

# Integration of Mine-to-Mill Production Planning Strategy for Oil Sands Mining and Waste Disposal

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## ABSTRACT

*For oil sands mining, the production schedule must be integrated simultaneously with in-pit and ex-pit dyke construction scheduling. In-pit dykes are constructed in the mined-out areas concurrently with the advancement of the pit phase mining. The mined ore that exceeds the plant capacity will be stockpiled for a limited duration. The topmost layer of the overburden will be used for land reclamation. Organic Rich Solids (ORSs) which represents about 5 wt% of the total ore and negatively affect the bitumen recovery will be used as a predictor for ore processability. In this research, a theoretical and conceptual mine planning framework based on Mixed Integer Linear Goal Programming (MILGP) for oil sands production scheduling and waste management is presented. New robust constraints that control the annual tonnage fluctuation for material mined and processed over the mine life are introduced in the model. The model generates an integrated mine plan with a waste management and stockpiling strategy over the mine life that maximizes the Net Present Value (NPV) of the operation. The model also integrates a mine-to-mill production planning strategy that uses ORSs content during optimization.*

## 1. Introduction

Extracting mining blocks from an open pit mine in a specific sequence that maximizes the NPV is known as open pit mine planning optimization. Open pit mine planning optimization aims to provide the plant with ore at full capacity which is subject to a variety of production, grade blending and pit slope constraints (Whittle, 1989). A major aspect of mine planning is optimizing the long term production planning. The very first and highly important step in the mine planning process is modeling the ore body appropriately. All other activities throughout the mine life starting from evaluating the economic viability of the entire mining operation to undertaking all the processes of mine planning will be based on the ore body model (Hustrulid and Kuchta, 2006). The main phases of the mine planning process are: 1) block model determination; 2) ultimate pit limit (UPL) definition; and 3) production planning (Chicoisne, et al., 2012).

The focus of this research is the integration of waste management into Long-Term Production Planning (LTPP) optimization as required by recent regulations from Alberta Energy Regulator (AER) Directive 085 (Alberta Energy Regulator, 2017). Oil sands mining of the McMurray formation is studied and used as a case study. The McMurray formation is the largest deposit in the world and mostly located near the surface. The Pleistocene unit is the topmost layer of the deposit. It contains muskeg (reclamation material), which is comprised mainly of organic matter. The Clearwater formation overlying the McMurray formation is comprised of marine clay, fine sand and siltstone. Both the Pleistocene and Clearwater formation known as overburden. The McMurray formation contains bitumen, the element of interest. It is informally subdivided into Upper, Middle and Lower formations based on the environment of sediments deposition. Devonian carbonates mark

the end of the oil sands deposit (Masliyah, 2010). Fig. 1 shows a sketch of the vertical soil profile for a typical oil sands formation.

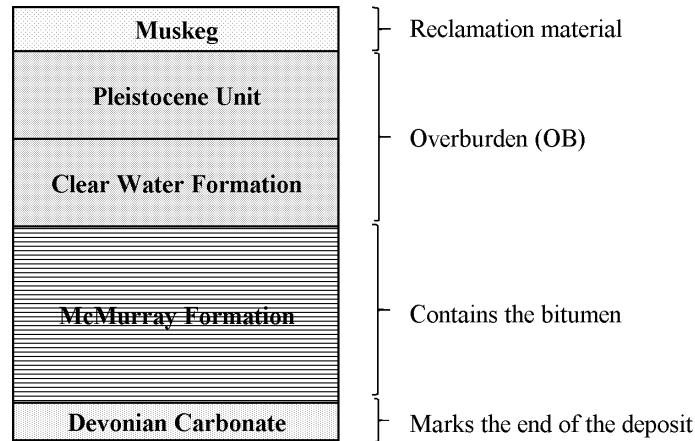


Fig. 1. Schematic view of the vertical soil profile for a typical oil sands formation.

Oil sands mining operations result in different types of material: ore, interburden (IB), overburden (OB), reclamation material (RM) and waste. Material with a bitumen grade of 7% or more will be classified as ore according to Directive 082 (Alberta Energy Regulator, 2016). Air flotation technique is used to separate bitumen from the fines. Processing the ore results in a huge amount of a mixture of water, fine materials, sands and residual bitumen known as tailings, the most unwanted by-product of oil sands processing (Masliyah, 2010). Using the hydro-cyclone, tailings is classified into Tailings Coarse Sands (TCS ) and Fine Sands or slurry (Kalantari, et al., 2013). TCS is used for dyke construction while tailings slurry is deposited in the disposal areas created with dykes.

Any ore material that has a bitumen grade less than 7%, known as interburden, will be reclassified based on the fines content. Material with fines content less than 50% will be used for dyke construction; otherwise, it will be sent to the waste dump. Overburden (OB) will be used either for roads or dyke construction if it meets the fines requirement. Muskeg will be stockpiled and used to reclaim the land at the end of mine life. Any material that does not meet the requirements of ore, dyke materials or reclamation material is classified as waste and will be sent to the waste dump.

Waste management requires special geotechnical considerations and tailings management techniques that may lead to economic liabilities and delayed reclamation if not well managed (Boratynec, 2003; Ben-Awuah and Askari-Nasab, 2013; Azam and Scott, 2005). There are three significant aspects in dealing with oil sands tailings. Firstly, the greenhouse gas emissions resulting from the Clark Hot Water Extraction (CHWE) process (Devenny, 2009). Secondly, the environmental challenges due to the toxicity of the tailings resulting in the contamination of the fresh water table by polluted tailings' water leaks. Thirdly, space limitations increase the need for in-pit tailings containment, and storage space since more mining processes lead to additional volume of tailings slurry (Devenny, 2009).

Presently, plans for tailings deposition and mine reclamation are prepared after the optimization of the long-term mine production plans (Ben-Awuah and Askari-Nasab, 2013). Directive 085 (Alberta Energy Regulator, 2017) issued by the Alberta Energy Regulator (AER) requires oil sands operators to periodically publish their waste disposal and tailings plans publicly (McFadyen, 2008; Ben-Awuah and Askari-Nasab, 2013). Additional documentation on oil sands solid waste and tailings management can be found in Ben-Awuah, et al., (2012); Badiozamani and Askari-Nasab, (2014); Badiozamani and Askari-Nasab, (2016). Fig. 2 shows a conceptual mining model, which includes an oil sands deposit area to be mined and simultaneously used as an in-pit tailings storage facility. As mining advances in the specified direction, the in-pit tailings dyke footprints are released for dyke construction Fig. 2.

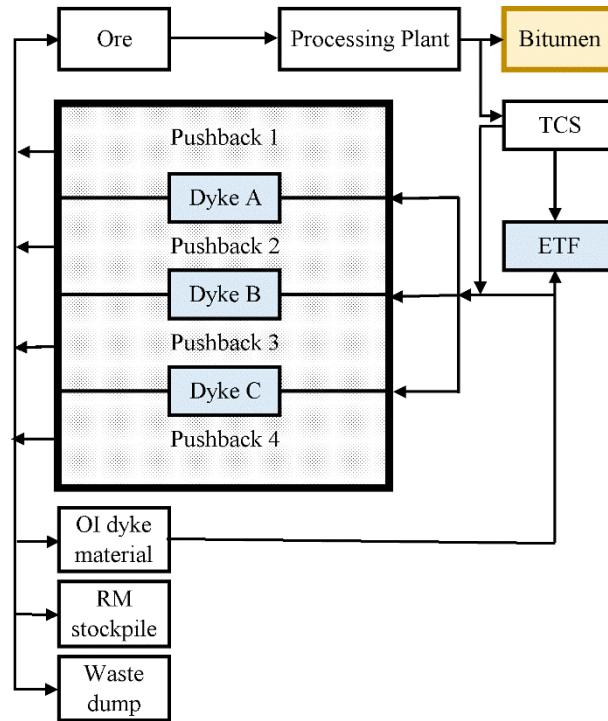


Fig. 2. Conceptual model for mining and waste management strategy modified after Ben-Awuah and Askari-Nasab (2013).

A schematic representation of the problem definition is presented in Fig. 3. The final pit block model is divided into pushbacks. The material intersecting a pushback and a bench is known as a mining-panel. Each mining-panel contains a set of mining-cuts and is used to control the mine production operation sequencing. Mining-cuts are clusters of blocks within the same mining bench that are similar in terms of location, grade, rock type and the shape of mining-cuts created on the lower bench. The figure depicts the scheduling of an oil sands ultimate pit block model containing  $K$  mining-cuts within  $P$  mining-panels. Each mining-cut  $k$ , could be made up of one or more of the following materials: ore,  $O_k$ , interburden and overburden dyke materials,  $ib_k$  and  $ob_k$ , muskeg reclamation material,  $mu_k$ , and waste,  $w_k$ .

The material in each mining-cut is to be scheduled over  $T$  periods based on the goals and constraints associated with the mining operation. The mined ore extracted from mining-cut  $k$  within mining-panel  $p$  in period  $t$  will be sent to the processing destination  $a$ . Any material that exceeds the processing capacity will be sent to the stockpile  $sp$  in period  $t$  and will be reclaimed in period  $t + ts$ , where  $ts$  is the stockpiling duration limit controlled by the planner to minimize oxidation of the stockpiled material that reduces processing recovery. The ore extracted in the current period  $t$  and the ore that has been sent to the stockpile in period  $t - ts$  together will be sent to the processing destination to extract the bitumen. The generated  $TCS$  material together with the  $OB$  and  $IB$  dyke materials will be used for constructing  $dyke_i$  at site  $i$ . Reclamation material will be sent to the reclamation material stockpile area. This is referred to as operational material scheduling.

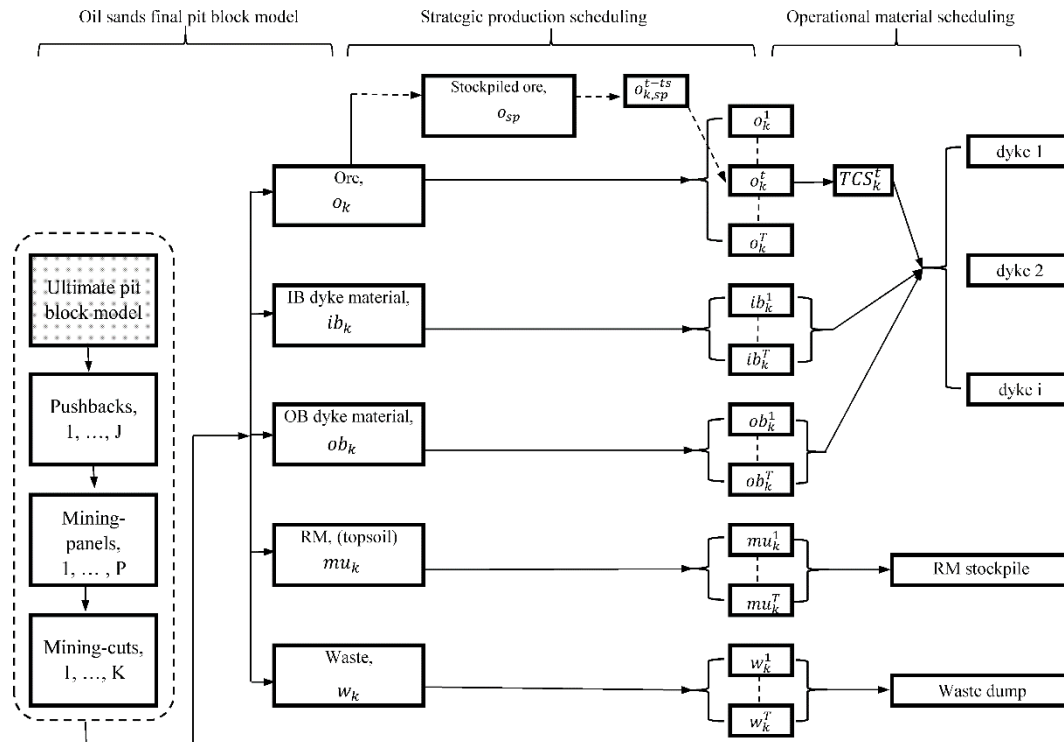


Fig. 3. Schematic representation of the problem definition showing strategic production and operational material scheduling.

These strategic and operational schedules to be developed are subject to a variety of economic, technical and physical constraints. The constraints control the mining extraction sequence, annual fluctuation of the tonnage mined and ore processed, and ore and dyke material blending requirements. The constraints also control reclamation and dyke material goals that specify the quantities of allowable material for reclamation works and dyke construction. The strategic and operational schedules determine the profitability and sustainability of the project. The schedules control the NPV of the operation and enable a robust waste management planning strategy. Improper waste management planning can lead to environmental issues, resulting in immediate mine closure by regulatory agencies.

The research seeks to develop a mine planning theoretical framework that maximizes the NPV of an oil sands mining operation and minimizes waste management cost using Mixed Integer Linear Goal Programming (MILGP) model. The model incorporates multiple material types with multiple elements for multiple destinations in oil sands long-term production planning. The proposed MILGP model aims to generate:

- A strategic schedule that determines the sequence of extracting ore, reclamation material, overburden and interburden from a predefined ultimate pit limit over the life of mine to maximize the NPV of the project;
- An operational schedule that determines the destination of reclamation material to minimize the extra mining cost and the destination of dyke materials to minimize dyke construction costs;
- A reclamation strategy for the ore that is stockpiled for a limited duration to reduce oxidation;
- A mine-to-mill production planning strategy that uses Organic Rich Solids content (ORs) during optimization;

The next section of this research covers a summary of the literature review on LTPP optimization problems based on deterministic approaches, clustering and paneling in mine planning, and

stockpiling. Section 3 gives details of ORS definition and calculations and Section 4 highlights the theoretical mathematical programming formulation. Section 5 explains the implementation of the MILGP model research methodology. A case study is presented in Section 6. Section 7 documents the research conclusions.

## 2. Summary of Literature Review

Long-term production planning (LTPP) focuses mainly on ore reserves, stripping ratio and major annual investment plans (Newman, et al., 2010). The geologic block model is the backbone of open pit mine design and scheduling processes. Assigning the geological characteristics of each block and their grade can be done using available estimation techniques. Using financial and metallurgical data, the economic value of each block is also calculated (Osanloo, et al., 2008). In the literature, the LTPP and scheduling methods are divided into deterministic algorithms that consider the input values and parameters as known and fixed, and uncertainty-based algorithms that consider some input parameters as uncertain. Since the 1960s, researchers have studied and applied Mathematical Programming Models (MPMs) for mine production scheduling. A variety of exact optimization methods including Linear Programming (LP), Integer Programming (IP), and Mixed Integer Linear Programming (MILP) are commonly used in addition to Dynamic Programming (DP) and Goal Programming (GP) (Osanloo, et al., 2008).

Johnson (1969) introduced linear programming as a MPM to the mine planning research area. The author's model was for a long-term multi-destination open pit production planning problem. The results were not optimum and the size of the problem was computationally intractable. Subsequently, the initial LP model was modified by Gershon (1983) and Dagdelen (1985) to a MILP model. The authors considered a set of binary variables to satisfy the precedence of block extraction. The modified models could handle multiple ore processing options and multiple grades. However, their formulations could not ensure feasible solutions for all cases. Also, the number of binary variables makes the model intractable for real-size mine planning projects and difficult to be solved with the current state of hardware and software. Other researchers proposed some methods to solve IP models. Dagdelen and Johnson (1986) used the Lagrangian relaxation and sub-gradient optimization algorithm approach. Akaike and Dagdelen (1999) used 4D-network relaxation and sub-gradient optimization algorithm approach. Caccetta and Hill (2003) used branch-and-cut approach which is a combination of cutting plane and branch-and-bound algorithms. They solved sequences of linear programming relaxations of the IP problem by reducing the size of the problem prior to optimization. None of the abovementioned methods to solve IP models could be used for large-scale problems with dynamic cut-off grades.

Ramazan and Dimitrakopoulos (2004a) developed MILP formulations to reduce the number of binary variables and solution times. They set certain variables as binary and others as continuous. Their model resulted in partial mining of blocks that have the same ore value affecting the NPV generated. An MILP model was also developed by Ramazan and Johnson (2005) and Ramazan (2007) based on an aggregation method to reduce the number of integer variables in scheduling. The model was solved based on fundamental tree algorithm. However, it did not guarantee a global optimum solution of the problem. Caccetta and Hill (2003) developed an MILP model and Boland, et al. (2009) developed an LP model to generate mine production schedules with block processing selectivity. They did not provide enough information on the generated schedules to evaluate the practicality of the solutions. Askari-Nasab, et al. (2011) developed MILP models that use block clustering techniques. The models use a combination of continuous and binary integer variables and were applied to a large-scale problem. The portion of a block to be mined is controlled by continuous variables while binary integer variables control the extraction sequence of the blocks. The authors stated that they successfully implemented the models for some basic large-scale production scheduling problems. A Dynamic Programming (DP) model that maximizes the NPV, subject to production and processing constraints was presented by Osanloo, et al. (2008). This model

considers both the time value of money and block sequencing to determine the UPL. However, it cannot be applied to large-scale problems and there is no guarantee that mining and processing constraints will be satisfied. Based on a combination of heuristics and DP, Newman, et al. (2010) proposed their methodology. They state that the ultimate pit limits, the cut-off grade, the mining sequence and production scheduling are related to each other and without the knowledge of one variable, the next variable could not be determined. Their method provides the required simultaneous solution for the variables of the problem. In general, applying MPMs to the LTPP result in large scale optimization problems with many integer and continuous variables which are difficult to solve with the available software and hardware and might need lengthy solution time. The efforts that have been made in reducing the solution time were inefficient for large-scale problems or could not generate integrated practical mining strategies.

One of the deterministic approaches used to solve long-term production planning and scheduling problems is Goal Programming (GP). It is a popular deterministic approach for solving multiple objective optimization problems. The main idea of GP is that the optimizer provides results for the objective very close to the required goals, regardless of whether the goals are achievable or not. GP minimizes the deviations between the target values of the objectives and the satisfying solution (Orumie and Ebong, 2014). GP was used by Chanda and Dagdelen (1995) for the mine planning of a coal deposit. Due to goal functions interactions involved in solving the problem, the optimal solution could not always be achieved. Ben-Awuah and Askari-Nasab (2013) formulated the oil sands' long-term production scheduling and waste disposal planning problem using a combination of MILP and GP formulations. The hybrid termed as Mixed Integer Linear Goal Programming (MILGP) has an objective function, goal functions for mining, processing and dyke construction and constraints. These goals are prioritized according to the impact of a deviation from their targets on the entire mining operation. The authors stated that using MILGP is appropriate for their framework because, based on the importance of the goals, the MILGP structure will allow the planner to achieve some goals while others are traded off. In other words, the model allows for flexible formulation and the specification of priorities among goals. According to the authors, solutions with known optimality limits are generated when using exact solution methods for LTPP problems. For the resulting production schedule, a higher NPV is achieved as the solution gets closer to optimality.

In LTPP, the size of the problem grows exponentially as the number of blocks increases resulting in insufficient computer memory during optimization. More blocks mean more decision variables are used to control the block extraction precedence. Researchers have tried to classify the large amount of data into relatively few classes of similar entities (cluster) by maximizing both intra-cluster similarity and inter-cluster dissimilarity. This classification is known as aggregation or clustering. Applying clustering is an efficient way of dealing with this problem. Clustering will minimize the number of integer decision variables as well as maintaining the minimum mining width for large mining equipment (Askari-Nasab and Awuah-Offei, 2009). Boland, et al. (2009) proposed a solution procedure based on using aggregated blocks for the order of extraction decisions while individual blocks are used for processing decisions. This reduces the number of integer variables in the optimization problem that leads to reduced solution times.

Clustering algorithms can be categorized into hierarchical clustering, partitional clustering or overlapping clustering (Tabesh and Askari-Nasab, 2011). In mine planning, only hierarchical and partitional clustering can be used because all blocks must belong to a single cluster. For this reason, Tabesh and Askari-Nasab (2011) reviewed different clustering algorithms and developed a new clustering approach more suitable to the mining industry. Paneling is another technique that has been introduced in production scheduling to maintain practical mining widths and reduce the size of the optimization problem. The intersections of pushbacks and mining benches generate mining panels (Ben-Awuah and Askari-Nasab, 2013). Each mining-panel contains a set of mining-cuts and is used to control the mine production operation sequencing. For this research, hierarchical clustering algorithm developed by Tabesh and Askari-Nasab (2011) was used.

In the implementation of most LP and MILP models, the material flow post-extraction is not considered (Moreno, et al., 2017). “In particular, the use of stockpiling to manage processing plant capacity, and the interplay of material flows from the mine to a stockpile, the mine to a processing plant, and a stockpile to a plant, have not been treated as an integrated part of mine extraction sequence optimization” (Moreno, et al., 2017). Stockpiling can be used in mine operations for many reasons such as the blending of material, storage of overproduced ore or low-grade ore for future processing, and storage of waste material for reclamation purposes. Gemcom Software International (2015b) has a stockpiling module that considers mixing material with different grades in the stockpile. However, it does not use optimization techniques to model the stockpile, so there is no guarantee of obtaining an optimal solution with respect to the number of stockpiles and/or the grade contained in each stockpile (Moreno, et al., 2017).

Smith and Dimitrakopoulos (1999) used mixed integer programming to solve a short-term production scheduling problem with blending, considering stockpiles both at the mine and mill. They noted that it requires nonlinear constructs to correctly capture the contents of the stockpile. Ramazan and Dimitrakopoulos (2013) used a stochastic framework to incorporate stockpiling. In their model, the authors ignored the mixing of material in the stockpile. Asad (2005) cautioned that long-term stockpiling could result in problems such as leaching, deterioration of material and oxidation, which might result in poor recovery in the treatment process. For oil sands mining, the stockpiled material must also be processed within a limited duration to avoid oxidation that affects the efficiency of the processing recovery process. For this research, the stockpiling duration is limited to a maximum of two years to ensure there is no significant effect on the ore recovery.

### **3. Oil Sands Properties and Organic Rich Solids Definition**

In oil sands mining, Clark Hot Water Extraction process (CHWE) is used to recover bitumen from the ore deposit. HWEP depends on the surface characterization of solid particle in the ore matrix. Measurements of fines ( $< 45\mu\text{m}$ ) used to predict the processability of the ore, however is not always effective. It has been found that certain solid fractions known as Organic Rich Solids (ORSs) still exist even after the treatment of oil sands by multiple extraction toluene. These ORSs comprise about 5% of the total ore, and potentially affect the processability of oil sands (O'Carroll, 2002; Sparks, et al., 2003). During the bitumen separation process, the ORS carry any associated bitumen into the aqueous tailings, thus reducing overall bitumen recovery. In this sense, these solids are considered to be active and their associated quantity per ore can be estimated, thus, a better predictor for ore processability than the traditional use of bitumen ore fines contents (O'Carroll, 2002 ; Sparks, et al., 2003).

Oil sands analysis shown that the main components are bitumen, water and solids (Masliyah, 2010; O'Carroll, 2002 ). Solids are further classified into fines, ultra-fines, organic rich solids, clays and mineralogical composition. O'Carroll (2002) noted that loss in bitumen recovery is associated with higher ORS content in the ore. The author calculated bitumen to ORS ratio for each sample and reported that the ratio increases with higher bitumen content and further plotted the primary bitumen recovery against the BIT:ORS ratio. After a raising trend, the recovery levels off at 90% for BIT:ORS ratio values of 20 and more. Based on this result, the author concluded that the ratio has potential for use as an index in the characterization of oil sands ores.

### **4. Recovery Calculations**

#### **4.1. Recovery calculations based on Directive 082: Alberta Energy Regulator**

For oil sands mining, the recovered volume of bitumen from the mining and processing operations, is specified by Directive 082 from (Alberta Energy Regulator, 2016). One of the four required

operating criteria for oil sands mining based on Directive 082 is the recovery of the processing plant which is uncertain and changes based on the average bitumen grade as defined by AER.

The recovery equals to 90% if the average bitumen content of the as-mined ore is 11% bitumen or greater. If the average bitumen content of the as-mined ore is less than 11 weight per cent bitumen, recovery is determined by Eq. (1), (Alberta Energy Regulator, 2016), where BIT is the average weight per cent bitumen content of the as-mined ore:

$$RECOV_{AER} = -2.5 * (BIT)^2 + 54.1 * (BIT) - 202.7 \quad (1)$$

#### 4.2. Recovery calculations based on Organic Rich Solids

To calculate the ORS and BIT:ORS ratio for our dataset the following steps are done:

- BIT against ORS is plotted For O'Carroll's samples and an exponential correlation exists as can be seen in Eq. (2) with the coefficient of determination (R-squared) of 62.6%. (R-squared is a statistical measure of how close the data are to the fitted regression line. It is also known as the coefficient of determination).
- For this small dataset, bitumen, water and fines were given. The rest considered as solids (Solids = 100% - (bitumen% + water% + Fines%)).
- Solids for our dataset contain organic rich solids, ultra-fines and other solids. For each block in the ultimate pit block model ORS is calculated using Eq. (2). ORS ranging from 0.00 to 2.60% with the average of 0.88%.
- For O'Carroll's samples primary bitumen recovery is plotted against BIT:ORS ratio, a raising trend is noted. Recovery levels off at 90% for BIT:ORS ratio values of 20 and more. The relationship is modeled as seen in Eq. (3).
- BIT to ORS ratio is calculated for each block in our dataset. It covers the range from 0.00 to 17.69 with the average of 4.1.
- Eq. (3) is used to calculate the recovery for each block based on ORS content.

$$ORS = 3.8145 \times e^{-0.094 \times Bit} \quad (2)$$

$$RECOV_{ORS} = -0.0219 \times \left(\frac{BIT}{ORS}\right)^2 + 0.3335 \times \left(\frac{BIT}{ORS}\right) - 0.3789 \quad (3)$$

- The recovery is used to calculate the revenue.
- Bitumen recovery is plotted against BIT:ORS ratio. Recovery levels off at 90% for BIT:ORS ratio values of 8 and more (Fig. 4).

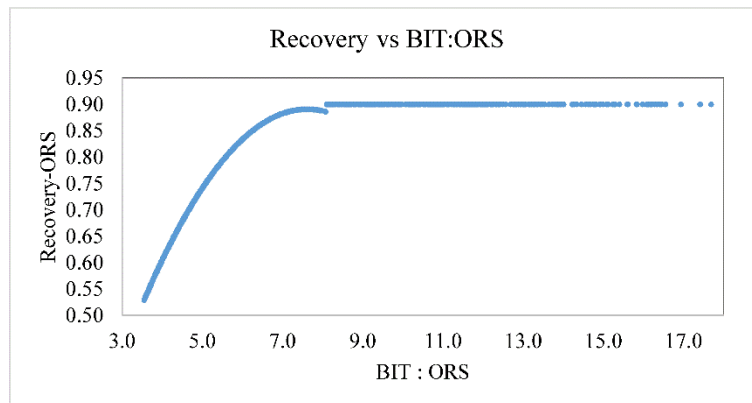


Fig. 4. Recovery vs. BIT to ORS ratio.

Subsequently, a case study with two different scenarios is examined using the proposed MILGP model. Both scenarios use tonnage fluctuation constraints to achieve the mining and processing



targets as part of the production scheduling optimization process with a limited duration stockpiling strategy for ore. However, the first scenario uses recovery calculated from AER to determine the revenue while the second uses recovery calculated from ORS. MATLAB (Mathworks, 2017) is used for coding the mathematical programming formulation and the resulting optimization problem is solved with a large-scale optimization solver IBM/CPLEX (ILOG, 2012). This solver uses a branch and cut algorithm which is a hybrid of branch-and-bound algorithm and cutting plane methods to solve the optimization problem (Horst and Hoang, 1996; Wolsey, 1998). The role of ORS on the ore processability comparing to the recovery calculations recommended by AER is investigated.

## 5. MILGP Theoretical Model Formulation

The strategic production schedule considers the time and sequence of extracting the ore, muskeg, overburden, interburden and waste blocks, as well as their destinations from a predefined UPL. The proposed MILGP model is capable of considering multiple mining locations, multiple pushbacks and different types of materials. The stockpiled ore can be reclaimed after pit mining is completed or simultaneously during active pit mining with a pre-determined reclamation duration. However, long-term stockpiling could result in problems such as leaching, the deterioration of material and oxidation, which can affect the efficiency of the processing recovery.

For oil sands mining, to avoid the risk of oxidation, the ore will be reclaimed in a pre-determined period controlled by the planner. The proposed oil sands production scheduling model integrates waste management through dyke construction and stockpiling for a limited duration. Stockpiling is for the mined ore that exceeds the plant capacity in any given year. The MILGP model is subject to economic, technical and physical constraints that control the mining operation. It is assumed that: 1) when a mining-panel is scheduled, all the mining-cuts, blocks or parcels within this mining-panel are extracted uniformly; 2) when modeling the relationship between the mining-panels and mining-cuts, the planner has access to all the mining-cuts within each mining-panel; 3) the stockpiling strategy is considered in the optimization problem for extra ore that exceeds the mill capacity and there are stockpile bins available for each period; 4) the exact amount of ore sent to the stockpile in period will be reclaimed after the stockpiling duration controlled by the planner. The notations used in the formulation of the oil sands long-term production planning and waste management framework have been classified as indices, sets, parameters and decision variables. The details of these notations can be found in the list of nomenclature.

### 5.1. Modeling of economic mining-cut value

The notion of economic block value is based on ore parcels which could be mined selectively. The profit from mining a block is a function of the value of the block and the cost incurred in mining, processing and dyke construction. Based on the value of the mining-cut and the costs incurred during mining and processing operations, the discounted profit of each mining-cut equals to the discounted revenue obtained by selling the final product contained in mining-cut  $k$  minus the discounted costs. For a mining-cut, if there are valuable elements, its discounted economic value if it is sent from the mine to the processing plant ( $dm_k^{d,t}$ ) or from the stockpile to the processing plant ( $ds_{k,sp}^{d,t}$ ) is given by Eqs. (4) and (5), respectively.

$$dm_k^{d,t} = rm_k^{a,e,t} - dw_k^{l,t} - dmu_k^{d,t} - dob_k^{d,t} - dib_k^{d,t} - dt_k^{d,t} \quad (4)$$

$$ds_{k,sp}^{d,t} = rs_k^{a,e,t} - dw_k^{l,t-ts} - dmu_k^{d,t-ts} - dob_k^{d,t-ts} - dib_k^{d,t-ts} - dt_k^{d,t-ts} \quad (5)$$

Eqs. (6) to (12) define the parameters used in Eqs. (4) and (5). It should be mentioned that the recovery is calculated using two different approaches: i) based on AER; ii) based on ORS.

$$rm_k^{a,e,t} = \sum_{e=1}^E o_k \times g_k^e \times rp_{avg}^{a,e} \times (p^{e,t} - sc^{e,t}) - \sum_{e=1}^E o_k \times pc^{a,e,t} \quad (6)$$

$$rs_k^{a,e,t} = \sum_{e=1}^E o_k \times g_k^e \times rp_{avg,sp}^{a,e} \times (p^{e,t} - sc^{e,t}) - \sum_{e=1}^E o_k \times pc^{a,e,t} - \sum_{e=1}^E o_k \times pc_{sp}^{a,e,t} \quad (7)$$

$$dw_k^{l,t} = (o_k + mu_k + ob_k + ib_k + w_k) \times mc^{l,t} \quad (8)$$

$$dmu_k^{d,t} = mu_k \times muc^{d,t} \quad (9)$$

$$dob_k^{d,t} = ob_k \times obc^{d,t} \quad (10)$$

$$dib_k^{d,t} = ib_k \times ibc^{d,t} \quad (11)$$

$$dt_k^{d,t} = t_k \times tc^{d,t} \quad (12)$$

Where:

$a \in A, A = \{1, \dots, A\}$	Index for possible processing destination.
$d \in D, D = \{1, \dots, D\}$	Index for possible destinations for materials.
$e \in E, E = \{1, \dots, E\}$	Index for elements of interest in each mining-cut.
$j \in J, J = \{1, \dots, J\}$	Index for pushbacks.
$k \in K, K = \{1, \dots, K\}$	Index for mining-cuts.
$l \in L, L = \{1, \dots, L\}$	Index for possible mining locations (pits).
$p \in P, P = \{1, \dots, P\}$	Index for mining-panels.
$sp \in SP, SP = \{1, \dots, SP\}$	Index for possible stockpiles in the model.
$t \in T, T = \{1, \dots, T\}$	Index for the scheduling periods, years.
$ts \in TS, TS = \{1, \dots, TS\}$	Index for possible stockpiling durations, years.
$dm_k^{d,t}$	Discounted economic mining-cut value obtained by extracting mining-cut $k$ and sending it to destination $d$ in period $t$ .
$rm_k^{a,e,t}$	Discounted revenue obtained by selling the final products within mining-cut $k$ in period $t$ if it is sent to processing destination $a$ , minus the extra discounted cost of mining all the material in mining-cut $k$ as ore from location $l$ and processing at destination $d$ .
$dw_k^{l,t}$	Discounted cost of mining all the material in mining-cut $k$ in period $t$ as waste from location $l$ .
$dmu_k^{d,t}$	Extra discounted cost of mining all the material in mining-cut $k$ in period $t$ as muskeg reclamation material at destination $d$ .
$dob_k^{d,t}$	Extra discounted cost of mining all the material in mining-cut $k$ in period $t$ as overburden dyke material for dyke construction at destination $d$ .
$dib_k^{d,t}$	Extra discounted cost of mining all the material in mining-cut $k$ in period $t$ as interburden dyke material for dyke construction at destination $d$ .
$dt_k^{d,t}$	Extra discounted cost of mining all the material in mining-cut $k$ in period $t$ as tailings coarse sand dyke material for dyke construction at destination $d$ .

$ds_{k,sp}^{d,t}$	Discounted economic mining-cut value obtained by extracting mining-cut $k$ and sending it to stockpile $sp$ and reclaiming it to destination $d$ in period $t$ .
$rs_k^{a,e,t}$	Discounted revenue obtained by selling the final products within mining-cut $k$ from stockpile $sp$ in period $t$ if it is sent to destination $a$ in period $t$ , minus the extra discounted cost of processing and re-handling.
$dw_k^{l,t-ts}$	Discounted cost of mining all the material in mining-cut $k$ in period $t-ts$ as waste from location $l$ .
$dmu_k^{d,t-ts}$	Extra discounted cost of mining all the material in mining-cut $k$ in period $t-ts$ as muskeg reclamation material at destination $d$ .
$dob_k^{d,t-ts}$	Extra discounted cost of mining all the material in mining-cut $k$ in period $t-ts$ as overburden dyke material for dyke construction at destination $d$ .
$dib_k^{d,t-ts}$	Extra discounted cost of mining all the material in mining-cut $k$ in period $t-ts$ as interburden dyke material for dyke construction at destination $d$ .
$dt_k^{d,t-ts}$	Extra discounted cost of mining all the material in mining-cut $k$ in period $t-ts$ as tailings coarse sand dyke material for dyke construction at destination $d$ .
$o_k, o_p$	Ore tonnage in mining-cut $k$ and mining-panel $p$
$g_k^e$	The required average head grade of element $e$ in ore portion of mining-cut $k$ .
$rp_{avg}^{a,e}$	Proportion of element $e$ recovered (processing recovery) if it is sent from the mine to processing destination $a$ .
$rp_{avg,sp}^{a,e}$	Proportion of element $e$ recovered (processing recovery) if it is sent from the stockpile to processing destination $a$ .
$p^{e,t}$	The selling price of element $e$ in present value terms per unit of product.
$sc^{e,t}$	Selling cost of element $e$ in present value terms per unit of product.
$pc^{a,e,t}$	Extra cost in present value terms per tonne of ore for mining and processing at processing destination $a$ in period $t$ .
$pc_{sp}^{a,e,t}$	Extra cost in present value terms per tonne of ore for stockpiling at stockpile $sp$ and processing at destination $a$ in period $t$ .
$mu_k, mu_p$	Reclamation material tonnage in mining-cut $k$ and mining-panel $p$ .
$ob_k, ob_p$	OB dyke material tonnage in mining-cut $k$ and mining-panel $p$ .
$ib_k, ib_p$	IB dyke material tonnage in mining-cut $k$ and mining-panel $p$ .
$w_k, w_p$	Waste tonnage in mining-cut $k$ and mining-panel $p$ .
$mc^{l,t}$	Cost in present value terms of mining a tonne of waste in period $t$ from location $l$ .
$muc^{d,t}$	Cost in present value terms per tonne of RM at destination $d$ .
$obc^{d,t}$	Cost in present value terms per tonne of OB dyke material for dyke construction at destination $d$ in period $t$ .
$ibc^{d,t}$	Cost in present value terms per tonne of IB dyke material for dyke construction at destination $d$ in period $t$ .
$t_k, t_p$	TCS dyke material tonnage in mining-cut $k$ and mining-panel $p$ .

$tc^{d,t}$  Cost in present value terms per tonne of tailings coarse sand dyke material for dyke construction at destination  $d$  in period  $t$ .

## 5.2. MILGP objective function for optimizing production schedule

To develop the models, the concepts presented in Ben-Auwah and Askari-Nasab, (2013) are used as the starting point. The objective function of the MILGP model for oil sands long-term production planning and waste management that maximizes the NPV of the mining operation is developed using the continuous decision variables  $y_p^{l,t}$ ,  $x_k^{a,t}$  and  $c_{k,sp}^{a,t}$  to model mining, processing from mine and processing from stockpile respectively; for all mining locations and processing destinations. Continuous decision variables are used to allow for fractional extraction of mining-panels and mining-cuts in different periods for different locations and destinations. The objective function of the MILGP model for maximizing the NPV of the mining operation is stated in Eq. (13):

$$\text{Max} \sum_{l=1}^L \sum_{j=1}^J \sum_{d=1}^D \sum_{sp=1}^{SP} \sum_{a=1}^A \sum_{e=1}^E \sum_{t=1}^T \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} (rm_k^{a,e,t} \times x_k^{a,t} + rs_{k,sp}^{a,e,t} \times c_{k,sp}^{a,t} - dw_p^{l,t} \times y_p^{l,t}) \right) \quad (13)$$

Tonnage fluctuation constraints defined by Eqs. (14) and (15) are introduced in the proposed MILGP model to control the mining and processing targets. These constraints control the consecutive periodic fluctuation of the tonnage mined and tonnage processed. For material mined, the sum of deviations between two consecutive years should be less than or equal to a set deviation tonnage,  $D_m$ , allowed for mining. For material processed, the sum of deviations between two consecutive years should be less than or equal to a set deviation tonnage,  $D_p$ , allowed for processing. The mine planner controls the parameters  $D_m$  and  $D_p$ . The planner also controls when to start and finish applying these constraints. For instance, the planner might want to allow for a positive deviation in a couple of the first years and negative deviation in the last years. In other words, Eqs. (14) and (15) can be used for a controlled ramping up in the first years and ramping down in the last year. Eq. (15) is valid only when ore reclamation starts. Otherwise, set the ore reclamation variable in Eq. (15) to zero.

The importance of these special constraints is that, the planner does not need to set mining and processing targets. The optimizer uses the set periodic tonnage fluctuation parameters to determine the mining and processing targets. These constraints provide varying practical production schedule options for mine planners.

$$\sum_{t=1}^T \left( \sum_{p \in MP_j} (o_p + mu_p + ob_p + ib_p + w_p) \times (y_p^{l,t+1} - y_p^{l,t}) \right) \leq D_m \quad (14)$$

$$\sum_{t=1}^T \left( \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} \left( x_k^{a,t+1} + \sum_{sp=1}^{SP} c_{k,sp}^{a,(t-ts)+1} \right) - \left( x_k^{a,t} + \sum_{sp=1}^{SP} c_{k,sp}^{a,t-ts} \right) \right) \times o_k \right) \leq D_p \quad (15)$$

Constraints that control ore stockpile tonnages are presented in Eqs. (16) and (17). These equations control the amount of ore sent from mining-cut  $k$  to stockpile  $sp$  in period  $t$ . The continuous decision variable  $s_k^{sp,t}$  is used to model the ore sent to the stockpile. Material sent to the stockpile in period  $t$  are reclaimed in period  $t + ts$ , where  $ts$  is the stockpiling duration controlled by the planner. The planner also controls the upper and lower capacity limits for stockpile bins.

$$\sum_{sp=1}^{SP} \sum_{k=1}^K o_k \times s_k^{sp,t} \leq \overline{OS}^{sp,t} \quad (16)$$

$$\sum_{sp=1}^{SP} \sum_{k=1}^K o_k \times s_k^{sp,t} \geq \underline{OS}^{sp,t} \quad (17)$$

The MILGP bitumen and fines grade blending constraints ensure that the quality requirements of the processing plant, stockpile and dyke construction destinations are achieved. These constraints are formulated using Eqs. (18) to (25). Ore bitumen grade blending constraints ensure the extracted ore from mining-cut  $k$  within mining-panel  $p$  sent to either processing destination  $a$  or to stockpile  $sp$  in period  $t$  meets the grade quality requirements. Ore bitumen grade blending constraints are formulated using Eqs. (18) to (21). Eqs. (18) and (19) represent inequality constraints that control the limiting ore bitumen grade sent from the mine and stockpile to the processing plant. Eqs. (20) and (21) represent inequality constraints that control the limiting ore bitumen grade sent from the mine to the stockpile.

$$\sum_{p=1}^P \sum_{k \in MP_p} g_k^e \times o_k \times (x_k^{a,t} + c_{k,sp}^{a,t-ts}) - \overline{g}^{a,e,t} \sum_{p=1}^P \sum_{k \in MP_p} o_k \times (x_k^{a,t} + c_{k,sp}^{a,t-ts}) \leq 0 \quad (18)$$

$$\underline{g}^{a,e,t} \sum_{p=1}^P \sum_{k \in MP_p} o_k \times (x_k^{a,t} + c_{k,sp}^{a,t-ts}) - \sum_{p=1}^P \sum_{k \in MP_p} g_k^e \times o_k \times (x_k^{a,t} + c_{k,sp}^{a,t-ts}) \leq 0 \quad (19)$$

$$\sum_{p=1}^P \sum_{k \in MP_p} g_k^e \times o_k \times s_k^{a,t} - \overline{g}^{a,e,t} \sum_{p=1}^P \sum_{k \in MP_p} o_k \times s_k^{a,t} \leq 0 \quad (20)$$

$$\underline{g}^{a,e,t} \sum_{p=1}^P \sum_{k \in MP_p} o_k \times s_k^{a,t} - \sum_{p=1}^P \sum_{k \in MP_p} g_k^e \times o_k \times s_k^{a,t} \leq 0 \quad (21)$$

Ore fines grade blending constraints ensure the extracted ore from mining-cut  $k$  within mining-panel  $p$  sent to either processing destination  $a$  or to stockpile  $sp$  in period  $t$  meets the fines requirements. Interburden fines grade blending constraints also ensure that the interburden fines for dyke construction are within the upper and lower limits required. Fines grade blending constraints are formulated using Eqs. (22) to (25). Eqs. (22) and (23) represent inequality constraints used to control the limiting grade of ore fines sent from the mine and stockpile to the processing plant. Eqs. (24) and (25) represent inequality constraints used to control the limiting grade of ore fines sent from the mine to the stockpile.

$$\sum_{p=1}^P \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} o_k \times fn_k^e \times (x_k^{a,t} + c_{k,sp}^{a,t}) \right) - \overline{fn}^{a,t,e} \sum_{p=1}^P \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} o_k \times (x_k^{a,t} + c_{k,sp}^{a,t}) \right) \leq 0 \quad (22)$$

$$\underline{fn}^{a,t,e} \sum_{p=1}^P \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} o_k \times (x_k^{a,t} + c_{k,sp}^{a,t}) \right) - \sum_{p=1}^P \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} o_k \times fn_k^e \times (x_k^{a,t} + c_{k,sp}^{a,t}) \right) \leq 0 \quad (23)$$

$$\sum_{p=1}^P \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} o_k \times fn_k^e \times s_{k,sp}^{a,t} \right) - \overline{fn}^{a,t,e} \sum_{p=1}^P \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} o_k \times s_{k,sp}^{a,t} \right) \leq 0 \quad (24)$$

$$\underline{fn}^{a,t,e} \sum_{p=1}^P \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} o_k \times s_{k,sp}^{a,t} \right) - \sum_{p=1}^P \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} o_k \times fn_k^e \times s_{k,sp}^{a,t} \right) \leq 0 \quad (25)$$

### 5.3. The MILGP model for optimizing reclamation and dyke material schedule

The objective function of the MILGP model that minimizes reclamation material cost and the dyke construction cost as part of the waste management operation can be formulated using the continuous decision variables  $v_k^{d,t}$ ,  $z_k^{d,t}$ ,  $u_k^{d,t}$ , and  $q_k^{d,t}$ . These variables used to model *RM*, *OB*, *IB* and *TCS* dyke material requirements, respectively, for all dyke construction destinations. Other continuous deviational variables,  $dv_1^{-,d,t}$ ,  $dv_2^{-,a,t}$ ,  $dv_3^{-,d,t}$ , and  $dv_4^{-,d,t}$  are defined to support the goal functions that control RM, OB, IB and TCS dyke materials, for all reclamation and dyke construction destinations. They provide a continuous range of units (tonnes) that can be determined by the optimizer to satisfy the set goals. In the objective function, these deviational variables are minimized. There are also deviational penalty cost and priority parameters in the objective function used to model the focus of mine management in the presence of multiple conflicting goals.

The deviational penalty cost parameters  $PN_1$ ,  $PN_2$ ,  $PN_3$ , and  $PN_4$  penalize the NPV for any deviation from the set goals. The priority parameters,  $P_1$ ,  $P_2$ ,  $P_3$ , and  $P_4$  are used to place emphasis on the most important goals. In general, the deviational penalty cost and priority parameters are set up to penalize the NPV if the set goals and the most important goals are not met. When setting up these parameters, the planner has to monitor how continuous mining proceed period by period, the uniformity of tonnages mined per period, and the corresponding NPV generated, to keep track of how parameter changes affect these key performance indicators. More weight should be assigned to a goal that has a higher priority for mine management. The objective function for minimizing the reclamation material and the dyke construction cost is represented by Eq. (26):

$$\text{Min} \sum_{l=1}^L \sum_{j=1}^J \sum_{d=1}^D \sum_{sp=1}^{SP} \sum_{a=1}^A \sum_{e=1}^E \sum_{t=1}^T \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} \left[ \begin{aligned} & \left( dm u_k^{d,t} \times v_k^{d,t} \right) - \left( do b_k^{d,t} \times z_k^{d,t} + di b_k^{d,t} \times u_k^{d,t} + dt_k^{d,t} \times q_k^{d,t} \right) \\ & - P_1 \left( PN_1 \times dv_1^{-,d,t} \right) - P_2 \left( PN_2 \times dv_2^{-,a,t} \right) \\ & - P_3 \left( PN_3 \times dv_3^{-,d,t} \right) - P_4 \left( PN_4 \times dv_4^{-,d,t} \right) \end{aligned} \right] \right) \quad (26)$$

There are tonnage targets for reclamation material and dyke material for dyke construction destinations. Eqs. (27) to (30) represent all required goal functions. Eq. (27) defines the RM tonnage goal  $MUG^{d,t}$  that control the total amount of RM to be mined from mining-cut  $k$  within mining-panel  $p$  in each period. The negative allowable deviation from the set RM goal is controlled by the planner using the  $dv_1^{-,d,t}$  decision variable. *OB*, *IB* and *TCS* dyke material goal functions control the dyke material production targets,  $OBg^{d,t}$ ,  $IBg^{d,t}$  and  $CSg^{d,t}$  for different dyke construction destinations. These are defined by Eqs. (28) to (30), respectively. These functions provide a feasible schedule for dyke construction. The negative allowable deviation from the set *OB*, *IB* and *TCS* dyke material goals are controlled by the planner using  $dv_2^{-,d,t}$ ,  $dv_3^{-,d,t}$  and  $dv_4^{-,d,t}$  decision variables.

$$\sum_{p=1}^P \left( \sum_{k \in MP_p} \left( mu_p \times v_p^{d,t} \right) \right) + dv_3^{-,d,t} = MUG^{d,t} \quad (27)$$

$$\sum_{p=1}^P \left( \sum_{k \in MP_p} \left( ob_k \times z_k^{d,t} \right) \right) + dv_4^{-,d,t} = OBg^{d,t} \quad (28)$$

$$\sum_{p=1}^P \left( \sum_{k \in MP_p} (ib_k \times u_k^{d,t}) \right) + dv_5^{-d,t} = IBg^{d,t} \quad (29)$$

$$\sum_{p=1}^P \left( \sum_{k \in MP_p} (cs_k \times q_k^{d,t}) \right) + dv_6^{-d,t} = CSg^{d,t} \quad (30)$$

Eqs. (31) and (32) represent inequality constraints used to control the limiting grade of interburden dyke material fines sent from the mine to dyke construction destinations.

$$\sum_{p=1}^P \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} ib_k \times fn_k^{ib} \times u_k^{d,t} \right) - \overline{fn}^{d,t,ib} \sum_{p=1}^P \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} ib_k \times u_k^{d,t} \right) \leq 0 \quad (31)$$

$$\overline{fn}^{d,t,ib} \sum_{p=1}^P \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} ib_k \times u_k^{d,t} \right) - \sum_{p=1}^P \left( \sum_{\substack{k \in MK_p \\ p \in MP_j}} ib_k \times fn_k^{ib} \times u_k^{d,t} \right) \leq 0 \quad (32)$$

#### 5.4. The MILGP model general constraints

##### 5.4.1. Mining-panels extraction precedence constraints

Five precedence constraints presented in Eqs. (33) to (37) are used to define the precedence extraction sequence for each mining panel  $p$  based on its spatial location. These equations use the binary integer decision variable  $b_p$ . This variable is equal to one if the extraction of mining-panel  $p$  has started by or in period  $t$ ; otherwise, it is zero. Specifically:

- Eq. (33) defines the vertical mining precedence. Prior to the extraction of a specific mining-panel, all the mining-panels above it must be extracted so that the mining-panel is accessible. The set  $IP_p(Z'')$  represents the set of immediate mining-panels that are above mining-panel  $p$ .
- Eq. (34) defines the horizontal mining precedence. Prior to the extraction of a specific mining-panel, all the mining-panels in a specified horizontal mining direction on a level must be extracted. The set  $IH_p(Z')$  represents the set of immediate mining-panels in the specified horizontal mining direction.
- Eq. (35) defines the pushback mining precedence. Eq. (35) checks all the mining-panels within the immediate predecessor pushback that must be extracted prior to the extraction of mining-panels in pushback  $j$ . The set  $MP_j(H'')$  represents the set of mining panels in the predecessor pushback.
- Eq. (36) ensures that mining-panel  $p$  can only be extracted if it has not been extracted before.
- Eq. (37) ensures that once the extraction of a mining-panel starts in period  $t$ , this mining-panel is available for extraction during the subsequent periods.

$$b_p^t - \sum_{c=1}^c \sum_{m=1}^t y_{u_i}^{c,m} \leq 0, \quad u_i \in IP_p(Z'') \quad (33)$$

$$b_p^t - \sum_{c=1}^c \sum_{m=1}^t y_{u_2}^{c,m} \leq 0, \quad u_2 \in IH_p(Z') \quad (34)$$

$$b_p^t - \sum_{c=1}^c \sum_{m=1}^t y_{u_3}^{c,m} \leq 0, \quad u_3 \in MP_j(H'') \quad (35)$$

$$\sum_{c=1}^c \sum_{m=1}^t y_p^{c,m} - b_p^t \leq 0 \quad (36)$$

$$b_p^t - b_p^{t+1} \leq 0 \quad (37)$$

#### 5.4.2. Decision variables' control constraints

In the MILGP model, all decision variables used to control mining, processing, stockpiling, reclamation material, dyke materials and goal deviations are continuous variables. Inequality Eq. (38) makes sure that all the material mined as ore (sent to either the processing destination  $a$  or the stockpile  $sp$ ), and all reclamation and dyke materials extracted from the mining-cuts belonging to mining-panel  $p$  in period  $t$  are less than or equal to the total material mined from mining-panel  $p$  in period  $t$  from any mining location.

Eq. (39) ensures that the total fractions of ore mined from a mining-cut (sent to either the processing destination  $a$  or the stockpile  $sp$ ) is less than or equal to one. Eq. (40) ensures that the fraction of ore extracted from mining-cut  $k$  and sent to the stockpile  $sp$  in period  $t - ts$  must be equal to the fraction of ore reclaimed from the stockpile  $sp$  and sent to the processing plant  $a$  in period  $t$ ; where  $ts$  is the stockpiling duration.

Eq. (41) ensures that the fractions of TCS dyke material produced from processed ore is less than or equal to the fractions of ore sent from the mine and stockpile to the processing plant in each period. Eq. (42) ensures that the fractions of mining-panel  $p$  extracted and sent to different destinations in different periods is less than or equal to one. Eq. (43) ensures that the total fractions of reclamation material extracted from the mine and sent to its destinations in different periods is less than or equal to one. Eqs. (44) to (46) ensure that the total fractions of dyke materials extracted from the mine (OB and IB) or generated from the processing plant (TCS) and sent to all destinations in different periods is less than or equal to one.

$$\left[ \begin{array}{l} \sum_{d=1}^D \sum_{sp=1}^{SP} \sum_{a=1}^A \sum_{\substack{k \in MK_p \\ p \in MP_j}} (o_k \times x_k^{a,t} + o_k \times s_{k,sp}^{a,t} + mu_k \times v_k^{d,t} + ob_k \times z_k^{d,t} + ib_k \times u_k^{d,t}) \\ \leq \sum_{l=1}^L \sum_{p \in MP_p} [y_p^{l,t} (o_p + mu_p + ob_p + ib_p + w_p)] \end{array} \right] \quad (38)$$

$$\sum_{a=1}^A \sum_{sp=1}^{SP} \sum_{t=1}^T (x_k^{a,t} + s_{k,sp}^{a,t}) \leq 1 \quad (39)$$

$$\sum_{a=1}^A \sum_{sp=1}^{SP} \sum_{t=1}^T (c_{k,sp}^{a,t} - s_{k,sp}^{a,t-ts}) = 0, \quad t - ts \geq 0 \quad (40)$$

$$\sum_{d=1}^D \sum_{t=1}^T q_k^{d,t} \leq \sum_{a=1}^A \sum_{t=1}^T x_k^{a,t} + \sum_{sp=1}^{SP} \sum_{t=1}^T c_{k,sp}^{a,t} \quad (41)$$

$$\sum_{d=1}^D \sum_{t=1}^T y_p^{d,t} \leq 1 \quad (42)$$



$$\sum_{d=1}^D \sum_{t=1}^T v_p^{d,t} \leq 1 \quad (43)$$

$$\sum_{d=1}^D \sum_{t=1}^T z_p^{d,t} \leq 1 \quad (44)$$

$$\sum_{d=1}^D \sum_{t=1}^T u_p^{d,t} \leq 1 \quad (45)$$

$$\sum_{d=1}^D \sum_{t=1}^T q_p^{d,t} \leq 1 \quad (46)$$

### 5.4.3. Non-negativity constraints

Eq. (47) ensures that the decision variables for mining, processing, stockpiling (ore sent and reclaimed), *RM*, *OB*, *IB* and *TCS* dyke material are non-negative. Eq. (48) ensures that the deviational decision variables that support the goal functions are non-negative as well.

$$y_p^{l,t}, x_k^{a,t}, s_{k,sp}^{d,t}, c_{k,sp}^{a,t}, v_k^{d,t}, z_k^{d,t}, u_k^{d,t}, q_k^{d,t} \geq 0 \quad (47)$$

$$dv_1^{-l,t}, dv_2^{-a,t}, dv_3^{-d,t}, dv_4^{-d,t}, dv_5^{-d,t}, dv_6^{-a,t} \geq 0 \quad (48)$$

## 6. Implementation of the MILGP framework

In general, this section documents the application and results from the developed MILGP model for an oil sands dataset. Whittle software (Gemcom Software International, 2015b), which is based on 3D LG algorithm (Lerchs and Grossmann, 1965) is used to generate the optimized pit limit for the oil sands mine. The optimized pit shell from Whittle is used to design the final pit in GEMS software (Gemcom Software International, 2015a). The blocks within the final pit design are used as input data for the MILGP model for subsequent integrated long-term production scheduling and waste disposal planning. An agglomerative hierarchical clustering algorithm is used in clustering blocks within each intermediate pushback into mining-cuts (Tabesh and Askari-Nasab, 2011). The intersection between benches and intermediate pushbacks are used in creating mining-panels.

As mentioned in section **Error! Reference source not found.**, organic rich solids ORS comprising about 5% of the total ore and it might reduce overall bitumen recovery by carrying any associated bitumen into the aqueous tailings (O'Carroll, 2002; Sparks, et al., 2003). In this sense, ORSs are considered to be active and might be a better predictor for ore processability than the traditional use of bitumen ore fines contents. The recovery is calculated based on the BIT to ORS ratio. Directive 082 identifies the recovery as one of the operating criteria used by the AER. It is required that the bitumen recovery must be calculated based on the bitumen content. The recovery equals to 90% if the bitumen content greater than or equal to 11%, otherwise, Eq. **Error! Reference source not found.** will be used. It is noted that the recovery calculated based on AER requirements is always greater than or equal to the recovery calculated based on BIT to ORS ratio. The recoveries are percentage difference between zero and 4% (Fig. 5)

The MILGP model has two new robust constraints, which control the periodic tonnage fluctuation for mining and processing material. The tonnage fluctuation constraints eliminate the need to set a mining and processing target. It requires the mine planner to set an acceptable cumulative periodic tonnage fluctuation throughout the mine life, together with any production ramp up or ramp down requirements. The optimizer then determines the appropriate mining and processing targets that meets the tonnage fluctuation requirements. The cumulative periodic tonnage fluctuation value also controls the maximum possible mining and processing capacities indirectly. The tonnage fluctuation constraints provide varying practical production schedule options for mine planners. This MILGP

model generates a smooth and practical production schedule, a NPV with known limits of optimality and is easy to setup with more flexibility for the optimizer.

Two implementation scenarios highlighting different aspects of the developed MILGP model are outlined in Fig. 6. These scenarios are designed to highlight features of the MILGP model including:

1. Determining the mining and processing annual targets as part of the production scheduling optimization process and not as an input;
2. Determining the NPV based on the revenue generated from AER recovery (Scenario 1);
3. Determining the NPV based on the revenue generated from BIT:ORS recovery (Scenario 2).

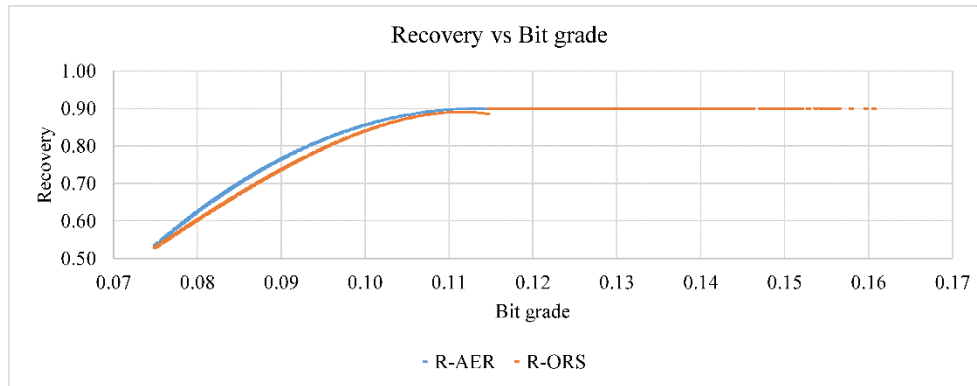


Fig. 5. Recovery vs bitumen grade.

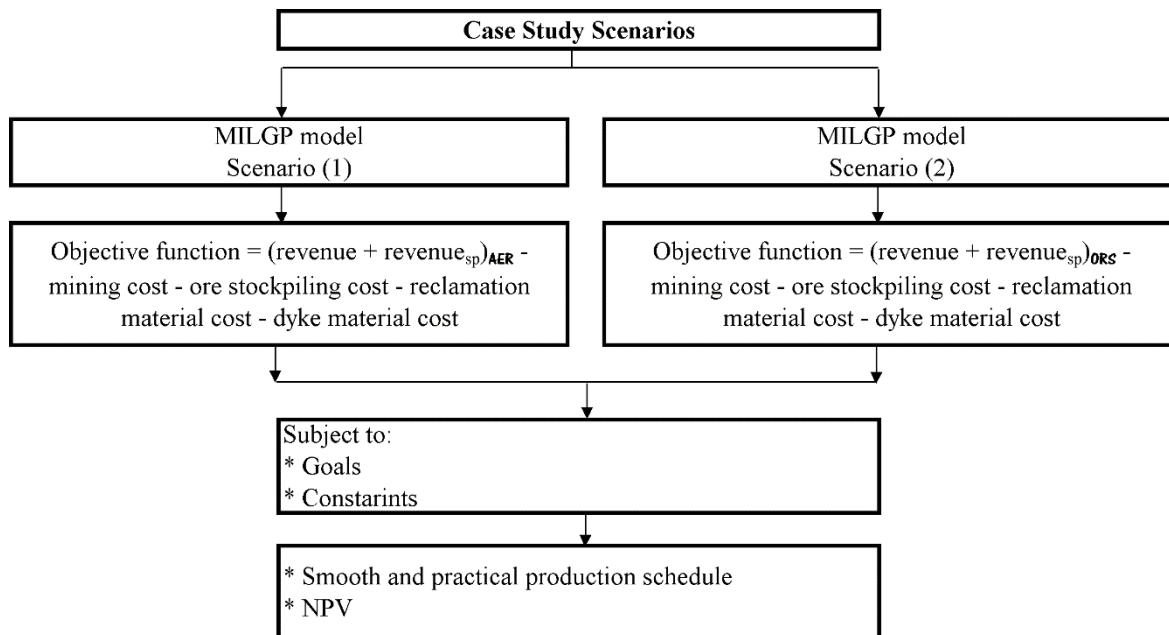


Fig. 6. Case study scenarios.

## 7. Case Study

For this case study, no pushbacks prior to the UPL were considered. However, to create mining-panels, the ultimate pit was divided into four pseudo pushbacks. Blocks in each mining-panel were clustered into mining-cuts using hierarchical clustering algorithm (Tabesh and Askari-Nasab, 2011). Two implementation scenarios as outlined in Section **Error! Reference source not found.** are investigated with the oil sands data. The deposit is to be scheduled for 8 years for the processing

plant, reclamation and dyke construction destinations. Summarized information on the oil sands deposit final pit design is presented in Table 1.

Table 2 shows the economic parameters and operational capacities for production scheduling. The economic data are extracted and compiled based on (Ben-Awuah and Askari-Nasab, 2013). Table 4 shows the upper and lower bounds of material quality requirements for ore and interburden dyke material. The model is implemented on a Lenovo Think Pad computer with i5 Core at 2.2 GHz, and 8.0 GB of RAM.

Table 1. Oil sands deposit final pit design characteristics.

Description	Value
Total tonnage of material (Mt)	182.23
Total ore tonnage (Mt)	88.44
Total TCS dyke material tonnage (Mt)	66.33
Total OB dyke material tonnage (Mt)	18.02
Total IB dyke material tonnage (Mt)	20.05
Total RM tonnage (Mt)	7.05
Number of blocks	2,523
Number of mining-cuts	155
Number of mining-panels	22
Number of benches	6

Table 2. Economic parameters and operational capacities.

Parameter (unit)	Value	Parameter (unit)	Value
Mining cost (\$/tonne)	4.60	Cumulative periodic processing tonnage fluctuation (Mt)	30.00
Processing cost (\$/tonne)	5.03	Mining recovery fraction (%)	100.00
Ore re-handling cost (\$/tonne)	0.50	Processing recovery (%)	90.00
Selling price (\$/bitumen %mass)	4.50	Discount rate (%)	10.00
TCS dyke material cost (\$/tonne)	0.92	RM capacity (MT/year)	1.42
OB dyke material cost (\$/tonne)	1.38	OB capacity (MT/year)	2.50
IB dyke material cost (\$/tonne)	1.38	IB capacity (MT/year)	2.40
RM extra mining cost (\$/tonne)	0.50	TCS capacity (MT/year)	9.00
Cumulative periodic mining tonnage fluctuation (Mt)	50.00		

Table 3. Material quality requirements.

Parameter	Value
Upper bound of ore bitumen grade (wt %)	16.0
Lower bound of ore bitumen grade (wt %)	7.0
Upper bound of ore fines percent (wt %)	30.0
Lower bound of ore fines percent (wt %)	0.0
Upper bound of IB dyke material fines percent (wt %)	50.0
Lower bound of IB dyke material fines percent (wt %)	0.0

In Scenario 1 and 2, the mining and processing capacities are not required to be set. The mine planner decides on an acceptable cumulative periodic tonnage fluctuation throughout the mine life for the mining and processing operations and allows the optimizer to determine the mining and processing limits that meets the cumulative periodic tonnage fluctuation value. The planner also controls how

many ramping up years is allowed at the beginning of the operation and how many ramping down years is allowed at the end. In this case study, 1-year ramping up is allowed at the beginning and 1-year ramping down is allowed at the end. The cumulative periodic tonnage fluctuation value used is 50.0 Mt for mining and 30.0 Mt for processing. This means the maximum possible mining and processing capacities allowed is 25 Mt and 15 Mt, respectively. In other words, not more than 50% of the cumulative periodic tonnage fluctuation value. The focus is to achieve a smooth processing rate throughout the mine life and generate a uniform production schedule that generates the highest NPV.

### 7.1. MILGP model: Scenario 1

In Scenario 1 for this case study, the overall NPV generated, including the reclamation and dyke material costs, is \$ 1,499.3 M. The results of the production schedule are presented in Table 4 and Fig. 7 to Fig. 9.

Table 4. Production schedule using cumulative periodic tonnage fluctuation of 50.0 Mt for mining and 30.0 Mt for processing for a 2-year ore stockpiling duration and based on AER recovery (Scenario 1).

Period	Average bitumen grade (wt %)	Material mined (Mt)	Material processed (Mt)
1	8.32	25.00	0.89
2	10.55	25.00	13.85
3	11.53	25.00	13.85
4	10.97	25.00	13.85
5	10.86	25.00	13.85
6	11.10	25.00	13.85
7	11.21	25.00	13.85
8	10.90	6.90	4.43

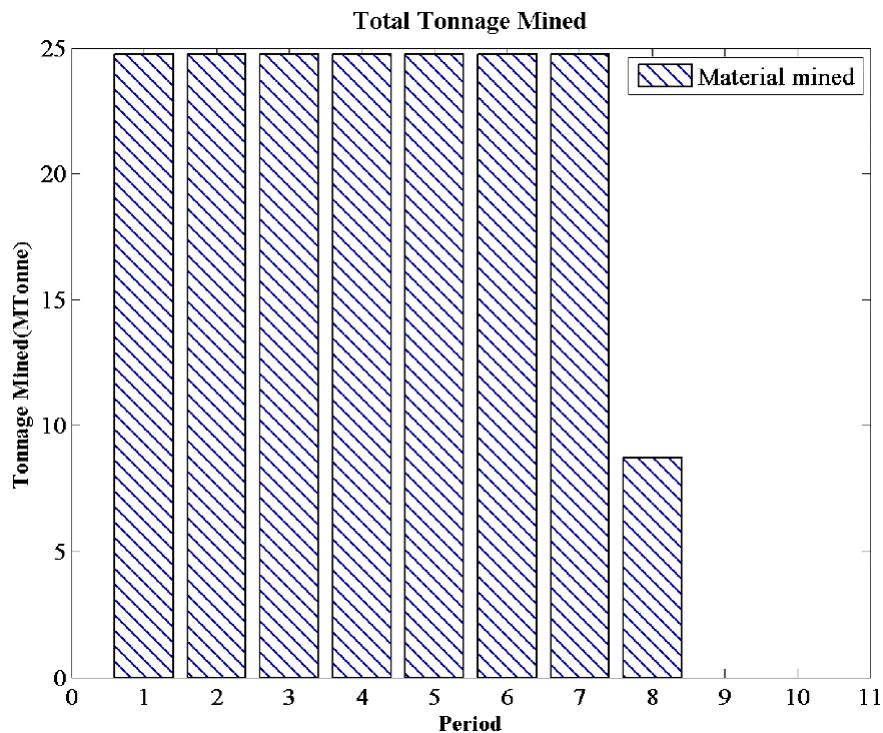


Fig. 7. Mining schedule using tonnage fluctuation constraints based on AER recovery.

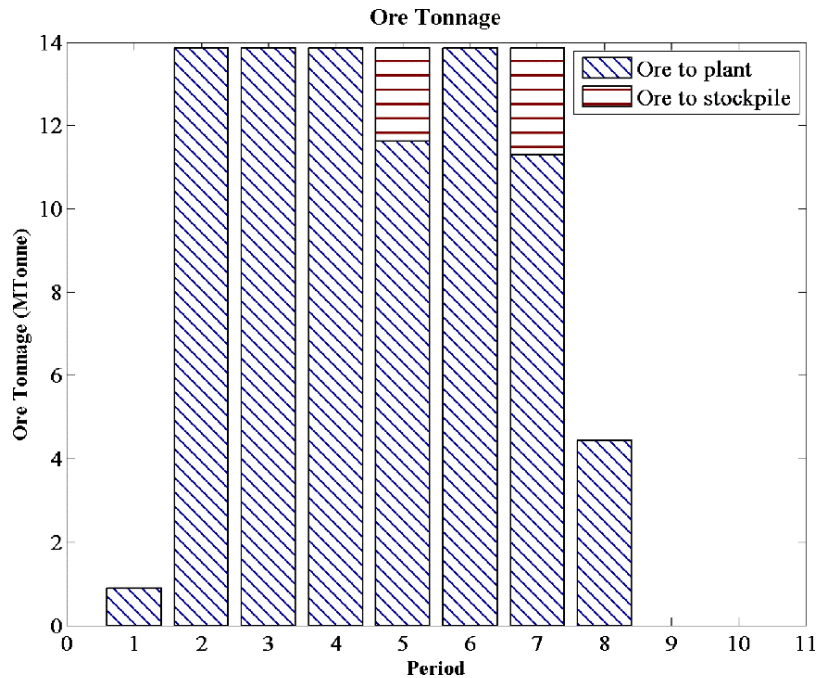


Fig. 8. Processing schedule using tonnage fluctuation constraints with a 2-year ore stockpiling duration based on AER recovery.

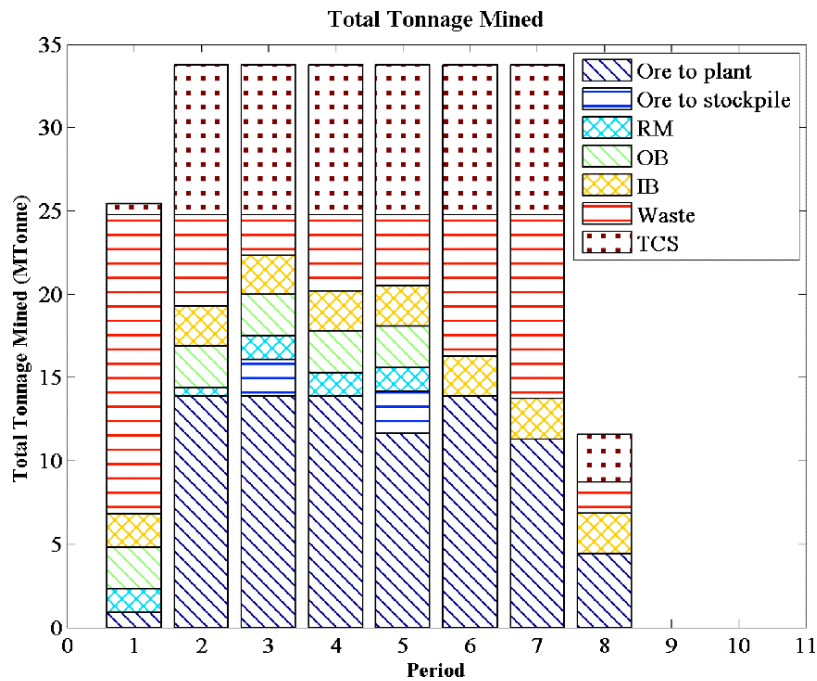


Fig. 9. Ore, RM, OB, IB and TCS dyke materials, and waste schedule using tonnage fluctuation constraints with a 2-year ore stockpiling duration based on AER recovery.

**7.2. MILGP model: Scenario 2**

In Scenario 2 for this case study, the overall NPV generated, including the reclamation and dyke material cost, is \$ 1,468.2 M. The results of the production schedule are presented in Table 4 and Fig. 7 to Fig. 9.

Table 5. Production schedule using cumulative periodic tonnage fluctuation of 50.0 Mt for mining and 30.0 Mt for processing for a 2-year ore stockpiling duration and based on ORS recovery (Scenario 2).

Period	Average bitumen grade (wt %)	Material mined (Mt)	Material processed (Mt)
1	8.32	24.78	0.89
2	10.57	24.78	13.80
3	11.53	24.78	13.80
4	10.98	24.78	13.80
5	10.85	24.78	13.80
6	11.09	24.78	13.80
7	11.25	24.78	13.80
8	10.90	8.43	4.75

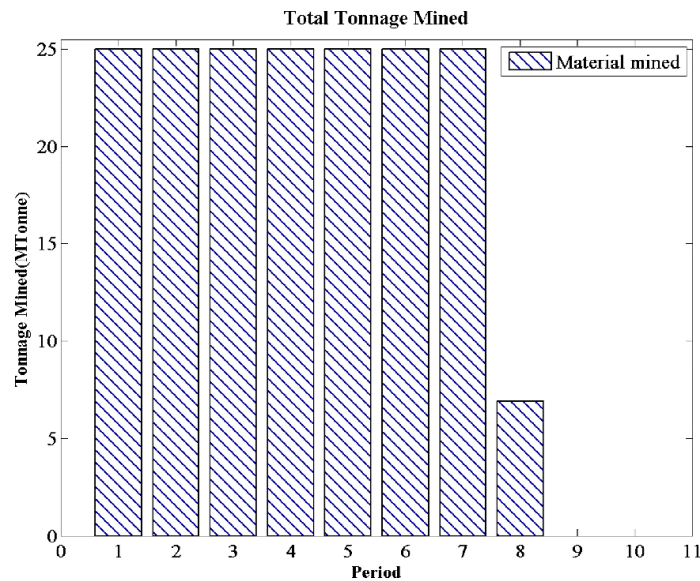


Fig. 10. Mining schedule using tonnage fluctuation constraints based on ORS recovery.

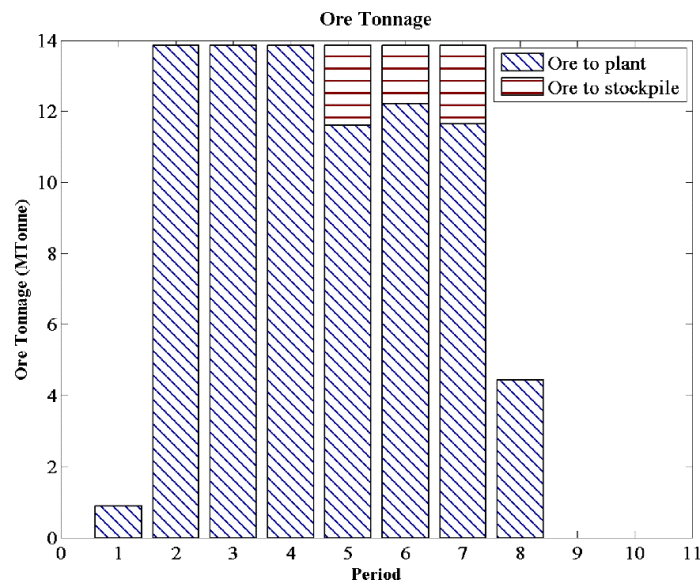


Fig. 11. Processing schedule using tonnage fluctuation constraints with a 2-year ore stockpiling duration based on ORS recovery.

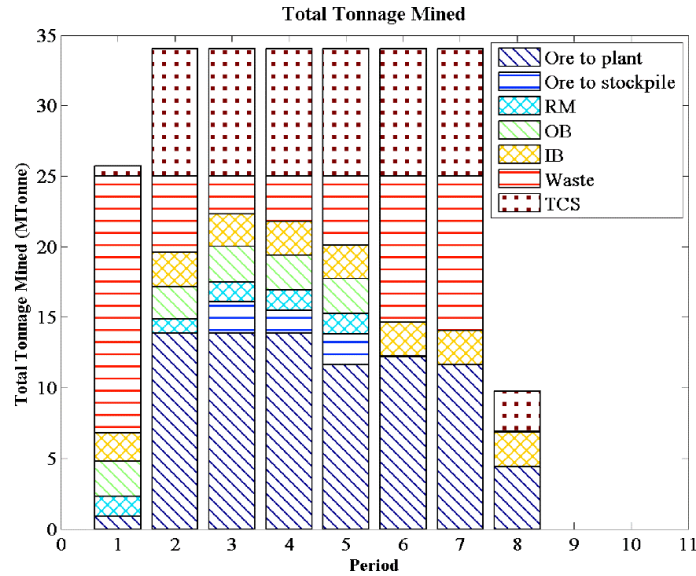


Fig. 12. Ore, RM, OB, IB and TCS dyke materials, and waste schedule using tonnage fluctuation constraints with a 2-year ore stockpiling duration based on ORS recovery.

### 7.3. Discussion of the results for Scenarios 1 and 2

The performance of the MILGP model in Scenarios 1 and 2 is analyzed based on: NPV, mining and processing production targets, bitumen grade profile and smoothness and practicality of the generated schedules. Both scenarios generate smooth schedules for mining, processing, reclamation and dyke materials. However, in Scenario 1, more material is mined and processed all through the years compared to material mined and processed in the scenario 2. The average bitumen head grade is slightly different due to the amount of material processed and the use of the stockpile. The overall NPV generated from scenario 1, including the reclamation and dyke material cost, is 2.05% higher (\$ 31.0 M) than the overall NPV generated from scenario 2.

There are two reasons explain the differences in the generated NPV. Firstly, and the primary reason is that the calculated recovery based on AER is always higher or equal to the recovery generated based on BIT:ORS ratio. Secondly, more ore is processed all through the years that means higher NPV. The average bitumen head grade is slightly different due to the difference in mining and processing schedules, and stockpile reclamation in different periods. The total material mined and processed, using Scenario 1 and 2, are the same as 181.9 Mt and 88.44 Mt, respectively. The model generated a uniform production schedule for OB, IB and TCS dyke material over the 8 periods for both scenarios. This ensures the effective utilization of the mining fleet and processing plant throughout the mine life.

The main advantage of using the new robust tonnage fluctuation constraints is that they are easy to set up and there is no need to decide on the periodic mining and processing targets. The only inputs required is how much total deviation is allowed throughout the mine life and if there is any production ramp up or ramp down requirements. These constraints provide varying practical production schedule options for mine planners.

## 8. Conclusions

The MILGP model for oil sands long-term production planning involves the interactions of their three main subcomponents: the objective function, the goal functions and the constraints in an optimization framework to achieve the research objectives. The MILGP model uses tonnage fluctuation constraints for mining and processing. Tonnage fluctuation constraints are easy to set up

and do not require the periodic mining and processing targets. This provides robust and practical production schedule options for mine planners. The model generates a strategic production schedule for the ore, reclamation and dyke materials using two different scenarios. For both scenarios, the MILGP model illustrates how production scheduling with limited duration stockpiling strategy for ore can be effectively integrated with waste disposal planning and reclamation material stockpiling in oil sands mining. Based on dyke construction requirements, schedules are generated to provide the required dyke materials to support engineered dyke construction that will help in reducing environmental impacts. This schedule gives the planner a satisfactory control over dyke materials and provide a solid platform for effective dyke construction and waste management planning. The MILGP model is extended to integrate a mine-to-mill production planning strategy that uses organic rich solids (ORSs) content during optimization.

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