on a comparative basis.

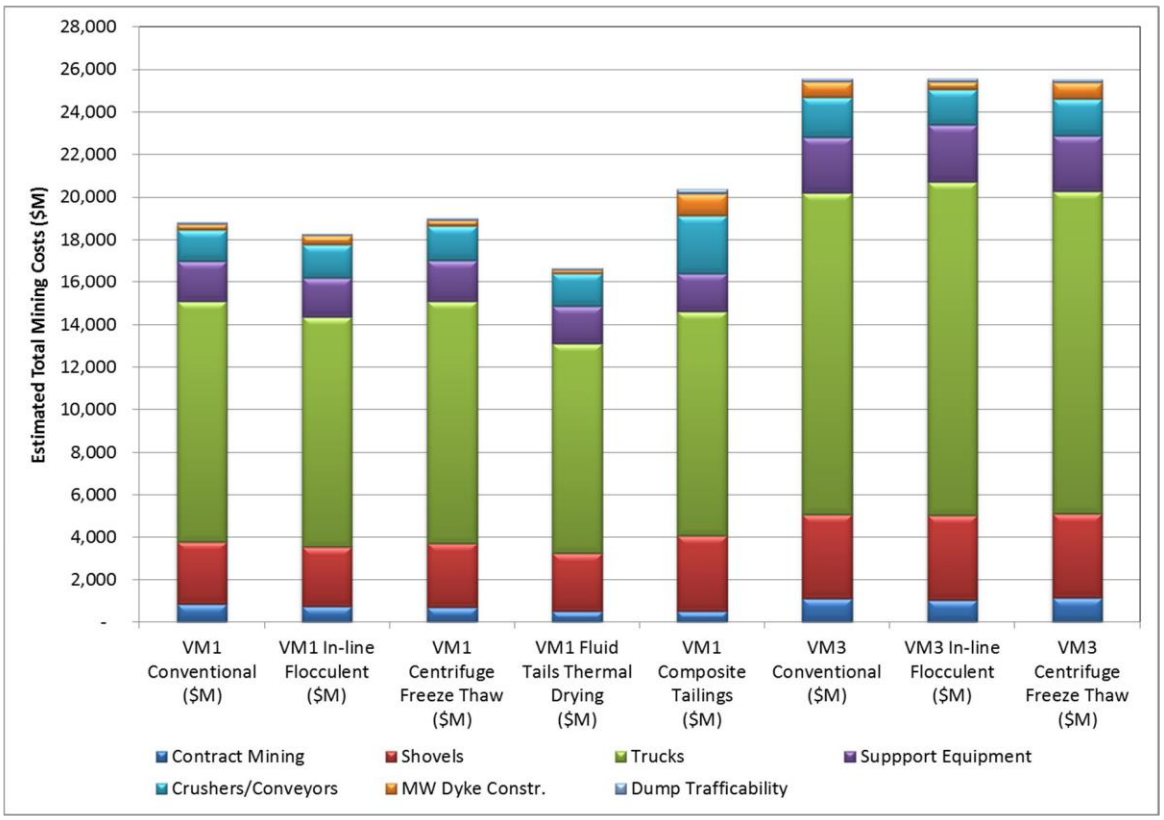


Figure 5-1. Summary of Estimated Mining Costs for all Cases

5.1.2. Tailings Costs

A summary of the non-discounted tailings costs is shown on Figure 5-2. The Conventional Cases had significantly lower tailings costs than any of the other cases. The total magnitude of the FFT treatment costs was calculated based on a combination of throughput rates (which affected CapEx) and total mass of fines processed (which affected OpEx and sustaining CapEx). The high throughput rate required for the ILF process caused CapEx to be higher than might be expected and explains why the tailings costs for the case were higher than for centrifuge-based FFT treatment processes and not much lower than for the high-fines VM3 ILF Case. The opposite effect occurred for the VM3 Cases, where the ILF process operated for a longer period than the CFTS process so had a lower CapEx relative to the CFTS Case.

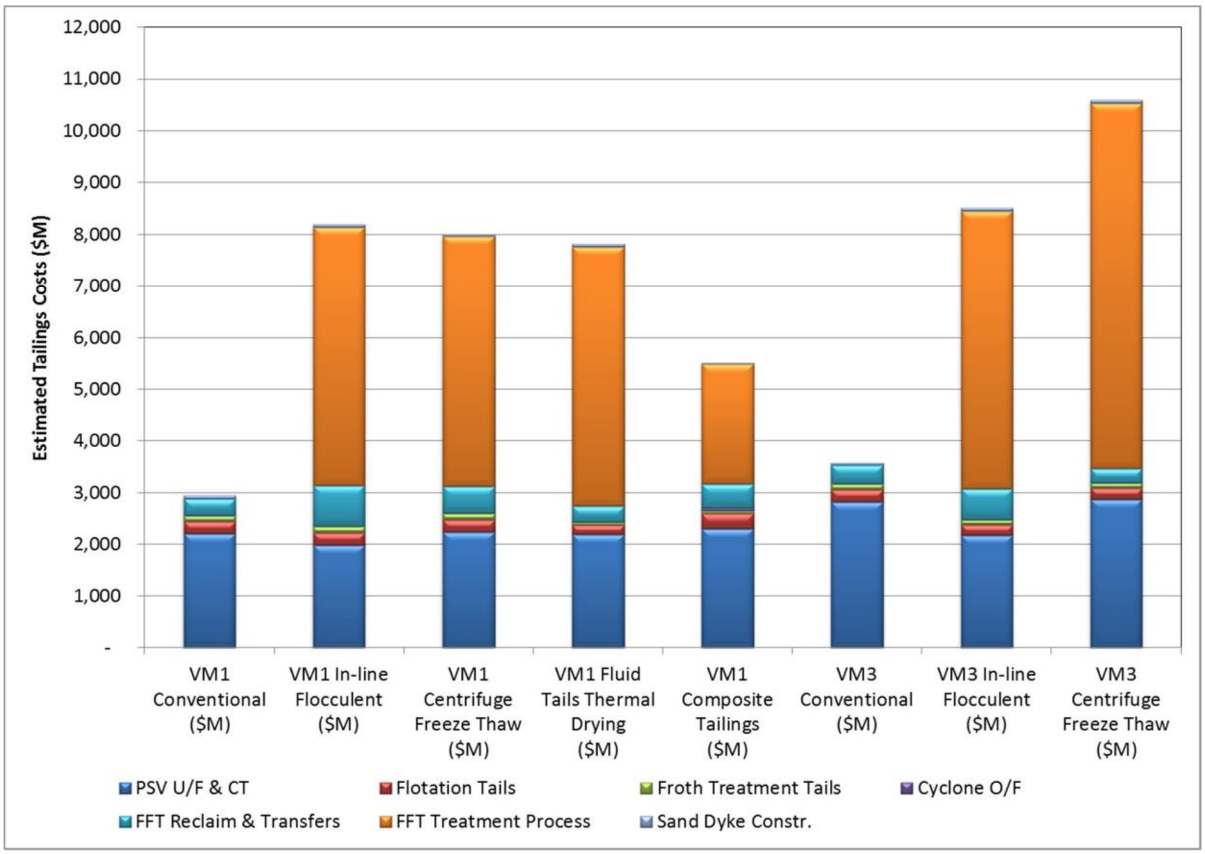


Figure 5-2. Summary of Estimated Tailings Costs for all Cases

5.1.3. FFT Treatment Costs

A very significant differentiating cost category for the Study was the FFT treatment cost. It is worth special mention that there is no industry experience with thermal drying technology and the heat recovery and reuse that would be essential to make it viable. The net results of the FFT processing cost assumptions are shown on Figure 5-3. The reasons for the highly variable unit initial CapEx values for the same technologies at different sites are due to the strategy employed for use of the technologies and resulting capacities of the treatment processes.

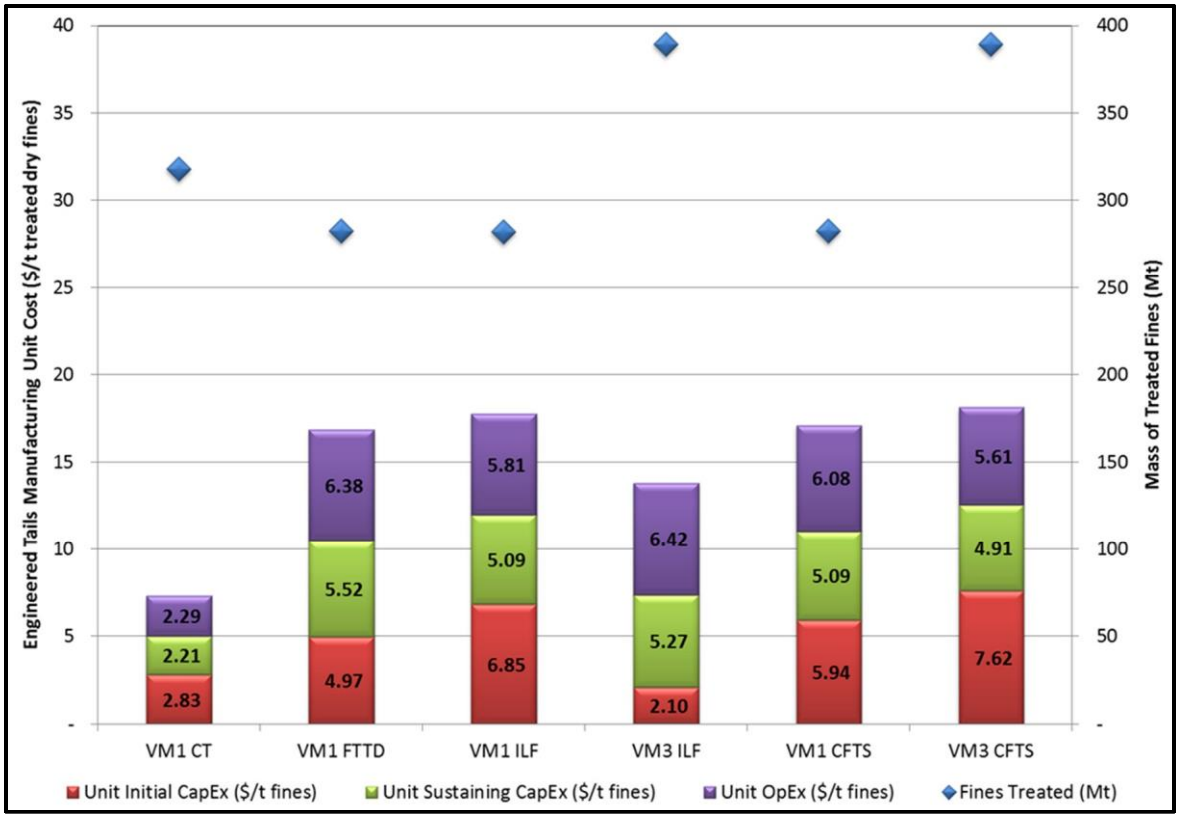


Figure 5-3. Summary of Estimated FFT Treatment Costs

5.1.4. Reclamation & Other Costs

A summary of reclamation and other costs for all cases is shown on Figure 5-4. The VM1 FTTD Case had the lowest total costs in these categories due to lower mine fleet emissions charges (but not including dryer emission charges), avoidance of reclamation stockpile relocations, and relatively short EPL shoreline requiring armoring. The VM3 Cases had higher total costs in the "Reclamation and Other" categories due primarily to larger land disturbance, long EPL shorelines requiring armoring, and higher mine waste volumes which led to higher mine fleet emissions.

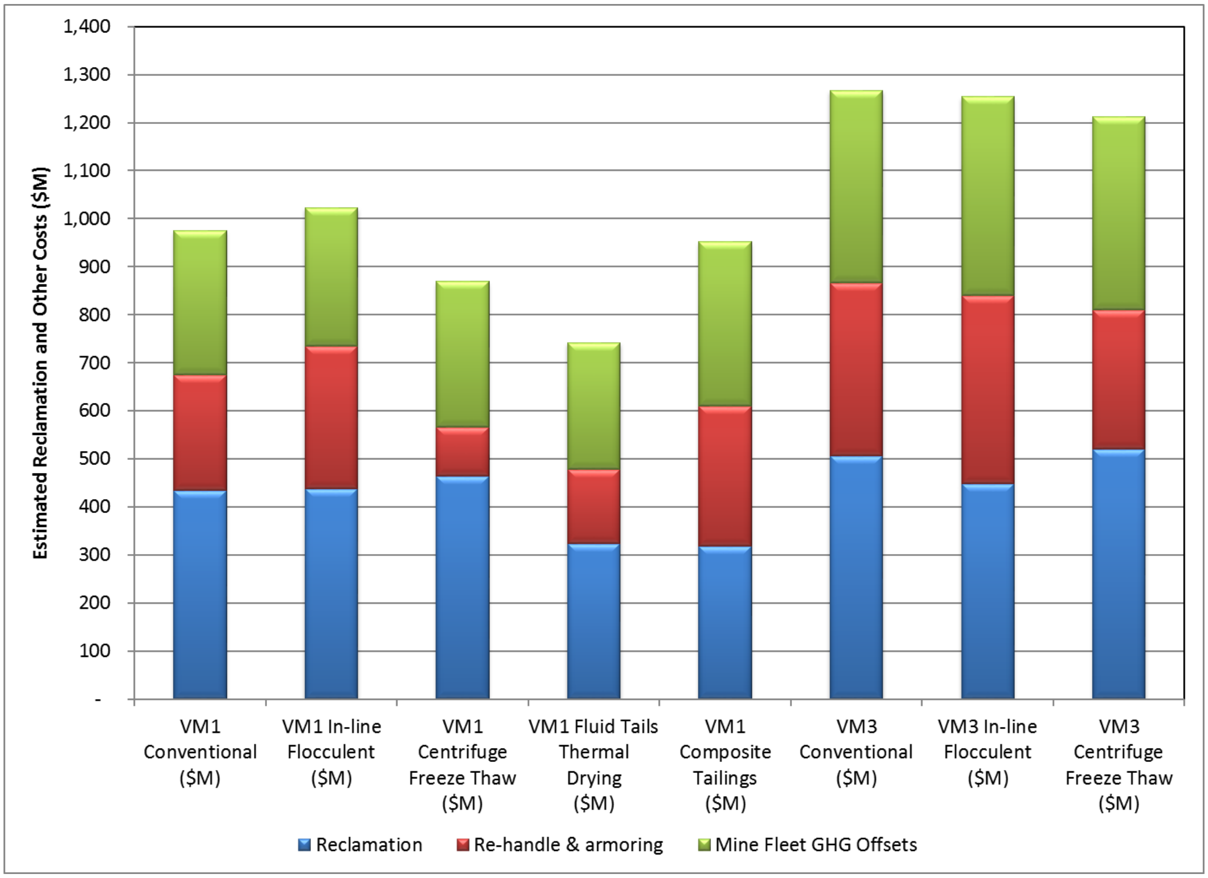
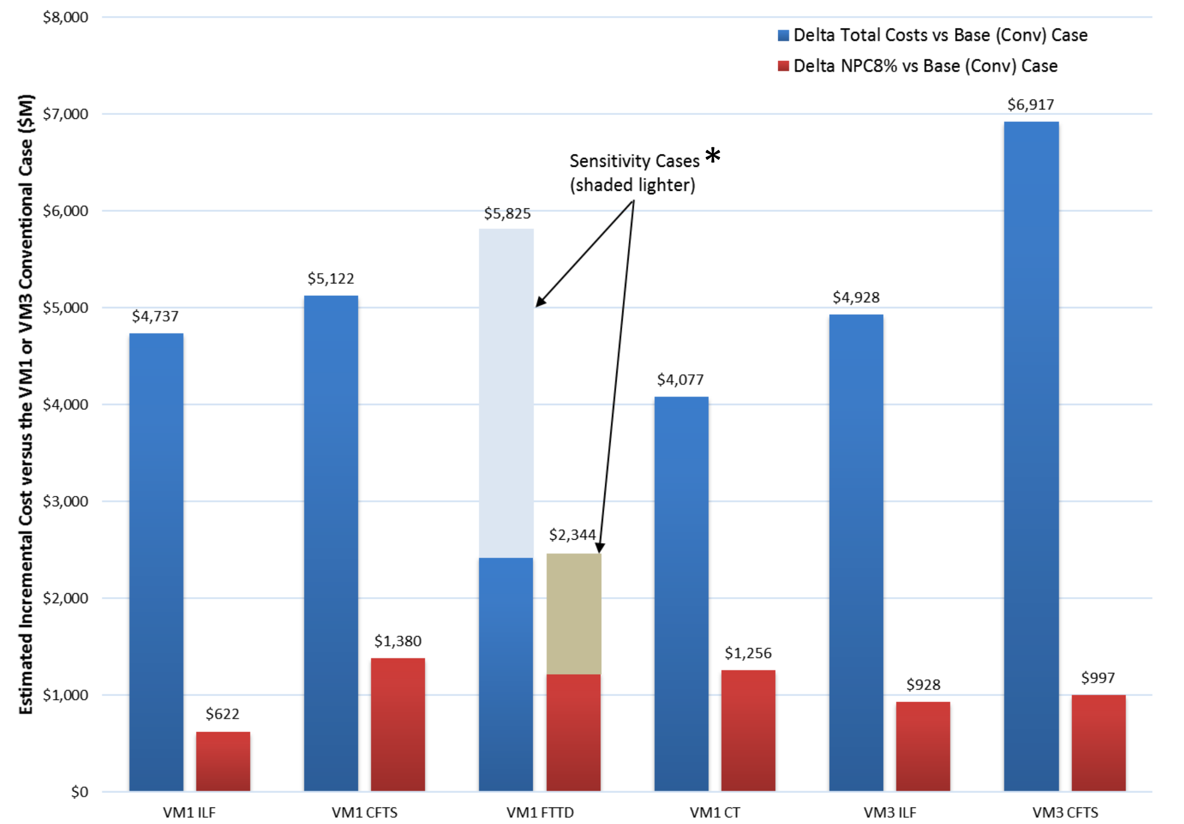


Figure 5-4. Summary of Estimated Reclamation and Other Costs for all Cases

5.1.5. Cost Increment Compared to Conventional Case

It is not surprising to find that the conventional tailings cases for both VM sites represent the lowest costs. The purpose of including the conventional cases was to provide a clear basis in terms of cost and conceptual environmental impacts against which the expected costs and impacts of the engineered tailings approaches could be measured. To that end, Figure 5-5 provides a summary of the incremental costs for the VM1 and VM3 engineered tailings cases compared to the respective Conventional Cases. A sensitivity analysis was performed on the heat recovery efficiency (80% vs. 50%) of the FTTD case, which is also presented on Figure 5-5.



*FTD heat recovery is 80% (solid bars) and 50% (shaded bars)

Figure 5-5. Cost Increment Relative to the Base (Conv) Case

5.2. Mine Fleet and Thermal Dryer GHGs

The cumulative mine fleet GHG emissions curves for all cases are shown on Figure 5-6. When considering only mine fleet emissions, the FTTD Case offers a clear reduction compared to the other VM1 Cases. However, when a portion of dryer emissions - 20% or 50% - is included, the FTTD case has the highest GHG emissions. For VM3, the CFTS Case offers a small advantage over the other VM3 Cases owing to shorter average haul distances required.

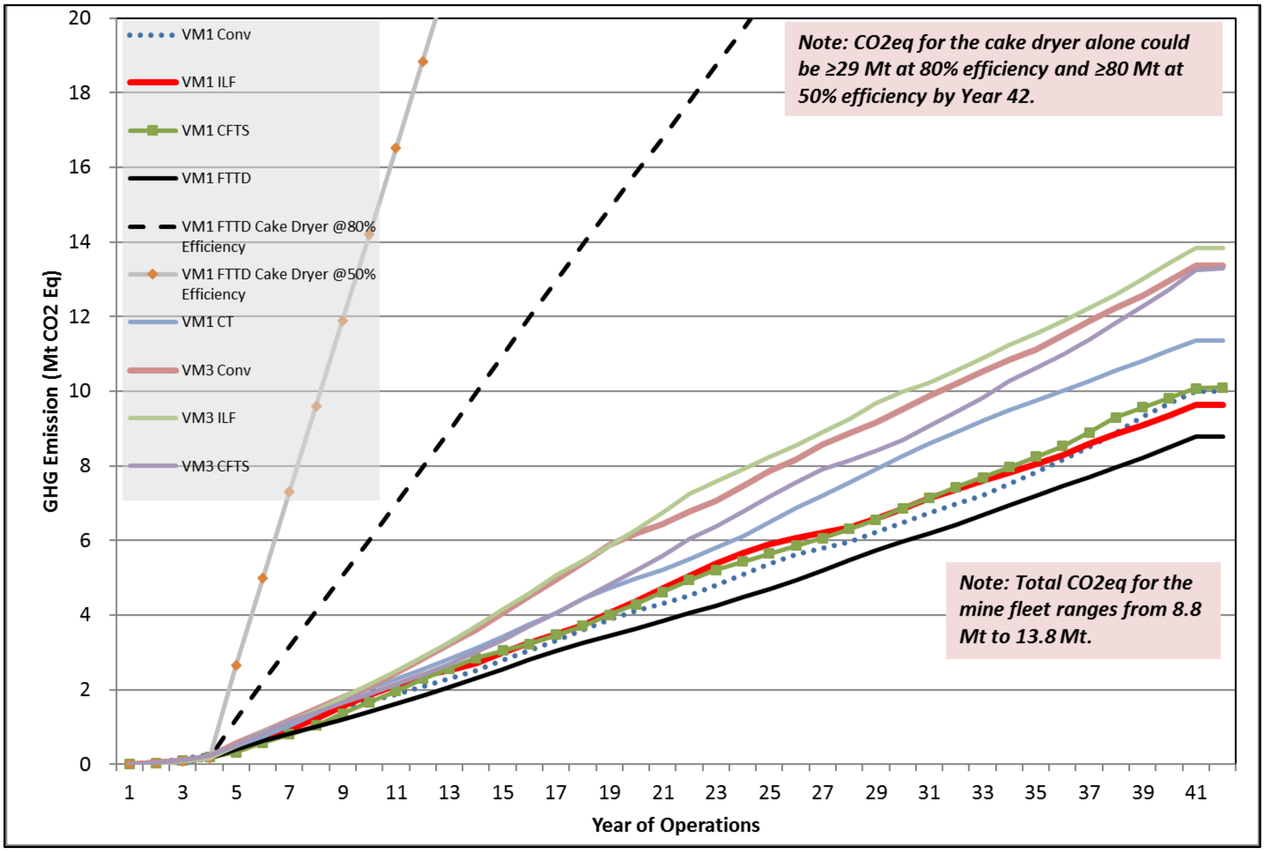


Figure 5-6. Estimated Mine Fleet and Thermal Dryer GHG Emissions for all Cases

5.3. Net Land Disturbance

The net land disturbance areas through mine life for all cases are shown on Figure 5-7. Differences in net disturbance between technologies within the VM1 and VM3 cases relate to the timing and duration of the different treated FFT deposits and therefore, when areas are available to reclaim. The more significant difference for both net disturbance and total project development area relates to the resource characteristics itself – with lower grade and especially higher waste:ore ratio requiring greater area for comparable production.

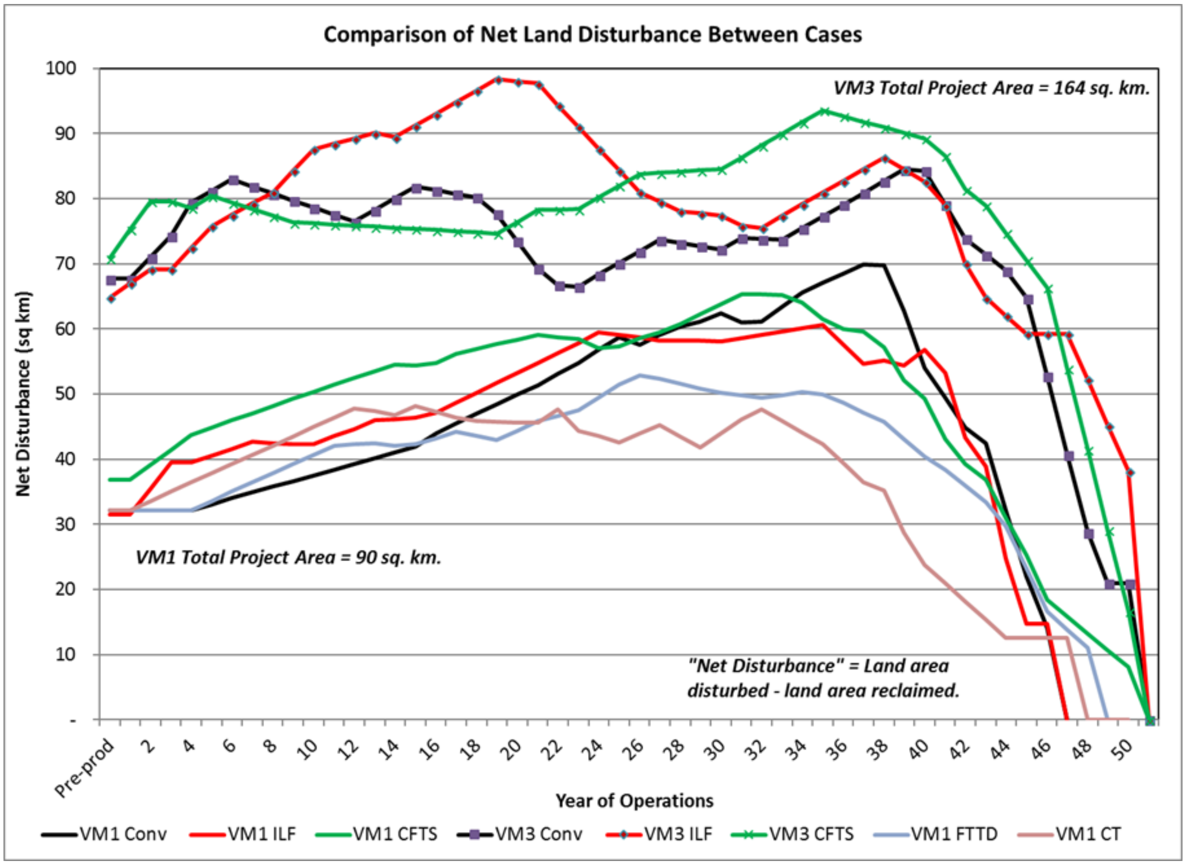


Figure 5-7. Comparison of Net Land Disturbance for all Cases

5.4. Closure Landscape Design and Total Water in the Closure Landscape

The closure landscape schematics for the VM1 Cases are compared on Figure 5-8. For the portions of the project areas that would be reclaimed as terrestrial landscapes, some differences exist in terms of substrates for the reclaimed landforms, slopes, and amounts of land reclaimed as terrestrial landscapes.

The VM1 FTTD Case - with the largest percentage of the project area allocated to terrestrial reclamation, significant use of mine waste as reclamation soil substrate and sensible landform slopes - appears to offer some advantages compared to the other VM1 Cases. Potential advantages offered by the VM1 CFTS Case are offset by the presence of elevated cake deposits in the DDA and ETA, which are at risk respecting the land area and time requirements to achieve the required strength. The VM1 Conventional and ILF Cases avoid the land area and timing challenges inherent in the stacked deposits of the CFTS cases but have larger fluid tailings deposits in the EPLs and greater EPL area.

The closure landscape schematics for the VM3 Cases are shown on Figure 5-9. There is little difference between the VM3 Cases with respect to terrestrial reclamation components.

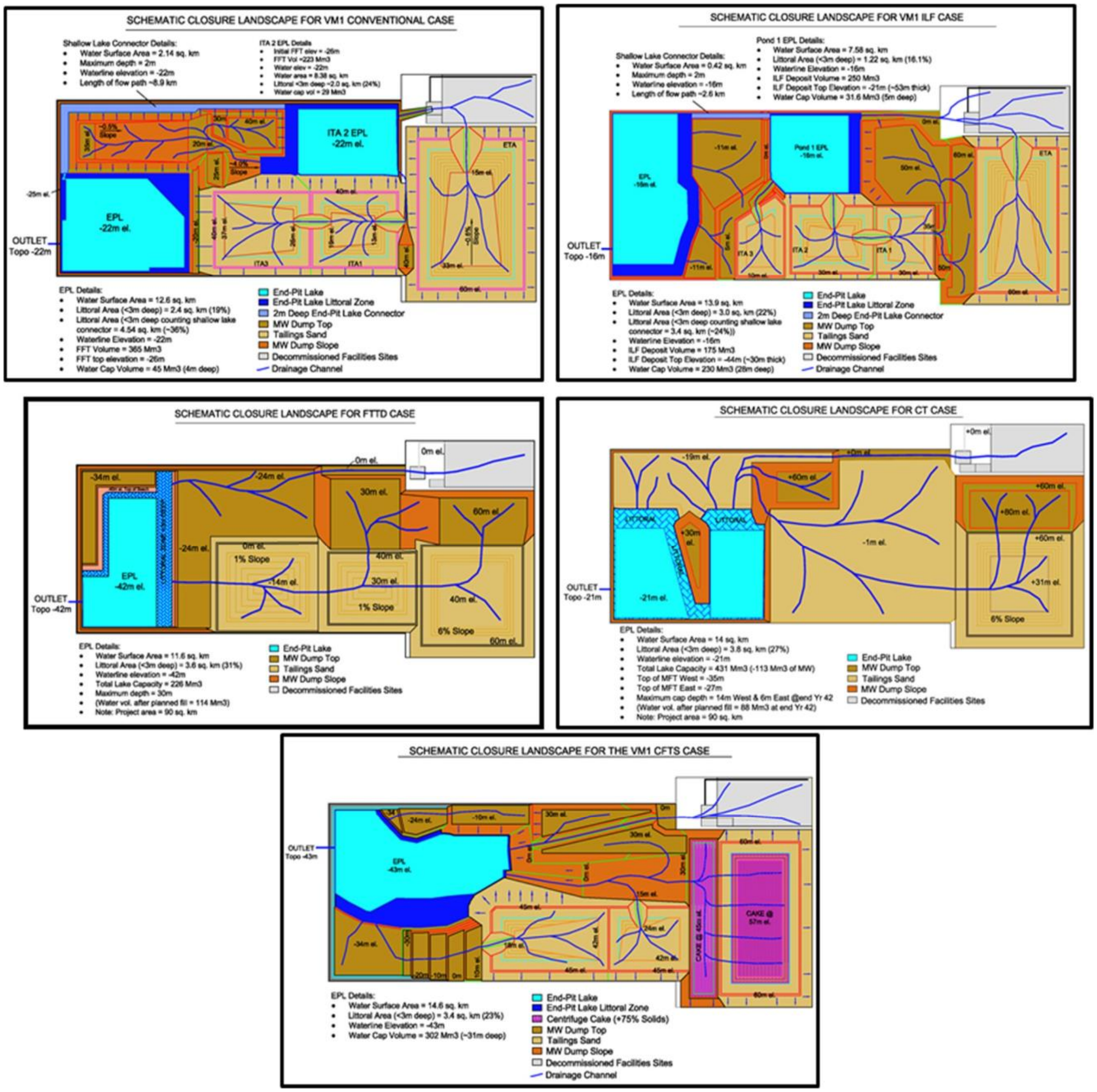


Figure 5-8. Compilation of Closure Landscape Schematics for all VM1 Cases

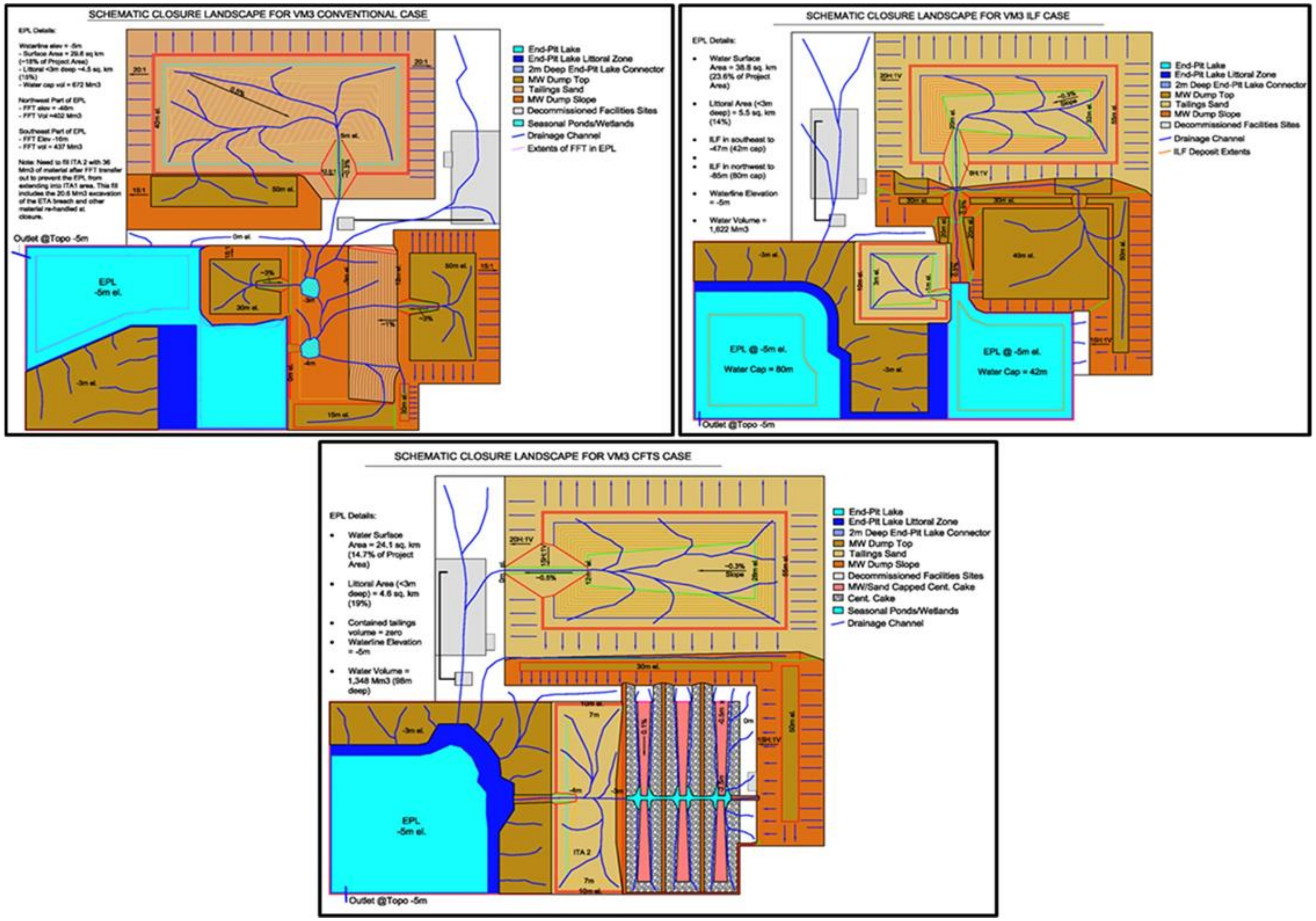


Figure 5-9. Compilation of Closure Landscape Schematics for all VM3 Cases

Table 5.2 summarizes key EPL design parameters for all cases evaluated in this Study. Apart from the uncertain effects of various forms of fine tailings deposits within the EPLs (which are the subject of ongoing research), key metrics for EPL feasibility and performance are thought to include, but are not limited to, littoral zone as a percentage of lake surface area, lake surface area as a percentage of the watershed area feeding the lake, and water cap depth.

Table 5.2.
Summary of EPL Statistics

Case/Lake		Water Level (m)	Tails Vol. (Mm3)	Initial Water Cap Vol. (Mm3)	Total Vol. (Mm3)	Lake Surface Area (sq. km)	Littoral Zone <3m Deep (sq. km)	Littoral Zone <3m Deep (% of lake)	Lake Depth (m)	Initial Water Cap Thickness (m)	Lake Surface Area Percent of Total Site Area (%)
VM1 Conv.	Lake 1	-22	FFT = 223	29	252	8.38	2.00	23.9%	52	4	9%
	Lake 2	-22	FFT = 365	45	410	12.60	2.40	19.0%	52	4	14%
	Connector	-22	--	4	4	2.14	2.14	100.0%	2	2	2%
	Total	**-22**	**588**	**78**	**666**	**23.12**	**6.54**	**28.3%**	**52**	**~4**	**26%**
VM1 ILF	Lake 1	-16	ILF = 250	32	282	7.58	1.22	16.1%	58	5	8%
	Lake 2	-16	ILF = 175	230	405	13.90	3.00	21.6%	58	28	15%
	Connector	-16	--	1	1	0.42	0.42	100.0%	2	2	0%
	Total	**-16**	**425**	**263**	**688**	**21.90**	**4.64**	**21.2%**	**58**	**var.**	**24%**
VM1 CFTS	EPL	-43	--	302	302	14.60	3.40	23.3%	31	31	16%
VM1 CT	EPL	-21	271	47	318	14.00	3.80	27.1%	53	6	16%
VM1 FTTD	EPL	-42	--	114	114	11.60	3.60	31.0%	32	32	13%
VM3 Conv.	Lake 1 (NW)	-5	FFT = 402	n.a.	n.a.	n.a.	n.a.	n.a.	99	43	n.a.
	Lake 2 (SE)	-5	FFT = 437	n.a.	n.a.	n.a.	n.a.	n.a.	99	11	n.a.
	Total	**-5**	**839**	**672**	**1,511**	**29.60**	**4.50**	**15.0%**	**99**	**var.**	**18%**
VM3 ILF	Lake 1 (W)	-5	ILF =173	n.a.	n.a.	n.a.	n.a.	n.a.	99	80	n.a.
	Lake 2 (E)	-5	ILF = 425	n.a.	n.a.	n.a.	n.a.	n.a.	99	42	n.a.
	Total	**-5**	**598**	**1,622**	**2,220**	**38.80**	**5.50**	**14.0%**	**99**	**var.**	**24%**
VM3 CFTS	EPL	-5	--	1,348	1,348	24.10	4.60	19.0%	99	99	15%

The littoral zones for all cases fall within 14% to 31% exceeding the guideline minimum of 10%. However, lake surface area to in-drainage area ratio and water depth require design attention on a site-specific basis.

At a scoping level, studies of project designs often use a rule-of-thumb minimum of 10:1 for watershed-to-EPL-area ratio to guide landform design. Several of the cases evaluated in this Study fall below that ratio by a wide margin, based on the assumption that the project area forms the entire available watershed feeding the EPL system. For example, the ratios for the VM1 Conv and VM3 ILF Cases are 3.8:1 and 4.2:1 respectively. (Additional drainage area outside of the project development area is usual on mine sites but is a site-specific factor beyond the scope of this Study.)

A second challenge associated with VM3 cases with their high ratio of overburden, is the total depth of the final pits. Without sufficient fill material at mine completion (i.e., ILF or centrifuge cake), water cap depths for the VM3 ILF and CFTS cases are greater than might be considered desirable to avoid meromictic behavior. If required in those cases, methods to reduce total lake depth could be possible without material change to the case outcomes. For example, access to a lower elevation water course would allow for a reduction of the lake surface level. This would also reduce the lake area and increase in-drainage area. These site-specific design considerations were beyond the scope of this study.

5.4.1. Water and Chloride Balance

Water volume and chloride concentration for “free” surface water, coarse tailings pore water, FFT pore water, and engineered treated tailings (CT, ILF, centrifuge cake, or thermally dried cake) at Mine Year 40, are shown in Table 5.3.

As expected, the treated tailings cases resulted in less total system water and higher chloride concentrations than the conventional cases. The water volumes in the balances were generally correlated with the water content of the treated FFT. In general, the lower the water content of the treated FFT (compared to the conventional case), the lower the volumes of “free” water and fine tailings (FFT plus treated FFT) pore water. Figures 5-10 and 5-11 show the trajectory of chloride concentrations through the mine life in the “free” water volume for the Conventional VM1 Case, which has the least chloride concentration buildup and the VM1 CT Case which reaches the highest peak concentration.

The higher levels of chloride reached in the high chloride cases may impact extraction processing and prolong stabilization of closure pit lakes. These high end-of-mining chloride concentrations could be mitigated by incorporating water discharge during the active mining period. Methods for accomplishing this were outside the scope of the Study.

Table 5.3.
Summary of Water/Chloride Balances

Water/Chloride Balance Component	VM1 - Conventional	VM1 - ILF	VM1 - CFTS	VM1 - CT	VM1 - FTTD	VM3 - Conventional	VM3 - ILF	VM3 - CFTS
Free Water Volume (Mm^3)	46.4	33.8	27.3	32.4	12.5	89.7	48.0	41.4
Free Water Cl Concentration - high scenario (g/L)	1.3	1.6	2.5	2.2	2.0	1.1	2.2	3.1
Free Water Cl Concentration - low scenario (g/L)	0.2	0.3	0.4	0.3	0.3	0.2	0.4	0.5
Free Water Cl Mass - high scenario (kT)	58.4	54.3	68.4	69.9	25.5	99.7	104.4	129.0
Free Water Cl Mass - low scenario (kT)	9.7	8.7	11.4	11.3	4.3	16.6	17.4	21.5
Coarse Tailings Pore Water Volume (Mm^3)	639.6	639.6	639.6	373.9	639.6	666.8	666.8	666.8
Coarse Tailings Cl Concentration - high scenario (g/L)	1.1	1.4	1.7	2.0	1.7	1.0	1.5	1.5
Coarse Tailings Cl Concentration - low scenario (g/L)	0.2	0.2	0.3	0.3	0.3	0.2	0.2	0.3
Coarse Tailings Cl Mass - high scenario (kT)	707.1	882.5	1081.0	736.2	1106.6	634.1	996.8	1009.7
Coarse Tailings Cl Mass - low scenario (kT)	117.9	153.6	180.5	123.2	184.7	105.7	166.3	168.4
FFT Pore Water Volume (Mm^3)	502.0	239.3	50.3	234.5	45.6	692.1	200.8	128.6
FFT Cl Concentration - high scenario (g/L)	1.1	1.4	1.2	2.1	2.4	1.0	1.1	0.6
FFT Cl Concentration - low scenario (g/L)	0.2	0.2	0.2	0.4	0.4	0.2	0.2	0.1
FFT Cl Mass - high scenario (kT)	566.2	330.1	60.5	502.5	108.9	674.8	216.8	77.6
FFT Cl Mass - low scenario (kT)	94.4	57.5	9.9	83.5	18.2	112.5	35.9	12.9
Engineered Tailings Volume (Mm3)	NA	243.8	107.8	251.0	48.0	NA	295.0	152.9
Engineered Tailings Cl mass - high scenario (kT)	NA	64.3	121.4	22.6	90.3	NA	90.0	191.7
Engineered Tailings Cl mass - low scenario (kT)	NA	2.1	20.1	3.9	14.8	NA	15.1	32.0

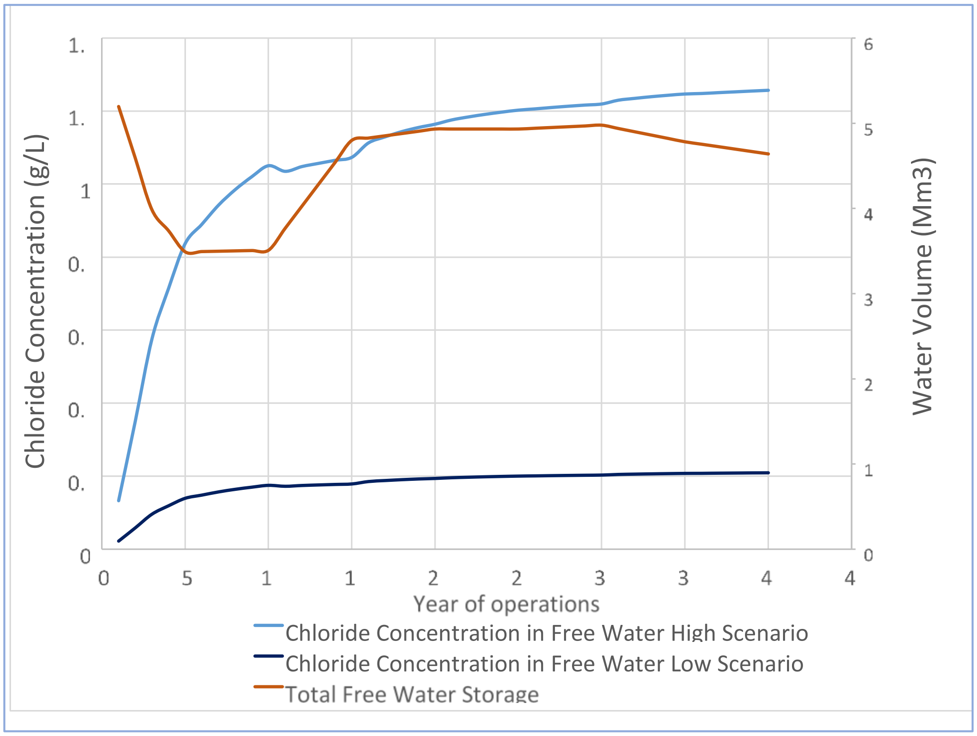


Figure 5-10. VM 1 Conventional Free Water Volume and Chloride Concentrations

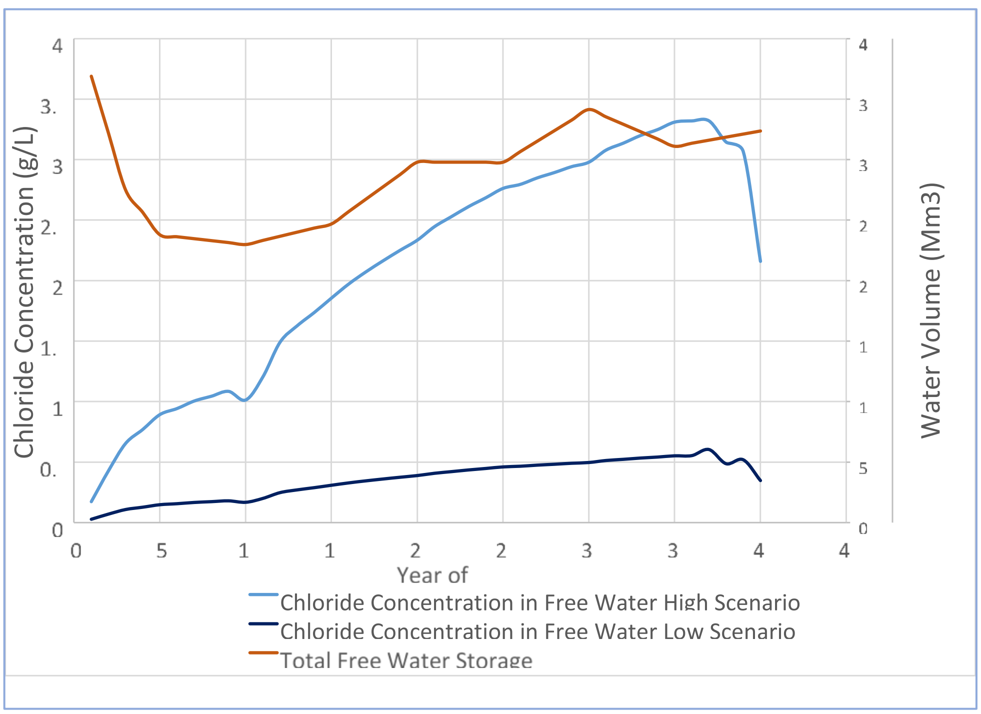


Figure 5-11. VM1 CT Free Water Volume and Chloride Concentration
(Note the scale difference from the previous chart.)

5.4.2. Excess Water Situations

As the area of disturbance grows with the advance of site mining, it is common that sites reach a state where there is excess recycle water. That is, more fresh water is imported into the recycle water system than required to maintain a clear water layer above the settling pond mudline – typically 3 metres. For this Study, a key water management assumption was that imported water would be the minimum to ensure a 3-m clarified water cap over the FFT mudline in the recycle water settling pond.

The impact of storing excess water varies with the volume and timing of the case-by-case storage requirement. For some of the cases in this Study, excess water could be incorporated into the plans with relatively small impacts to the plan and costs. However, other scenarios may require a complete redesign of the case. For example, the VM1 FTTD Case was designed to store minimum fluid volumes but it also provided the most flexibility to construct additional containment structures if required. As another example, the VM1 CT Case already incorporated close to the practical maximum dyke construction volume and it could be very difficult to provide additional water storage capacity within the first 30 to 35 years of the operations. The ability to discharge water during mine operation would avoid containment requirements for excess water and control the buildup of chloride (salt) concentrations assisting timely closure.

5.4.3. EPL Outlet Elevations

The importance of the assumptions related to EPL outlet elevation mentioned previously in this report is a critical topic that should be examined further in a regional context. EPL outlet elevation control has the potential to greatly influence the attractiveness and feasibility of tailings technologies for different mine sites.

5.5. Fines Release Rate

A base assumption for all cases except for the VM1 CT Case was that 45% of ore fines would be released to form FFT annually. However, there have been reports in the industry of average release rates as high as 55% and as low as 30%. Figure 5-12 shows the approximate impacts of that potential range of ore fines release rates on the different fines deposits types included in this Study, with the Base Case values used for this Study shown by the red diamonds.

In terms of potential impacts to estimated costs, the cases expected to be most affected by a change in the assumption regarding rate of ore fines release would be those with the highest unit costs for engineered tailings - in order of reducing unit costs: FTTD, CFTS, ILF, and CT. For some cases, especially the VM1 Conv and ILF Cases and the VM3 Conv Case, a much different rate of fines release could require substantial changes to the mine plan since re-handle volumes, dyke requirements, FFT processing rates, etc., could change dramatically.

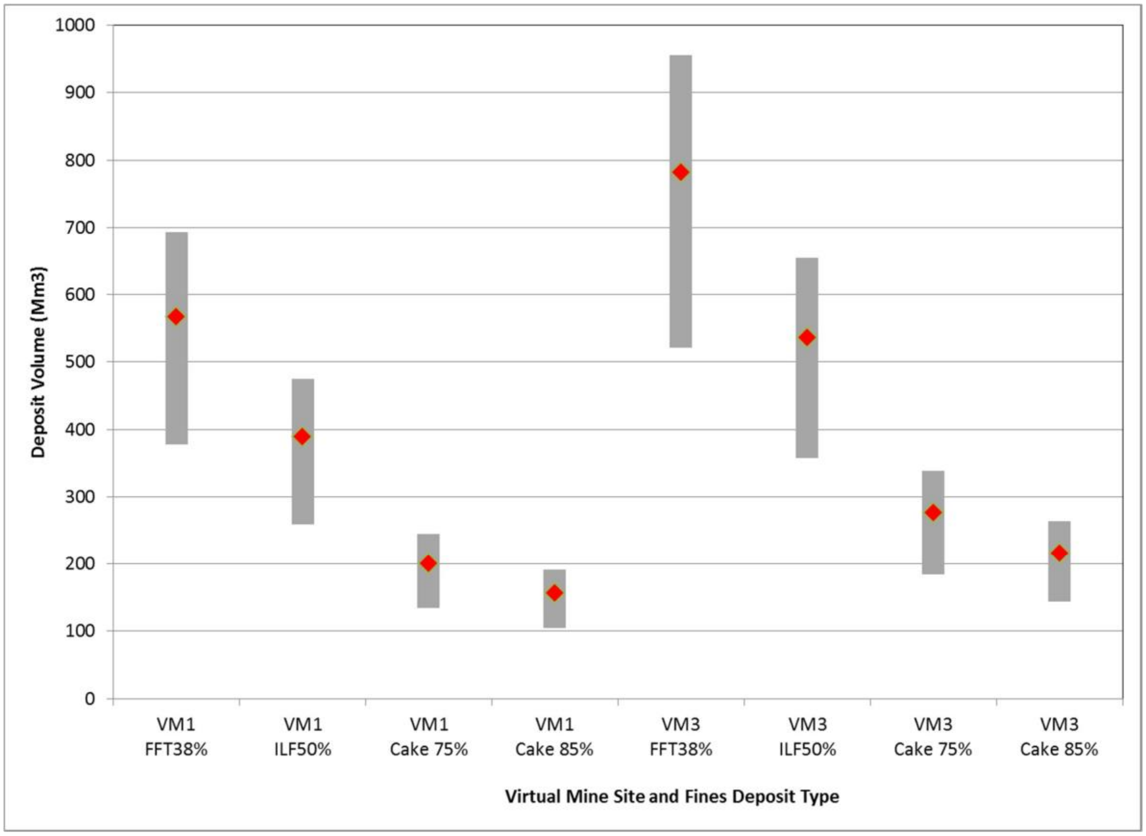


Figure 5-12. Fines Deposit Sensitivity to Ore Fines Release Rate

5.6. Single-Pit versus Multiple-Pit Scenarios

This Study developed and assessed mine and tailings plans for virtual mines with two different sets of assumed site conditions. Each mine site was treated independently. In reality, some mine projects have developed multiple pits, and some have leases in close proximity, making use of an exhausted mine pit for disposal of tailings from the next mine area technically feasible and highly advantageous in terms of both cost and environmental impacts.

Figure 5-13 shows the pit outline for several oil sands mining operations. Unlike the single rectangular mine pits used in this Study, some sites have multiple pits or irregular pit shapes. Depending upon the mine sequence, these situations present an opportunity to dispose of fine tailings in pit early in the mine life. This is significant in that it could avoid excessive re-handle and the need for high-capacity dewatering process facilities. Additionally, there is less engineered containment requirement. This illustrates the importance of the actual site conditions, which can influence tailings technology selection.

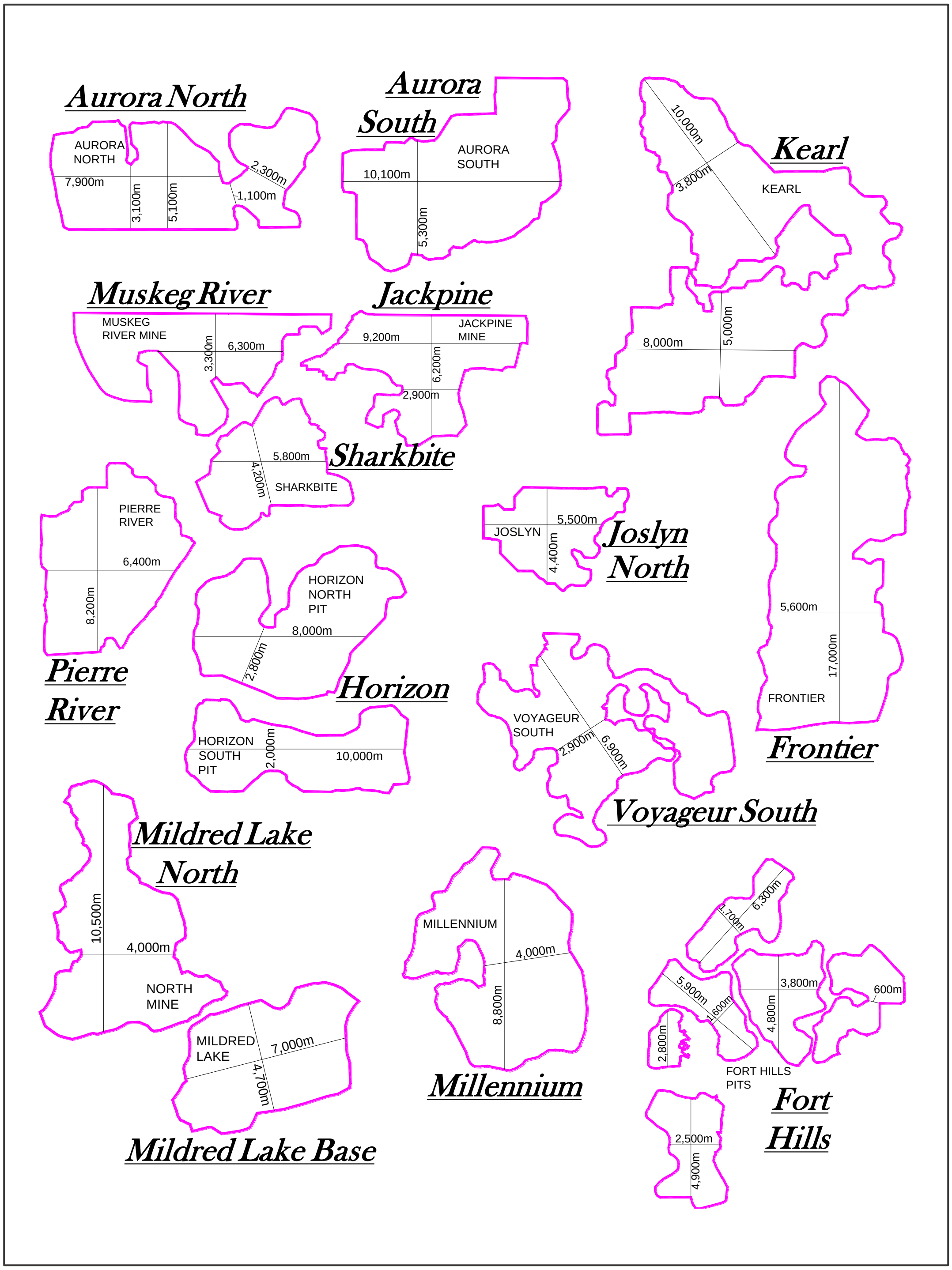


Figure 5-13. Pit Geometry for Several Oil Sands Mining Sites

6. REFERENCES

COSIA, 2012. Technical Guide for Fluid Fine Tailings Management. August 2012.

COSIA, 2014. Guidelines for Performance Management of Oil Sands Fluid Fine Tailings Deposits to Meet Closure Commitments. February 2014.

Government of Alberta, 2015. Tailings Management Framework for Mineable Athabasca Oil Sands (TMF), March 13, 2015.

Kemp, I. C. (2012) Fundamentals of Energy Analysis of Dryers, in Modern Drying Technology: Energy Savings, Volume 4 (eds E. Tsotsas and A. S. Mujumdar), Wiley-VCH Verlag GmbH & Co.

MacKinnon, M., and Sethi, A. 1993. A Comparison of the Physical and Chemical Properties of the Tailings Ponds at the Syncrude and Suncor Oil Sands Plants. AOSTRA. Proceedings of the Fine Tailings Symposium: Oil Sands - Our Petroleum Future Conference, April 4-7, 1993, Edmonton, Alberta.