UNIVERSITÉ DE MONTRÉAL

A MIXED-INTEGER PROGRAMMING MODEL FOR AN IN-PIT CRUSHER CONVEYOR LOCATION PROBLEM

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Ce mémoire intitulé :

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DEDICATION

To those who love me
and never stopped supporting me...

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RÉSUMÉ

Les coûts de transport représentent environ la moitié du coût total de fonctionnement (d'exploitation) dans les grandes mines à ciel ouvert. Une manière de réduire les coûts de transport est de raccourcir les distances de transport en rapprochant le point de déchargement du camion ou même de le placer dans la mine. Il y a une tendance à utiliser des systèmes de convoyeurs à grande vitesse et à grande capacité, lesquels ont été très productifs. Les systèmes de transport camion-pelle qu'utilisent des convoyeurs comparés aux conventionnels offrent une rentabilité opérationnelle supérieure et une grande fiabilité du concassage dans la fosse, ce qui les rend plus attrayants pour les activités minières modernes. Les principaux éléments à considérer dans la planification minière pour implémenter un système de concassage dans la fosse sont la disposition du convoyeur et la position du concasseur. Ce projet vise à résoudre le problème de localisation d'un système de convoyeur à concasseur dans les fosses à travers l'utilisation d'un modèle d'installation à capacité dynamique, en prenant compte des paramètres opérationnels et financiers, et de l'ordonnancement du plan minier. La méthodologie a été construite pour localiser l'équipement et la disposition du convoyeur pour un projet d'une mine de fer. Les résultats sont applicables en considérant certaines exigences liées à la géologie, à la géométrie des fosses et aux distances de transport.

ABSTRACT

Haulage costs account for around a half of the total operating costs in large open-pit mines. One way to reduce the haulage costs is to shorten the haulage distances by bringing the truck dump point closer or even into the mine. There is a tendency in the direction of the high speed, large capacity conveyor systems, and these arrangements have been very productive. Conveying and truck-shovel systems compared to conventional truck-shovel systems alone, provide operating cost efficiency and high reliability of in-pit crushing, making those types of systems more appealing to be implemented in modern mining activities. The main elements to be considered in mine planning to implement an in-pit crusher system are conveyor layout and crusher position. This project aims to solve the location problem of an in-pit crusher conveyor system through the use of a dynamic uncapacitated facility problem model, considering operative and financial parameters and mine plan scheduling. The methodology was constructed for locating the in-pit crusher equipment and conveyor layout for an iron mine project. The results are applicable for considering certain conditions related to geology, pit geometry and transport distances.

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LIST OF ACRONYMS AND ABBREVIATIONS

AMPL	A Mathematical Programming Language
BCM	Bank Cubic Meter
BEV	Block Economic Value
CAPEX	Capital Expenditure
DCF	Discounted Cash Flow
DFUL	Dynamic Facility Uncapacitated Location
DTWR	Davis Tube Weight Recovery
EBV	Economic Block Value
EFH	Equivalent Flat Haul
EPGAP	Relative MIP Gap Tolerance
IP	Integer Programming
IPCC	In-Pit Crushing and Conveying
LCM	Loose Cubic Meter
LP	Linear Programming
LTP	Long-Term Plan
m	Meters
Mt	Million Tonnes
MILP	Mixed Integer Linear Programming
MIP	Mixed Integer Programming
NPV	Net Present Value
OPEX	Operating Expenses
OPMPS	Open-Pit Mine Production Scheduling
QP	Quadratic Programming
ROM	Run Of Mine
SMP	Strategic Mine Planning
TS	Truck and Shovel
UPL	Ultimate Pit Limit
UFL	Uncapacitated Facility Location

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CHAPTER 1 INTRODUCTION

Open pit mining is a large operation of excavation within which a considerable amount of material should be extracted and removed out of the mining site. When materials inside of an open pit are excavated and loaded to the haulage system, according to material types, they are transported to the predefined locations. In open pit mines, transportation costs are about 50% of the total operating costs (Czaplicki, 2009). The traditional system for this operation is composed by shovels and trucks working as a semi continuous system.

In recent years, the application of In-Pit Crushing and Conveying (IPCC) has been of interest to open-pit mine design and planning. Beside from choosing a suitable type of crushing and loading system, the transport system should also be optimized to reduce the mine transportation costs.

Selecting a transportation system for a new project is based primarily on operating experience at similar type deposits and on methods already in use in the region of the deposit. Then, the chosen mining method is modified during the early years of mining as ground conditions and ore characteristics are better understood. Nowadays, however, the large capital investment required to open a new mine needs that the transportation systems is examined during the feasibility studies. By doing this the selected system have a high probability of obtaining the estimated production rates.

The presented formulation in this thesis is part of the linear programming family. This mathematical model determines dynamically an optimal solution for a mine plan, belt conveyor and crushers locations using an IPCC transportation system. Current IPCC methods are based in fixed mine scheduling as input to determine the IPCC system location; it generates suboptimal results due to the inability to deal with various mine scheduling scenarios.

The innovating approaches presented in this thesis to integrate IPCC's economic and operational factors to the Strategic Mine Planning (SMP) add opportunities to evaluate certain problems such as fixed crusher location and IPCC system viability against truck and shovel.

1.1 Definitions

The mining engineering terms used are briefly explained in this section. Some terms may appear with different meanings in other disciplines; the definitions provided are given below and will be applied in the remaining parts of this thesis.

1.1.1 Block Model

A block model consists of a discretization of a mineral deposit in parallelepipeds, or blocks, of the same size. Bases on geostatistical methods, each block is assigned various values representing attributes of the rock mass, for example: geographic location of the coordinates, amount of ore it contains, density, grade content as Figure 1.1 depicts. Geostatistical methods are typically applied to calculate the attributes for every block made from mineral exploration data like drill holes, laboratory samples and geological mapping.

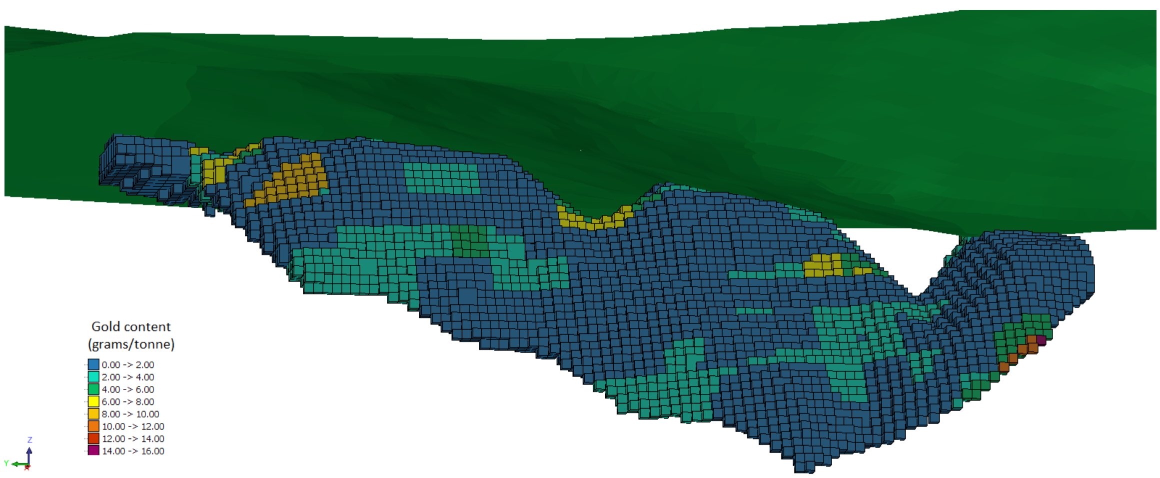

Figure 1.1 Mineral deposit represented as a block model

1.1.2 Block Economic Value (BEV)

The economic classification of a block is based on estimate of its profitable material compared to its total weight. If a block is profitable, then it is classified as ore; otherwise, it is classified as waste. If a block is ore or waste is define by the cut-off grade parameter. As stated by Rendu (2009) cut off grade corresponds to the lowest mineral grade that can have a mineralized body to be extracted and sent to the processing plant to obtain an economic benefit. All material having a grade content over the cut off is classified as a mineral for processing, while the remainder having a lower grade content is considered waste and must be sent to waste dump. Cut off is also used to decide if a material need to be stockpiled or processed immediately. In a deterministic model the Block Economic Value (BEV) can be evaluated using:

$$BEV = g \times R^P \times (P - C^R) - (C^M - C^P - C^O) \tag{1.1}$$

g	average grade of profitable product of a tonne of material.
R^P	recovery rate from a tonne of mined material.
P	commodity price of one unit of recovered profitable product contained in a tonne of ore.
C^R	refining costs per unit of profitable product contained in a tonne of ore.
C^M	mining cost per tonne of material.
C^P	processing cost per tonne of ore.
C^O	overhead cost per tonne of material.

1.1.3 Precedence relations

The order of blocks extraction to maintain pit slope walls stable and equipment can access to the work bench. For example, if a maximal slope of 45° is permitted, each block there is a set which contains five (5) connected blocks that must be mined previously to expose the block to be mined. Figure 1.2 shows blocks needed to extract before mining the block at bottom.

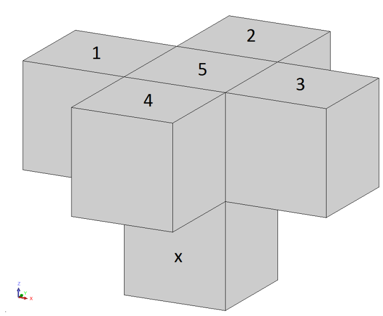

Figure 1.2 Block precedence relations using pattern 1:5

1.1.4 Ultimate pit limit

The ultimate pit limit or the final pit contour determines a set of blocks containing the portions of the mineral deposit that might be mined to maximize the profit subject to precedence relations and omitting the dimension of time. The ultimate pit limit corresponds to the final topology of the open-pit mine. An open-pit mine is composed of benches and ramps. The benches are formed by blocks having the same size and ramps are used as haul road for material transportation. This problem can be expressed as an integer linear program (Newman et al., 2010) :

Sets:

- $\mathcal{B}$: set of blocks b.
- $\mathcal{B}_b$: set of precedences of block b.

Parameter:

- C_b: value obtained from extracting and processing block b.

Variable:

- $y_b = \begin{cases} 1 \text{ if block } b \text{ is in the ultimate pit,} \\ 0 \text{ otherwise} \end{cases}$

$$\text{Maximize: } \sum_{b \in \mathcal{B}} y_b C_b \tag{1.2}$$

Subject to:

$$y_b \leq y_{b'} \qquad \forall b \in \mathcal{B}, \forall b' \in \mathcal{B}_b \tag{1.3}$$

$$y_b \in \{0, 1\} \qquad \forall b \in \mathcal{B} \tag{1.4}$$

The objective (1.2) function maximizes the undiscounted value of extracted blocks. The constraint set (1.3) consists simply of block precedence relations. Figure 1.3 shows an example of an ultimate pit limit presented as block model containing waste blocks(grey) and ore blocks(bluish).

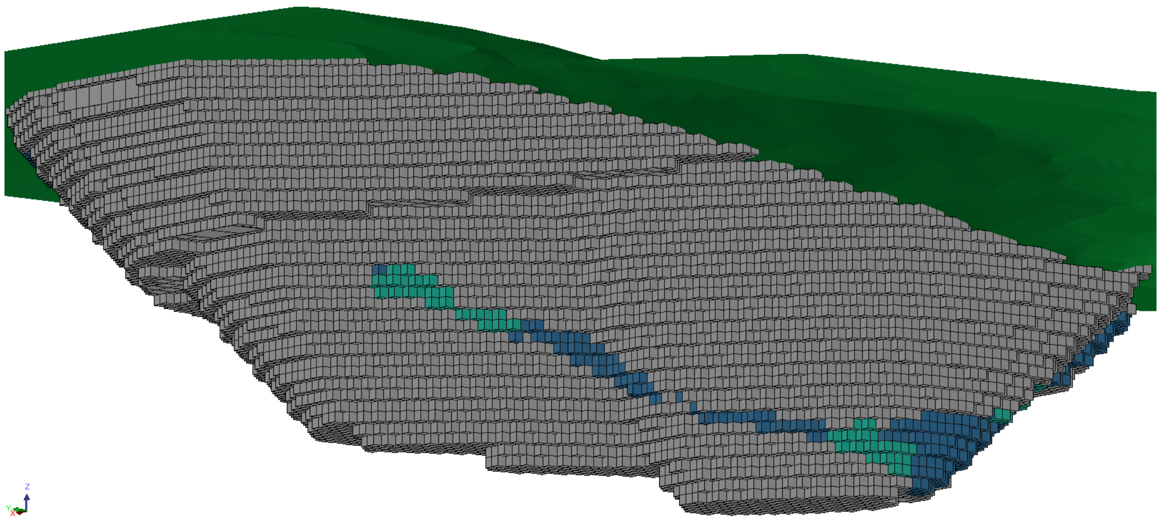

Figure 1.3 Ultimate Pit Limit

1.1.5 Strategic mine planning

The strategic mine planning consists of identifying the sequence in which the blocks must be removed from the mine with the objective of maximizing the Net Present Value (NPV), subject to technical and economic constraints. This problem can be formulated using mixed integer linear programming as follows:

Sets:

- $\mathcal{B}$: set of blocks b.
- T: set of periods t.
- $b' \in \mathcal{B}_b$: set of precedences of block b.

Parameters:

- C_{bt}: value obtained from extracting and processing block b in period t.
- m_b: tonnage block b.
- M_t^l, M_t^u: are lower and upper limit for mining capacity in period t.

Variable:

- $y_{bt} = \begin{cases} 1 \text{ if block } b \text{ is extracted in period } t, \\ 0 \text{ otherwise} \end{cases}$

$$\text{Maximize: } \sum_{b\in\mathcal{B}}\sum_{t\in T} y_{bt}C_{bt} \tag{1.5}$$

Subject to:

$$\sum_{t\in T} y_{bt} \leq 1 \qquad \forall b \in \mathcal{B} \tag{1.6}$$

$$M_t^l \leq \sum_{b\in\mathcal{B}} m_b y_{bt} \leq M_t^u \qquad \forall t \in T \tag{1.7}$$

$$y_{bt} \leq \sum_{\tau=1}^{t} y_{b'\tau} \qquad \forall b \in \mathcal{B}, \forall b' \in \mathcal{B}_b, t \in T \tag{1.8}$$

$$y_{bt} \in \{0,1\} \qquad \forall b \in \mathcal{B}, t \in T \tag{1.9}$$

The first set (1.6) of constraints ensure that a block can be only extracted one time. The second set (1.7) of constraints limits the number of blocks extracted during each period. The third set (1.8) of constraints ensure that a precedence constraints are validated. Additionally, a discounting factor can be applied to affect block's economic value as a function of the extraction period.

1.2 Transportation systems

1.2.1 Trucks and Shovels System (TS)

Truck sand shovel/loader is a discontinuous mining system that uses large shovels, excavators or loaders to extract and load material which is transported by off-road trucks to the processing plant or waste dump.

1.2.2 In-pit Crushing and Conveying System (IPCC)

Crushing operation can take place inside the mine using a fixed, either a semi-mobile or a fully mobile crusher. The crusher is fed directly by an excavation/loading equipment or also trucks dumping at the crusher feeder.The crushed material is conveyed to surface using belts.

The in-pit crushing systems developed and operated to date have varying degrees of mobility ranging from fully mobile units to permanently fixed plants, which resemble traditional in-ground crushing plants. The crushing plants can be stationary (mounted on concrete foundations) or semi-mobile style, supported on steel. As the mining operation progresses, the semi-mobile crushing plants can be relocated within the mine using multi-wheeled trailers or transport crawlers.

Typically, shovels load the material in heavy-duty haul trucks that transport the material to the crushing plant. Relocating the crushing plant as the mine expands reduce the hauling distance from the working face. The following terms are presented to help distinguish the range of mobility within the generic term of in-pit crushing systems.

- Fixed Crushers: Stationary in-ground or rim-mounted crushing plants. The station in-ground crusher is installed out of the pit, at a near the edge of the pit, in a concrete structure below grade. The stationary rim-mounted crusher is usually installed for 15 or more years and is never moved. Some degree of disassembly is required to move the structure. The planned frequency of moves for a semi-fixed crusher is no fewer than 5 to 10 years.

- Semi-mobile Indirect Feed Crushing Plant: The plant typically consists of three major modules: the apron feeder, the crushing plant with the crusher, and a separate tower that houses the control room. The control room module is bolted to the crusher module when the plant is moved. The construction works required to install the crusher are relatively simple and offsets the cost of an apron feeder. The crusher is typically located near the centroid of the working portion of the mine to minimize truck haul distance. The planned frequency of moves for a movable crusher is between 1 and 5 years.

- Semi-mobile Direct Dump Crushing Plant: The semi-mobile direct-dump crushing plant is mounted on a steel structure that houses all of the auxiliary equipment and subsystems to operate the crusher. The structure is self-supporting and rests on the mine floor, either with or without footers. The plant design allows for two or three dump points. To minimize truck haul distance, the crusher is typically located near the centroid of the working portion of the mine. The planned frequency of moves for a movable crusher is between 3 and 10 years.

- Fully Mobile Crushing Plant: The fully mobile crusher is mounted on a steel platform and is self-propelled. The platform houses all auxiliary equipment and subsystems to operate the crusher and is self-supported and rests on the mine floor. To minimize truck or front-end loader haulage, the crusher is located at the working face. Wheels, crawlers, or pneumatic pads are integrated into the platform, and drive power to move the equipment is included on board. The planned frequency of moves for a fully mobile crusher is between 1 day and 1 week.

Strategic mine planning using TS transportation system lies in maximize NPV following a set of operational constraints. Introducing IPCC in strategic mine planning starts the need to formulate additional constraints related to pit geometry, belt conveyor location and crusher position which limits production schedule flexibility. Adding these constraints increase the difficulty to solve a large scale strategic mine planning problem.

1.3 Problem Statement

In the last thirty years, technical publications raised concerns in truck operating costs in open-pit mines, in low mineral prices, and in rapid climb of mining costs. Efforts to improve truck productivity are focus on the machine itself, i.e. increasing power and capacity aimed to optimize the truck utilization and to minimize cost. By increasing the truck size capacity there is a reduction of operating costs per tonne but, a much higher investment cost for these trucks. These efforts resulted in marginal improvements and no serious innovation can be anticipated.

A different approach to decrease transportation costs is to reduce truck haulage distance by dumping in-pit; this method requires an IPCC system. IPCC systems has low operational costs due to its equipment which is powered by electricity and the high reliability of belt conveyors make IPCC systems more attractive to be used in large open-pit mine operations.

Implementing IPCC systems in strategic mining planning has been conducted through strict methodologies, for example, Figure (1.4) presents a common method used which is based mostly on the opinions and experiences of professionals working with these systems. A static mine plan is used as input to implement the IPCC system and due to the sequential nature of this methodology the proposed solutions are not always optimal.

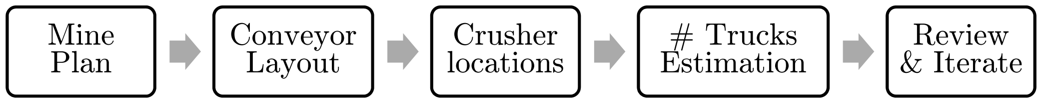

Figure 1.4 IPCC traditional methodology

The implementation of IPCC systems are related mainly to mine scheduling because it affects the success of a mine operation in searching for a reduction in transportation cost. This research aims at modeling and solving an IPCC system implementation by solving a mine scheduling with a facility location model. The model includes equipment estimation to be used as a main constrains and takes into account pit geometry, haulage distances among other constrains.

The trucks parameters, conveyor layout and possible crusher locations can all be incorporated within the strategic mine planning model, as presented schematically in Figure (1.5). Proposing a methodology that unifies these constraints was the motivation for this thesis.

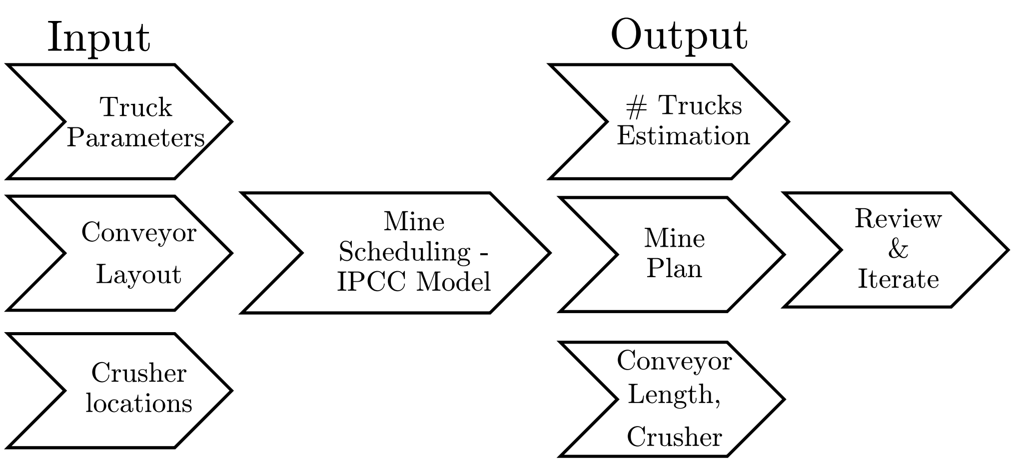

Figure 1.5 IPCC proposed methodology

1.4 Objectives

1.4.1 General

In this research, a linear programming model will be developed to take into account the use of an in-pit crusher in a long-term mine planning problem. In this model is included all mining costs such as blasting, loading, hauling and crusher relocation. It will adapt the previous planning model integrating the particularities of a semi-continuous mining system.

1.4.2 Specifics

Since the main problem of long-term mine planning is to evaluate the feasibility related to a functional operating plan in open-pit mines, the model will be dealing, among other, with the constraints related to belt conveyor construction, crusher allocation, equipment productivity, equipment movement, stockpile and blending among other constraints. The objectives of this thesis are:

- To develop a linear programming model considering the allocation and movements of an in-pit crusher in a deterministic approach. This model must provide a detailed and an applicable long-term plan for an open-pit mine using an IPCC system.

- To solve the model using real data from an iron mine project. This model can maximize the NPV while taking into account the linear programming constraints according to the project requirements.

- To verify and analyze the results in the linear programming model. By considering different scenarios where the main operational constraints and IPCC layout will be modified to know the bounds of the system.

1.5 Originality

The mathematical model to be developed in this research consists of a mine scheduling formulation that includes an IPCC system evaluation and implementation. This procedure lies on steps which allow the mathematical model to achieve a dynamic optimal solution that results in a long-term mine plan with an optimal IPCC system implementation.

The proposed model diverges particularly from others by considering the existence of a belt conveyor layout that represents the possible transportation system of the material already crushed in-pit; it is based on the location of the in-pit crusher which may change during the planning horizon reducing transportation costs.

1.6 Thesis Outline

In Chapter 1, an overview of the problem and the research objectives are presented. Chapter 2 contains the literature review that offers an overview of common methodologies and approaches used in studying open pit mine scheduling for IPCC systems implementation. Chapter 3 describes the IPCC transportation system selection. Chapter 4 proposes the linear programming model. An application of proposed model for an iron mine project is presented in Chapter 5. Finally, Chapter 6 presents conclusions and suggestions for future work.

CHAPTER 2 LITERATURE REVIEW

The literature on IPCC systems and long term mine scheduling problem is reviewed in this chapter. In the first part, different IPCC studies that have been focused in operational and financial parameters are reviewed, then the facility location problem is introduced to review the literature on IPCC implementation, mainly from mine scheduling point of view. The second part provides a review on different mine scheduling models that have been used to include operational constraints and problem size reduction. This chapter concludes with a summary of the reviewed literature and remarks on the IPCC mine planning research gap.

2.1 IPCC

Since 1980, several studies performed on existing in-pit crushing installations and models have showed cost savings (Darling, 2011). As shown in Table 2.1, Yakovlev et al. (2016) summarize the development of IPCC technology combined with TS for open-pit mining. As a conclusion, there is a tendency in the direction of high speed, large capacity conveyors, and well as these systems have demonstrated very productive.

Classic belt conveyors can transport materials at angles up to 37°. This type of conveyor has been used in numerous hard rock operations in Canada with a maximum capacity of 57,500 tonnes per day (de la Vergne, 2014). High angle belt conveyors, such as the sandwich and the pocket wall conveyors can transport material at high angles up to 90° while keeping the positive features of conventional conveyors (Duncan and Levitt, 1990). The Conveyor Equipment Manufacturers Association (CEMA) recommends to design belt conveyors with a maximum width of 3.2 m and belt speeds of 8 m/s for rough crushed material. Refer to “Belt Conveyors for Bulk Materials” (CEMA, 2002).

Dean et al. (2015) reviewed factors affecting the implementation of an IPCC system composed by fully mobile crushers and backhoe excavators. The main factors identified were: capital expenditure, system flexibility, selectivity and mine planning. In order to minimize the use of additional belt conveyor extensions, mine planning was classified as the key factor. Beside, the study presents a new mining method sequence. This method incorporates a sequence of parallel and circular pushbacks to optimize IPCC productivity, reduce belt conveyor length and use of TS system at pit bottom. Yakovlev et al. (2016) analyzed technical and economical indicators like haulage distance, power consumption, operating costs, and capital investment. The methodology consisted in calculating parameters for an IPCC and TS system separately

Table 2.1 Development of IPCC technology in open-pit mining. Adapted from Yakovlev et al. (2016)

Years	**Quantity volume, Mt (minerals)**	**Transportation distance, km**		**Equipment**
		Collecting transport	**Conveying transport**	
1945 -1960	4–5 (copper and iron ore)	0.3–1 (sometimes 3)	0.3-3 (maximum 12)	- Shovels: bucket 3–4 m3 - Dump trucks: 20–45 tonnes - Jaw crushers - Belt width: 760 and 914 mm - Belt velocity: up to 3 m/s
1961 -1970	20–25 (copper and iron ore)	0.4-2 (sometimes 3)	0.4-3.8 (maximum 15.4)	- Shovels: 11, 19, 23 m3 - Dump trucks: 65–120 tonnes - Cone crushers - Belt width: 814–1524 mm - Belt velocity: 2–4 m/s
1970 -2000	22–36 (copper and iron ore, hard overburden)	1.2-2.5	1.5-3	- Shovels: bucket 8, 19, 23 m3 - Dump trucks: 75–138 tonnes - Mobile crushing-and-loading units - Belt width 1600–2000 mm - Belt velocity to 4–5 m/s - High-angle belt conveyors
After 2000	IPCC is commonly used. The IPCC units are introduced into operation for hard overburden handling, crushing-and-loading units with high-angles and double conveyors at hoist angle 37° and hoist height 270 m. The basic trends of development in IPCC continues.			

and, based on crusher position relocation. Authors concluded that an IPCC system for deep open-pit mine is suitable from the beginning of the project. Londoño et al. (2014) evaluated parallel conveying alternatives for in-pit crusher conveyor systems in overburden material. They confirmed that using a parallel conveyor and spreader, the productivity of the system can improve between 9% – 12%. A parallel conveyor system has a better equivalent unit cost than a single conveyor system according to the authors.

Czaplicki (2004) focus on identifying reliability indexes for a TS configuration with in-pit crusher conveyor system. Equipment reliability was calculated by using operational parameters for this system. Author concludes that conventional belt conveyors have high reliability and the crusher is the equipment with the lowest reliability of the system.

McKenzie et al. (2008) study the use of at-face fully mobile crusher and a belt conveyor system. Authors developed a model to determine the movement frequency from the mining face to a crusher fed by a wheel loader. This model consists in a network of nodes and arcs representing the crusher movement through the mine. In order to reduce the loader cycle time, a shortest path algorithm was used to solve the model. Based on this study, it is important to remark that the modeled constraints are very similar to a fully-mobile crusher operation because belt conveyor extensions are added based on transportation and belt extension cost. The authors claim that verified costs savings are near to 33 % compared with the previous operation without using the algorithm through mine planning process. In Que et al. (2015), the ground articulating pipeline system used to transport oil sands slurry from mining faces was studied as an alternative to conventional TS system for oil sands mining. Authors examined the interaction between shovels and continuous mining transfer systems. The interactions between these equipments are complex to model as a linear programming. They studied those interactions using a discrete event simulation model. Conclusions indicated that shovel capacity is more important than shovel cycle time, and more significant than transport system throughput.

Mine transportation problems target to minimize equipment purchases and operational costs. Mining transportation models should be strongly integrated with design and mine scheduling models. Besides, some authors incorporated material haulage as an adjusted cost or a separate model with an iterative methodology looking for a satisfactory solution.

Kawalec (2008) presented a multistage procedure to create a feasible final pit to implement an in-pit conveying system. Author redefined the transportation cost by including energy cost based on block geometrical position for transportation the to the plant or to the waste dump. Horizontal and inclined distances were the basis to estimate power consumption required to haulage material in-pit and ex-pit. By using the final pit, the author generated different mine sequences. These were constrained by circular or parallel mining advance directions that evaluate different crusher locations based on indicators such as mineral production, stripping ratio, NPV and cash flow.

According to Que et al. (2015), a number of researchers have considered how to determine the best position for an in-pit semi-mobile crusher in which ore is transferred to a belt conveyor to be transport out of the mine site. Most of the studies about this problem have been completed to find the best position for a crusher for long term plan horizon while scheduling, mining, stacking and processing are simple constraints.

de Werk et al. (2016) conducted an economical evaluation between TS and IPCC systems on a theoretical open-pit as an inverted cone model. The evaluation focused on capital investments and operation costs incorporating technical and operational parameters such as: transition from pure trucks to conveyor, equipment purchases, infrastructure acquisition and crusher relocations. Authors concluded that capital investment for IPCC system is greater than TS system. When mine goes deeper, the IPCC operational costs are considerable cheaper compared with TS, and the total NPV increases throughout project life. An analysis showed that both systems were sensitive to production rate. On one side, fuel prices fluctuation impact TS system operating cost and on the other side conveyors use electricity with stable prices and high mechanical availability.

A recurrent methodology to handle the issue of implementing an IPCC system is to build a progressive-iterative procedure which contains the following non-sequential steps:

- Solve the ultimate pit and/or mine scheduling problem including IPCC operational parameters and estimated transportation system costs.
- Draw a belt conveyor layout and choose feasible in-pit crusher positions based on technical, economical and environmental factors. These positions are mostly selected to decrease haulage distance.
- Evaluate period by period the best in-pit crusher location based on reduction of transportation costs from mining faces to crusher position, and infrastructure investments of belt extensions and crusher relocations.
- Review the solution considering economic and operational parameters and, if needed, redesign the final pit or the mine topography at specific periods of the mine plan to optimize the selected crusher locations as well as the equipment purchases.

By using the described methodology, the best in-pit crusher location is statically determined for a specific period. But in a real operation, the mining faces are dynamic through time and a fixed in-pit crusher location does not respond to changes neither in the material flow nor in the capital investments over the planning horizon.

Konak et al. (2007) presented a decision-taken process to study in-pit crusher location and selection. To solve this combinatory problem, they developed a method to evaluate the relocation of an in-pit crusher in each bench for every period in order to minimize haulage distance cost. The increasing number of combinations after each possible relocation made this procedure very complex. In order to reduce the computational time, authors performed a maximum of three crusher relocations. This paper emphasizes on pit geometry, crusher location and shortening trucks distance as the key factors to start using a belt conveyor under economic and mine planning constraints.

Recently, IPCC problems got the attention of researchers using facility location models. Facility Location models are intended to find the optimal location of a finite set of facilities that minimize the transportation costs to a finite set of customers.

Uncapacitated version of the facility location problem is a formulation that presume that each facility can produce and transport unlimited quantities of the commodity under consideration. Single period facility location problem can be modeled using Integer Programming (IP):

Sets:

- I: set of customers i.
- J: set of facilities j.

Parameters:

- p = number of facilities operating.
- f_j: cost of establishment facility j.
- c^t_{ij}: transport cost to customer i from facility j.

Variables:

- $q_j = \begin{cases} 1 \text{ if facility } j \text{ is used,} \\ 0 \text{ otherwise} \end{cases}$
- $x_{ij} = \begin{cases} 1 \text{ if customer } i \text{ is supplied for facility } j, \\ 0 \text{ otherwise} \end{cases}$

$$\text{Minimize: } \sum_{j \in J} f_j q_j + \sum_{i \in I} \sum_{j \in J} c^t_{ij} x_{ij} \tag{2.1}$$

Subject to:

$$\sum_{j \in J} x_{ij} = 1 \qquad \forall i \in I \tag{2.2}$$

$$\sum_{j \in J} q_j \leq p \tag{2.3}$$

$$x_{ij} \leq q_j \qquad \forall i \in I, \forall j \in J \tag{2.4}$$

$$q_j, x_{ij} \in \{0, 1\} \qquad \forall i \in I, j \in J \tag{2.5}$$

First set (2.2) of constraints ensures that all customers are assigned to a facility. Second set (2.3) of constraints limits the number of operating facilities to p. Finally the third set (2.4) of constraints ensures that a customer i can be supplied from a facility $j \in J$ only if a facility is established at location j.

Figure 2.1 presents the configuration for an in-pit crusher location problem, where crushers are facilities and blocks the customers.

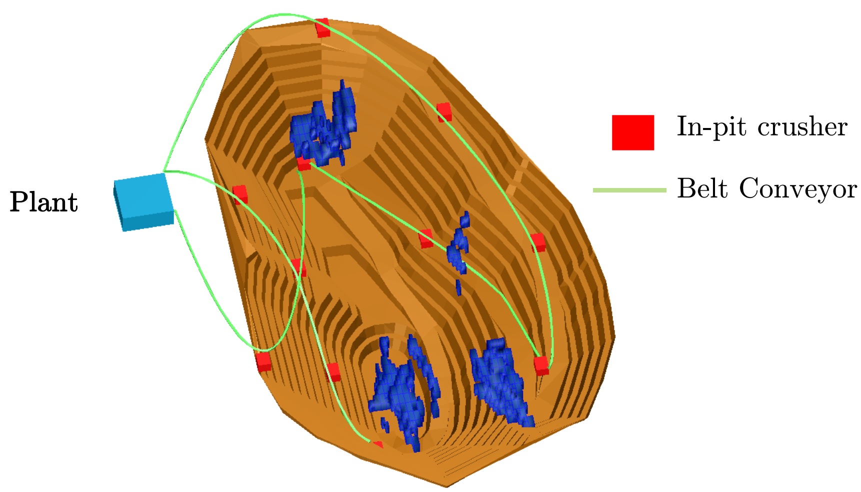

Figure 2.1 Crusher location problem depiction

Roumpos et al. (2014) presented a procedure to find an optimal fixed in-pit crusher location under a spatial analysis perspective. This procedure was associated to the family of location problems and it is also classified as a p-median model. This model minimizes costs that depend of Euclidean distance between mining pushback perimeters and possible crusher locations. The results of this study indicated a preferred location near to the pit border. The procedure applied can be used for continuous and non-continuous mining equipment, and complex cases regarding pit geometry or mineral variability. Also, a modification on elevation increases the haulage cost but there still a reduction on the total mining cost.

Rahmanpour et al. (2014) presented a facility location model to determine the optimum location of a semi-mobile crusher for a copper mine; however, the problem was solved for every period based on preassigned material flows from the mine scheduling. Paricheh and Osanloo (2016) used a stochastic p-median static facility location model to allocate an in-pit crusher in every period including production requirements and haulage cost uncertainties.

The single period methodology to allocate an IPCC results in a short-sighted and not optimal solutions. The location of an in-pit crusher is a dynamic problem that needs to consider the option of relocating the in-pit crusher more than once during the planning horizon. This means that an in-pit crusher location can be installed, unmounted and reinstalled many times as needed; it decreases the transportation of materials hauled and conveyed, and maximizes NPV.

Paricheh et al. (2016) used a dynamic facility location model based on a trial and error heuristic. They found a good solution to allocate an in-pit crusher for a specific mining sequence. Also, the transportation costs of TS and IPCC systems were calculated to determine the optimal transition period from TS system to IPCC system. Then, the ultimate pit limit and mine scheduling were recalculated using a dynamic facility location model to obtain crusher relocations over the mine planning horizon.

2.2 Mine Scheduling

There is an extensive research in the area of long-term open-pit mine production scheduling using mathematical modeling. The approach using MILP models for mine scheduling, enable simultaneous optimal mine production plans on ultimate pit limit without push back phases estimation. These models use binary variables that are equal to one if the block is mined in certain period or zero, otherwise. MILP has been used in mine planning operations to cut-off grade optimization, equipment allocation, ore blending, stockpile management and process control. According to the reviewed literature, most of the researches on mine scheduling consists in selecting blocks to maximize NPV value.

Caccetta and Hill (2003) proposed a MILP model that is a reference in the mining industry. The objective of this model is to maximize the profits over the sequenced blocks. Authors added constraints on extraction sequence, mining, milling, refining capacities, grades of mill and concentrates, stockpiles and operational conditions such as pit bottom width and depth limit. To solve this model, the authors use a branch-and-cut strategy. However, due to confidentiality contracts, they only presented key features; the fully details of the branch and cut solution strategy were not delivered. In Bley et al. (2010), a set of precedence constraints were defined based on a knapsack problem to identify the earliest period in which a block can be mined. Authors defined another set including all predecessor blocks for each block. This set allows to calculate the total tonnage preceding the extraction of each block; the preceded total tonnage is compared to the accumulated mining capacity for this period. If preceded tonnage is greater than mining capacity, this block cannot be mined until a later period. All continuous and integer decision variables associated to this block are changed to zero for the evaluated period. Then some decision variables are eliminated from the model before solution.

Eivazy and Askari-Nasab (2012) presented a MILP model for open-pit mine production schedule problem. The objective function minimizes the operational cost including mining, transportation, processing and rehabilitation. Also, the model includes three important improvements in the framework of mathematical models for mine production scheduling. The first one was considering multiple destinations, stockpiles, processes and waste dumps, close to a real mining operation. The second addition was taking into account the pit ramps because they were able to be included in the model. By doing so, it allowed minimize the haulage costs by selecting the best route to carry the material out of pit. And the last consideration was to include the bench mining direction.

Saavedra-Rosas et al. (2014) proposed a new IP model to produce mine plans appropriate to operational spaced requirements of equipment introducing the concept of exposed ore. A new set of variables and constraints were introduced to represent the extraction and the processing decisions. The main purpose of the news constraints was to leave sufficient ore material available at the beginning of each period.

Andrade et al. (2014) emphasized on a scheduling problem that considered the allocation of loading equipment under medium-term planning horizon for an open-pit mine. Authors proposed MILP model that included the existence of a stockpile for crushed material. They considered ore or waste materials because a mixed fleet equipment was available to every to both materials. Ore production was constrained by grade, crusher capacity, processing plant capacity and stockpile inventory. The model optimizes through the periods and targets an optimal mine scheduling plan considering loading equipment plan. Also, they found that including a ROM stockpile inventory and fluctuating production ore capacity impact considerably the solution value.

Smith (1998) developed a MILP model for production scheduling to find an optimal extraction sequence of mining blocks. To analyze this problem, two approaches were studied. The first one was to take the solution close to a long-term plan including constraints that will produce a blend within certain limits. The second one was to minimize the deviations of mined material constrained by a maximum and a minimum acceptable level of chemical elements and target production. Author concluded that was difficult to make a sequence of short-term schedules that will match the long-term production targets. They suggested that by reducing the problem size like decreasing the number of blocks can reduce the numbers of precedence constraint increasing solution speed.

Several researchers suggested that the size of real problems had made mine production integer programming models difficult to use. Consequently, it lead to increase the heuristic methods and aggregation techniques to reduce the problem size. The approach for this issue is to obtain suboptimal solutions using aggregation techniques to reduce the number of variables and constraints.

Ramazan (2007) presented the Fundamental Tree Algorithm (FT) based on linear programming to aggregate blocks of material and reduce the number of integer variables and constraints for a mixed integer programming formulation. Author indicated that FT algorithm was applied to the blocks inside a pushback calculated by any optimization method. Then, blocks are aggregated into larger units to be mined. The formulation was done by handling each FT as a block with ore material, average grade and maybe some waste material. Subsequently, binary variables are associated with FT instead of sequencing constraints blocks reduction. The FT algorithm has some important properties: all the blocks are connected between them by creating a tree, and the complete pushback is formed by those trees. As a result, a feasible solution for the linear programming formulation was created. The other important property is honoring the slope constraints even if a near tree is extracted during mining sequence.

Badiozamani and Askari-Nasab (2016) presented a solved MILP model for an oil sands mining sequence and tailings slurry management. Two techniques were used in this study to reduce the size of the problem and make it useful for real cases. Authors aggregated mining blocks as mining panels and some decision variables were fixed to zero considering the earliest and latest start times for extraction. The panel definition allowed to introduce constraints to control vertical and horizontal precedence extraction reducing the problem size.

2.3 Summary

In-pit crusher conveyor systems location are problems related to block sequencing mainly because the belt conveyor installation and crusher movement are attached to mine geometry, excavation sequence and material availability. Open-pit mine block scheduling is a problem which can affect the success of an mine operation. Some researchers have considered how to determine the best position for an in-pit semi mobile crusher in which ore is transferred to a conveyor belt to take out from the mine site. Most of the studies reviewed for this problem have been completed to find the best position for a crusher over a medium - long term plan horizon while sequencing, mining, stockpiling and processing are simple fixed constraints.

The literature review determines that there are two key gaps for an in-pit crusher system implementation related to mine scheduling. The first gap is to model the block scheduling problem for semi-continuous and continuous mining transportation systems. And the second gap is the non-optimal solutions of IPCC implementation methods used by the industry and the academics.

Some considerations should be taken into account to setup a block sequencing model for semi-continuous and continuous mining transportation systems. The first one is that all mining rules must be modeled; these rules include: crusher movement, conveyor belt considerations, equipment productivity, waste extraction management, and other operational parameters. The second one is related to financial parameters such as capital investment and operational costs. These parameters will need to be included in the mine scheduling model to include financial boundaries that are not included in the operational point of view.

The expected contribution of this study is to develop a practical linear programming model of open-pit block sequencing for an in-pit crusher conveyor system over the long term planning horizon.

CHAPTER 3 IN-PIT CRUSHER TRANSPORTATION SYSTEM

The IPCC system implementation is based on the assumption that the material mined from different areas increases the mining transportation costs as the pit expands and deepens. The goal to implement an IPCC system is to minimize mining costs focusing on transportation while meeting the different constraints and requirements. Some considerations related to the mineral deposit and the final pit geometry need to be addressed; these considerations are key for the practicality of formulation presented in this thesis.

As part of the study to be developed, a comparative analysis at scope level between mine transportation systems is elaborated for the iron project. Also, the selected transportation system is going to be modeled as a linear programming formulation to obtain a mine scheduling plan under the presence of the chosen transportation system.

Several methods like analytic hierarchy process (AHP), multiple attribute decision making (MADM) and fuzzy multiple attribute decision making (FMADM) assist the decision maker in selecting an appropriate alternative from a set of viable alternatives using multiple selection criteria and different criteria priorities.

The process is usually iterative in nature looking at many possible approaches and determining how all the variables interact between them. Now, mining companies and consultants use detailed and sophisticated models that incorporate all the technical and financial data providing detailed outputs such as mine and mill production, direct and indirect costs, taxes and royalties, cash flows, internal rate of return, and NPV for each considered alternative.

3.1 Analytic Hierarchy Process - AHP

The AHP is based on the multi-criteria decision making principle where the most suitable alternative is selected out of a group of available alternatives on the basis of a defined number of decision making criteria. This method is particularly suitable in cases when there is not enough information on the reviewed alternatives in the decision making(Saaty, 1990).

In this study, to select a suitable transportation system, an Analytic Hierarchy Process (AHP) is going to be used, considering the specific relative importance of each of the main characteristics and requirements of the project. Some examples in mining applications of this method are presented in Table 3.1.(Ataei et al., 2008)

This methodology consists of defining the problem, determining the relative importance of the criteria attributes, calculating the relative importance of each of the alternatives with respect

Table 3.1 Applications of AHP in mining engineering. Adapted and updated from Ataei et al. (2008)

Application areas	**No. of alternatives**	**Proposed by**
Site selection for limestone quarry expansion	3	Dey and Ramcharan, 2008
Optimum support design selection	9	Yavuz, 2015
Environmental reclamation of an open-pit mine	4	Bascetin, 2007
Underground Mining Method Selection	5	Alpay and Yavuz, 2009
Rock mass classification on tunnel engineering	3	Chen and Liu, 2007
Alumina–cement plant location	5	Ataei, 2005
Equipment selection at open-pit mine	4	Bascetin, 2004
Mining method selection	7	Bitarafan and Ataei, 2004
Implementation of the AHP with VBA in ArcGIS	2	Marinoni, 2004
Drilling waste discharges	8	Sadiq et al., 2004
Optimal equipment selection in open-pit mining	4	Ataç, 2003
Selection of opencast mining equipment	5	Samanta et al., 2002
Evaluating the environmental impact of products	6	Hertwich et al., 1997
Mining method selection by PROMETHEE	13	Bogdanovic et al., 2012
AHP for underground mining method selection	6	Gupta and Kumar, 2012
Yager's method in underground mining method	5	Yavuz, 2015
Monte Carlo AHP selecting optimum mining method	11	Ataei et al., 2013

to each criteria attribute and, estimating the priority weight of each of these alternatives. The procedure suggested for the AHP is:

1. State the problem and identify the criteria that influence the decision to take. Structure the problem establishing objective, criteria and alternatives.

2. Compare each criterion in terms of pairs respect to a given criterion. The pairwise comparison is based on intuition, data or previous analysis and experiences. The procedure consists of comparing n criterion creating an $n \times n$ matrix with elements of the diagonal equal to 1. The matrix must be constructed using the Table 3.2; the rest of comparisons are reciprocal of the previous comparisons.

3. Given the pair comparison matrix then calculate the eigenvalue $\lambda_{\max}$ and eigenvector $w = (w_1, w_2, \ldots, w_j)$, where weights are estimated as relative priorities of criteria. The eigenvector of a reciprocal square matrix can be determinate by normalization of

Table 3.2 Pairwise comparisons scale

Relative intensity	**Definition**	**Explanation**
1	Of equal value	Two requirements are of equal value
3	Slightly more value	Experience slightly favors one requirement over another
5	Essential or strong value	Experience strongly favors one requirement over another
7	Very strong value	A requirement is strongly favored and its dominance is demonstrated in practice
9	Extreme value	The evidence favoring one over another is of the highest possible order of affirmation
2, 4, 6, 8	Intermediate values	When compromise is needed

a column vector of the matrix.

$$\lambda_{\max} = \frac{1}{n} \sum_{i=1}^{n} \left\{ \frac{\sum_{j=1}^{n} a_{ij} \times w_j}{w_i} \right\} \tag{3.1}$$

Where $\lambda_{\max}$ is the maximal eigen vectors, and n is the square matrix size, a_{ij} is an element of pairwise comparison matrix, w_j and w_i is the jth and ith element of eigen values, respectively.

4. Since the comparison is based on the subjective evaluation, a Consistency Index (CI) and Consistency Ratio (CR) for each criteria or alternative is needed to evaluate the analysis.

   AHP Consistency Ratio (CR) reflects the consistency of the pairwise judgments. For example, if X is considered more important than Y, and Y more important than Z, then X should be more important than Z. If the judge rates X as important as Z, the comparisons are inconsistent and the judge should go back to the analysis.

$$CI = \frac{\lambda_{\max} - n}{n - 1} \tag{3.2}$$

To determine whether CI is acceptable, the CR must be estimated.

$$CR = \frac{CI}{RI} \tag{3.3}$$

Where RI is the random indices. RI values are given in Table 3.3:

Table 3.3 Random indices of randomly generated reciprocal matrices

Order of the matrix	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
RI value	0.00	0.00	0.58	0.90	1.12	1.24	1.32	1.41	1.45	1.49	1.51	1.48	1.56	1.57	1.59

As a general rule, a CR of 0.10 or less is considered acceptable (Saaty, 1990). In practice, however, a CR exceeding 0.10 occurs often.

3.1.1 Problem statement

An iron deposit with sub-horizontal seams is ideal for a long and a deep open-pit mining with low stripping ratio. Trucks and shovels are going to be used as a mining method due to the large number of tonnages expected to be mined. Due to the long project life, the implementation of a semi-continuous or continuous transportation system makes an interesting option to be evaluated. Thus, taking into account the need of purchasing additional trucks among other important cost, the selected transportation system can lower its costs over the project life.

If an IPCC system can be implemented, then the evaluation of using in-pit crusher has to be integrated to reduce mining costs.

3.1.2 Alternatives proposal

The problem to evaluate consist in selecting a suitable transportation system for an iron ore deposit based on technical and economical characteristics. Consequently, three cases will be considered:

- Case 1: Semi-mobile crusher in-pit and material transported by belt conveyor to ex-pit
- Case 2: Semi-mobile crusher in-pit and material transported by rail-veyor to ex-pit
- Case 3: Full-mobile crusher in-pit and material transported by belt conveyor to ex-pit

3.1.3 Identification of criteria

There are too many factors that have a relation with mining transportation system selection such as geological and geotechnical properties, mine geometry, production requirements, economic parameters and system particularities. As well, it is very difficult to define the determining criteria for transportation system selection. In general, operational costs, mine geometry, haulage distance, and production rate are the most important criteria that define transportation system in mining. The criteria to be evaluated are presented below:

1. Production rate
2. Flexibility: Overproduction or underproduction can be achieved
3. Relocation time: Production can be affected by system relocation
4. CAPEX: Capital costs
5. OPEX: Operational costs
6. Mine geometry: Pit geometry, topography, deep, haul road, face length and ground condition
7. Material movement: Transportation distances, dumping levels, dump configuration
8. Material properties: Material size, density and swell factor, material moisture, mining selectivity
9. System Properties: Availability, reliability, management required, life equipment
10. Environment & Safety: Climate and weather, environmental factors, dust emission, land disturbance and industrial safety

3.1.4 Criteria comparison

Table 3.4 contains the pair-wise comparison among the criteria to select the transportation system.

Figure 3.1 shows that the key criteria impacting the transportation system evaluation are related directly with mine geometry, CAPEX and OPEX. The economic factors are the two main criteria to take into account at every stage of this study because the goal is to model a transportation system into the mine scheduling problem to generate a mine plan using the selected transportation system.

Table 3.4 Decision criteria for mining transportation system selection

Criteria	Production rate	Flexibility	Relocation time	CAPEX	OPEX	Mine geometry	Material movement	Material properties	System properties	Enviroment & Safety
Production rate	1	2	7	1	1/4	1/2	1/2	2	2	7
Flexibility	1/2	1	4	1/2	1/6	1/4	1/3	6	1	7
Relocation time	1/7	1/4	1	1/4	1/6	1/6	1/3	1	1/3	6
CAPEX	1	2	4	1	1	1/3	5	5	1	7
OPEX	4	6	6	1	1	1	1	7	3	7
Mine geometry	2	4	6	3	1	1	3	5	3	8
Material movement	2	3	3	1/5	1	1/3	1	4	1	9
Material properties	1/2	1/6	1	1/5	1/7	1/5	1/4	1	1/5	4
System properties	1/2	1	3	1	1/3	1/3	1	5	1	7
Enviroment & Safety	1/7	1/7	1/6	1/7	1/7	1/8	1/9	1/4	1/7	1

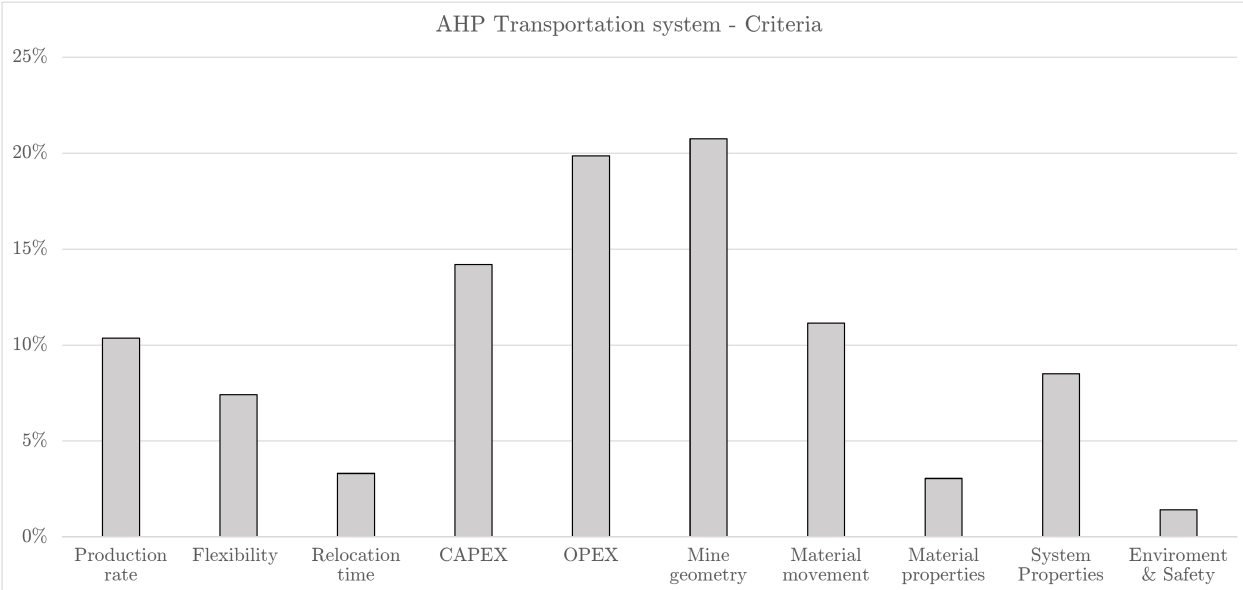

Figure 3.1 Mining transportation system criteria attributes

3.1.5 Mining transportation system selection

After determining each criterion weight, the comparison of each system is based on a particular criterion which is placed in a comparison matrix. The, ten (10) matrices were formed presenting pair-wise comparisons of the cases according to each criterion (See Annex A).

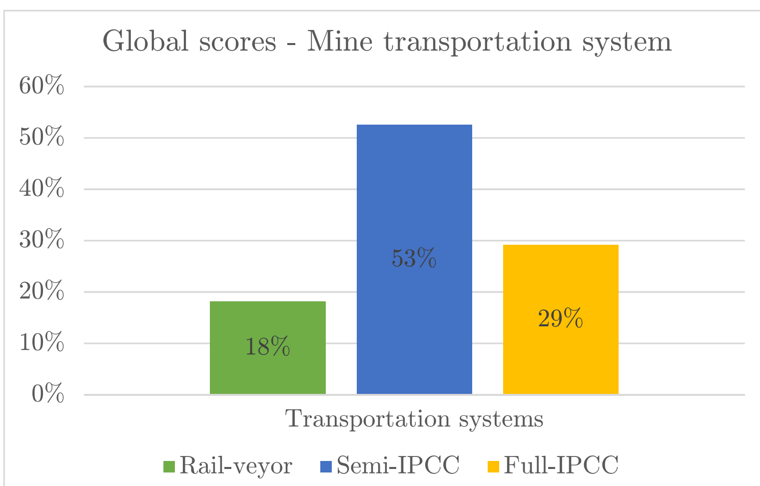

Figure 3.2 Global scores -Mining transportation systems

The final score for each of the cases is calculated by multiplying the weight of each case by the criterion. Finally, considering the cases scores, the semi-mobile IPCC system with overall priority of 53% is the best mining transportation system for the iron project. The AHP makes possible to select the transportation system, and this methodology is clear and easy to comprehend as well as to apply.

CHAPTER 4 MATHEMATICAL MODEL

This section aims to model the location problem of an in-pit crusher conveyor system using a dynamic facility location model, considering operational and financial parameters to deliver a mine plan scheduling.

4.1 Crusher location problem

In general, the decisions about crusher locations are made on a long-term basis. Once semi-mobile crushers are established, they are usually used for a couple of periods. However, factors influencing such a decision change over time. These factors are pit expansion, belt conveyor length and haulage cost, but the relocation of crushers can be quite costly. In order to cope with such issues, a dynamic crusher location model was formulated.

A Mixed Integer Linear Programming formulation to generate a long term production plan including an IPCC system is presented. The formulation maximizes NPV, and minimizes extraction costs, production deviation penalties, IPCC capital investment, and operational costs. The following notation was used:

Sets

$\mathcal{N}$	set of blocks in the model, where N=$\|\mathcal{N}\|$.
$\mathcal{A}$	set of all directed arcs in the graph of precedences denoted by $\mathcal{G}(\mathcal{N},\mathcal{A})$.
Γ_i^+	set of node i' such that $(i,i')\in\mathcal{A}$, i.e. block i' must be extracted before block i, $\Gamma_i^+=\{i'\in\mathcal{N}\|(i,i')\in\mathcal{A}\}$.
$\mathcal{J}$	set of waste dump and potential crusher location sites, $\mathcal{J}=\{1,..,J\}$, where J=$\|\mathcal{J}\|$.
$\mathcal{B}$	set of all directed arcs in the graph of belt conveyor segment precedences installation.
Γ_j^+	set of precedences on (j,i) when blocks i must be extracted before a crusher can be installed at location j, including the necessary blocks that guarantee the belt conveyor construction; $\Gamma_j^+=\{i\in\mathcal{N}\|(j,i)\in\mathcal{B}\}$

Index

$i \in \{1, ..., N\}$	index for blocks.
$j \in \{1, ..., J\}$	index for dumping locations: ex-pit crusher, in-pit crusher and waste dumps.
$t \in \{1, ..., T\}$	index for periods.

Mining parameters

T_i^O	ore tonnage of block i.
T_i^W	waste tonnage of block i.
g_i	estimated grade value of block i.
g_c	cut-off grade value.
$\overline{g}$	the average input grade to the mill.
I	interest rate. (%)
P	commodity price.
d_t	discounted factor in period t.
R^p	processing recovery rate.
c^{ODB}	drilling and blasting cost per ore tonne.
c^{OL}	loading cost per ore tonne.
c_{ij}^{OTR}	ore transportation cost by truck from block i to location j.
c_j^{OBC}	conveying cost per tonne of ore from crusher location $j \in J$ to plant or stockpile.
c^{ORH}	rehandling cost per tonne of stockpiled ore.
c^{OP}	cost processing per tonne.
c^{WDB}	drilling and blasting cost per waste tonne.
c^{WL}	loading cost per waste tonne.
c_i^{WTR}	waste transportation cost by truck from block i to waste dump.
C_{it}^{DB}	drilling and blasting cost of block i .

C_i^L	loading cost of block i .
C_{ij}^T	transportation cost of block i to location j.
C_t^{up}	the cost of underproduction.
C_t^{op}	the cost of overproduction in period t.
υ_{it}	revenue of block i in period t.

IPCC parameters

F_{jt}	cost of setup the system at location j in period t.
F_{jt}^m	cost of operate infrastructure at location j in period t.
F_{jt}^d	cost of disassemble infrastructure at location j in period t.
F_{jt}^r	cost of re-install crusher at location j in period t.
k	portion of period required to install belt conveyor segments for an in-pit crusher location.
k_j^d	portion of period required to disassemble an in-pit crusher.
k_j^r	portion of period required to re-install an in-pit crusher located at j.
D_t	the maximum number of crusher locations for ore in period t.
U_j	capacity of crusher located at j.

Trucks parameters

H_{ij}^O	the hours-truck required to transport ore from block i to location j.
H_i^W	the hours-truck required to transport waste from block i to waste dump.
CT_t	truck operational cost in period t.
PT_t	purchase price of a truck in period t.
IT_t	truck idle cost in period t.
OP	the total number of operating hours in period t.

Limits

Gl_t , Gu_t	are the allowable lower and upper limits of the head grade in period t.
Pl_t , Pu_t	are the lower and upper limits (targets production) for the designed processing plant in period t.
Ml_t^O , Mu_t^O	are lower and upper limits for ore mining capacity in tonnes in period t.
Ml_t^W , Mu_t^W	are lower and upper limits for waste mining capacity in tonnes in period t.
Sl_t , Su_t	are lower and upper limits for stockpile in tonnes in period t.

Mining variables

$$y_{it} = \begin{cases} 1 \text{ if block } i \text{ is extracted in period } t, \\ 0 \text{ otherwise} \end{cases}$$

$$z_{it} = \begin{cases} 1 \text{ if block } i \text{ is processed in period } t, \\ 0 \text{ otherwise} \end{cases}$$

x_{ijt}	continuous variable representing the portion of block i transported to location j in period t.
T_t^{up}	continuous variable representing the tonnage of underproduced ore in period t.
T_t^{op}	continuous variable representing the tonnage of overproduced ore in period t.

IPCC variables

$$s_{jt} = \begin{cases} 1 \text{ if infrastructure is installed for the first time in location } j \text{ in period } t, \\ 0 \text{ otherwise} \end{cases}$$

$$q_{jt} = \begin{cases} 1 \text{ if dumping location } j \text{ is used in period } t, \\ 0 \text{ otherwise} \end{cases}$$

$$f_{jt}^{d} = \begin{cases} 1 \text{ if dumping location } j \text{ is closed in period } t, \\ 0 \text{ otherwise} \end{cases}$$

$$f_{jt}^{r} = \begin{cases} 1 \text{ if dumping location } j \text{ is re-installed in period } t, \\ 0 \text{ otherwise} \end{cases}$$

Trucks variables

tp_t	integer variable representing the number of trucks purchased in period t.
tw_t	continuous variable representing the number of trucks working in period t.
td_t	continuous variable representing the number of trucks idle in period t.

4.1.1 Assumptions

For the development of the MILP model, the following assumptions apply:

- Ultimate pit: The ultimate pit shell has already been obtained, and the mining method has been chosen.
- Infrastructure location: Processing plant, stockpile and waste dump locations are known in advance.
- Drill and blast operations: There is no change in drill patterns and explosive consumptions due to the use of an in-pit crusher. The costs associated to rock fragmentation are constant for all materials.

- Operative hours: The operative hours are known taken into account planned downtime, blasting and weather delays.
- Transportation equipment selection: Equipment selection is assumed with it respective productivity parameters. The transportation equipment is composed by a set of trucks and belt conveyor segments which fit the specific mining requirements.
- Equipment retention: All equipment is kept until the end of the mine life without replacement or salvage.
- Belt conveyor layout: The proposed layout for the belt conveyor segments is drawn starting from the final pit boundary to the final walls reaching the pit bottom. Belt conveyor systems operate independently among them.
- Stockpile: The stockpile is employed to decrease deviation from a goal production and the overproduced ore is stockpiled and used in the next period. To maintain the linearity of this model, a single initial approximation for the average grade of the stockpile is used and it is fixed over all periods.

4.1.2 Formulation

The objective function contains four elements: Revenue, penalties for production target deviation, crusher and conveyor installation, and fleet equipment.

First of all, we defined for each block the amount of ore and waste using the Block Economic Value (BEV) concept presented in Section 1.1.2. To classify and quantify the different materials contained in each block, the cut-off grade criteria was used as follows:

- T_i^O: Ore tonnage in block i, which corresponds to the total sum of tonnages of materials with $g_i \geq g_c$ in block i.
- T_i^W: Waste tonnage in block i, which corresponds to the total sum of tonnages of materials with $g_i < g_c$ in block i.

The profit from a block in period t is equal to the revenue v_{it} generated by selling the final product contained in block i less the extraction costs. Cut-off grade is used to determine if a block is ore or waste. If the estimated grade of a block is less than the cut-off grade, the block is considered as waste. The revenue of such a block is zero.

$$v_{it} = \begin{cases} T_i^O \times \left(g_i \times R^p \times P - c^{OP}\right) & \text{if } g_i \geq g_c, \forall i \in \mathcal{N}, t \in \{1, ..., T\} \\ 0 & \text{if } g_i < g_c \end{cases} \tag{4.1}$$

The extraction costs consists of the main operations costs: drilling and blasting, loading, and transportation. Costs are estimated separately for ore and waste to have a better approximation to actual operating conditions; i.e., different designs for drill and blasting based on lithology, equipment productivity are different according to material properties and haulage distance, or waste material can be only be transported by trucks. The extraction cost per block is presented as follows:

Extraction costs = drill and blast costs + loading costs + transportation costs

$$C_i^{DB} = (c^{ODB} \times T_i^O) + (c^{WDB} \times T_i^W) \qquad \forall i \in \mathcal{N} \tag{4.2}$$

$$C_i^{L} = (c^{OL} \times T_i^O) + (c^{WL} \times T_i^W) \qquad \forall i \in \mathcal{N} \tag{4.3}$$

$$C_{ij}^{T} = \left[\left(c_{ij}^{OTR} + c_j^{OBC}\right) \times T_i^O\right] + \left(c_i^{WTR} \times T_i^W\right) \qquad \forall i \in \mathcal{N}, j \in \mathcal{J} \tag{4.4}$$

- $j = \{1\}$: If the total material contained in block i is classified as waste under the cut-off criteria, its only possible destination is the waste dump. The block will be transported by truck. This is equivalent to setting $c_1^{OBC} = 0$ and $x_{ijt} = 0$, $\forall j \in \{2, ..., J\}$, $t \in \{1, ..., T\}$.

- $j \in \{2, ..., J\}$: If a block contains "ore" or "waste and ore", is classified as ore under the cut-off criteria and block i will be sent to a crusher(i.e location j when $j \geq 2$). Blocks containing ore do not have waste dump as destination, which is equivalent to setting variable $x_{i1t} = 0$, $\forall t \in \{1, ..., T\}$. When blocks containing ore are transported by trucks to crusher location $j = \{2\}$, which is outside of the pit commonly near to the final pit boundary, called ex-pit crusher, then the ore crushed in this location only incurs transportation costs by trucks, which means that $c_2^{OBC} = 0$. This option represents the traditional mine scheduling problem using a TS fleet with a fixed ex-pit crusher. Figure 4.1 shows an example of blocks with different potential crushers destinations and their respective distance reduction compared to TS transporting material to an ex-pit crusher in the plant.

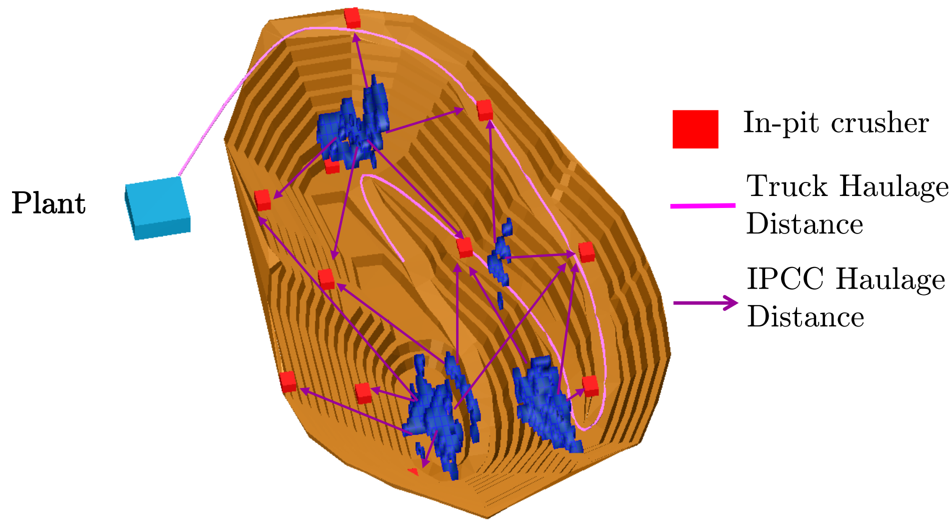

Figure 4.1 IPCC truck haulage routes

Finally in-pit crusher candidate locations $j \in \{3, ...J\}$, are only enable under excavation and installment constraints. Ore is transported by trucks to those locations, then crushed and conveyed out of the pit to be processed or stockpiled.

The profit of a block is based on the block value and the costs incurred in extracting and processing it. We represent the discount for those values incurred on certain period t using a discount factor:

$$d_t = \frac{1}{(1+I)^t} \qquad \forall t \in \{1, ..., T\} \tag{4.5}$$

The discounted profit of the project is the summation of discounted revenue minus discounted extraction cost over all periods.

$$NPV = d_t \times \sum_{t=1}^{T} \sum_{i=1}^{N} \left\{ (v_{it} \times z_{it}) - \left[(C_{it}^{DB} + C_{it}^{L}) \times y_{it} + \sum_{j=1}^{J} (C_{ijt}^{T} \times x_{ijt}) \right] \right\} \tag{4.6}$$

The approximation for the cost of under production, i.e. Cup, assumes that under production will lead to a loss of revenue due to the mill running at lower capacity. Is estimated using the following equation:

$$C^{up} = \overline{g} \times P - C^{OP} \tag{4.7}$$

The cost of underproduction is calculated over all periods except for the final period, because any mineral that is left for the final period will be processed and will probably keep the annual target production (Koushavand et al., 2014). Therefore we set $C^{up} = 0$, where is the final period or the mine life.

We can say that in this model $\overline{g}$ is the same for each period, but if a better estimation can be done for each period, then the model will allow this precision.

If an extra amount of ore is extracted, then a cost for stockpiling and rehandling C_t^{op} must be considered. this material is conveyed to the stockpile and processed in subsequent periods. Delaying the extraction of extra ore to period $t+1$ affects the value of the processed ore because of discounting, and discounting factor additionally affects the processing cost. The cost of over production have three different elements which can be described as:

Discounted cost of over production =(cost of overproduction in period t - cost of overproduction in period $t+1$) + stockpiling and rehandling cost

$$\text{Cost of overproduction in period } t\text{: } \bar{g}_t \times P - C^{OP} \qquad \forall t \in \{1, ..., T\} \tag{4.8}$$

$$\text{Cost of overproduction in period } t+1\text{: } \frac{\bar{g}_t \times P}{(1+I)} - \frac{C^{OP}}{(1+I)} \qquad \forall t \in \{1, ..., T\} \tag{4.9}$$

$$\text{Stockpiling and Rehandling cost in period } t \text{ is defined by: } C^{RH} \qquad \forall t \in \{1, ..., T\} \tag{4.10}$$

$$C_t^{op} = \left[(\bar{g}_t \times P - C^{OP}) \times \left(\frac{I}{(1+I)^t} \right) + C^{RH} \right] \qquad \forall t \in \{1, ..., T\} \tag{4.11}$$

It is assumed that any possible over produced ore that has been stockpiled at period t will be processed in period $t+1$.

The discounted cost of deviation from target production is the summation of discounted cost of under production and discounted cost of over production over all periods.

$$DEVIATION = \sum_{t=1}^{T} d_t \times [(C_t^{up} \times T_t^{up}) + (C_t^{op} \times T_t^{op})] \tag{4.12}$$

IPCC discounted costs are represented by system installation/re-open, maintenance, disassemble/close and re-installation/re-open. The parameters $F_{jt}, F_{jt}^m, F_{jt}^d, F_{jt}^r$ and binary variables $s_{jt}, q_{jt}, f_{jt}^d, f_{jt}^r$ are introduced to represent the dynamic of IPCC systems allowing multiple material destinations and crusher relocations over all periods. The elements of IPCC cost function to be minimized are:

- $F_{jt} \times s_{jt}$: When an infrastructure location j is installed/opened in period t, binary variable $s_{jt} = 1$ and a cost F_{jt} is added. There are two assumptions for waste dump and ex-pit crusher. First, $s_{11} = 1$ to allow waste extraction from period $t = 1$ and waste dump opening cost is ignored, i.e. $F_{1t} = 0, \forall t \in \{1, ..., T\}$. Second, $s_{21} = 1$ to allow the access to the initial ex-pit crusher. Moreover by setting is necessary setting $F_{2t} = 0, \forall t \in \{1, ..., T\}$ it will permit to study if the resulting mining sequence is optimal using only TS system with an ex-pit fixed crusher or if it takes advantage of the proposed IPCC system.

- $F_{jt}^m \times q_{jt}$: If a location j is used in period t, then the binary variable $q_{jt} = 1$, and the operating costs are accounted by F_{jt}^m. The waste dump location and ex-pit crusher in this study do not incur in operational/maintenance costs because they are negligible, i.e. $F_{1t}^m = 0$ and $F_{2t}^m = 0, \forall t \in \{1, ..., T\}$.

- $F_{jt}^d \times f_{jt}^d$ and $F_{jt}^r \times f_{jt}^r$: F_{jt}^d and F_{jt}^r are the costs to disassemble and re-install crusher infrastructure required to operate at location j in period t. The binary variables f_{jt}^d and f_{jt}^r allow a position to be opened and closed in the same period. Waste dump location does not require theses costs, $F_{1t}^d, F_{1t}^r = 0, \forall t \in \{1, ..., T\}$.

Based on these definitions, we define the following function, referred to as IPCC:

$$IPCC = d_t \times \sum_{j=1}^{J} \sum_{t=1}^{T} \Big[(F_{jt} \times s_{jt}) + (F_{jt}^m \times q_{jt}) + (F_{jt}^d \times f_{jt}^d) + (F_{jt}^r \times f_{jt}^r)\Big] \tag{4.13}$$

Equipment function represents truck purchases and their operating costs. It takes into account fuel consumption and productivity which are known for changing by the haulage distance between loading areas and dump locations. An integer variable tp_t keeps track of the truck purchases for each period t. However, the utilization of trucks can be represented by continuous variables tw_t and td_t based on the number of hours that the trucks were working or idling, respectively

The following function is defined as $EQUIPMENT$ and represents the operational costs and equipment purchases:

$$EQUIPMENT = d_t \times \sum_{t=1}^{T} [(PT_t \times tp_t) + (CT_t \times tw_t) + (IT_t \times td_t)] \tag{4.14}$$

4.2 Objective Function

The objective function is to maximize the discounted NPV (4.6).

$$\text{Max Z} = NPV - DEVIATION - IPCC - EQUIPMENT \tag{4.15}$$

$$\begin{aligned}\text{Max Z} = \sum_{t=1}^{T} d_t \Bigg\{ & \sum_{i=1}^{N} (v_{it} \times z_{it}) - (C_{it}^{DB} + C_{it}^{L}) \times y_{it} + \sum_{j=1}^{J} (C_{ijt}^{T} \times x_{ijt}) \\ & - [(C_t^{up} \times T_t^{up}) + (C_t^{op} \times T_t^{op})] \\ & - \sum_{j=1}^{J} \Big[(F_{jt} \times s_{jt}) + (F_{jt}^{m} \times q_{jt}) + (F_{jt}^{d} \times f_{jt}^{d}) + (F_{jt}^{r} \times f_{jt}^{r}) \Big] \\ & - [(PT_t \times tp_t) + (CT_t \times tw_t) + (IT_t \times td_t)] \Bigg\}\end{aligned} \tag{4.16}$$

4.3 Constraints

The following section describes the constraints that are applied for the presented formulation. All constraints are stated in a linear form.

Ore extraction capacity constraints: Ore extraction capacity constraints establish a lower and an upper limit on the ore tonnage that can be extracted in each period. This ore extraction capacity is affected by crusher relocation events. To integrate this into the model, a parameter a_t indicates the percentage of crushing capacity of the system which is calculated as follows:

$$a_t = 1 - \sum_{j=1}^{J} \left[(s_{jt} \times k_j) + \left(f_{jt}^d \times k_j^d \right) + \left(f_{jt}^r \times k_j^r \right) \right] \qquad \forall t \in \{1, ..., T\} \tag{4.17}$$

When a crusher is relocated, the ore production is reduced equivalently to the time required disassembling k_j^d and reinstalling k_j^r crusher located at j. The upper and lower limit of ore extraction constraints are:

$$\sum_{i=1}^{N} \left(T_i^O \times y_{it} \right) \geq Ml_t^O \times a_t \qquad \forall j \in \mathcal{J}, t \in \{1, ..., T\} \tag{4.18}$$

$$Mu_t^O \times a_t \geq \sum_{i=1}^{N} \left(T_i^O \times y_{it} \right) \qquad \forall j \in \mathcal{J}, t \in \{1, ..., T\} \tag{4.19}$$

Waste extraction capacity constraints: Waste extraction capacity constraints establish a lower and an upper limit on total waste tonnage that can be extracted in each period. The maximum waste extraction capacity is increased by a crusher relocation event. When a crusher is relocated, the waste capacity can be increased equivalently. To manage a limit in waste production an upper limit constraint is introduced.

$$\sum_{i=1}^{N} \left(T_i^W \times y_{it} \right) \geq Ml_t^W \qquad \forall j \in \mathcal{J}, t \in \{1, ..., T\} \tag{4.20}$$

$$\sum_{i=1}^{N} \left(T_i^W \times y_{it} \right) \leq Mu_t^W \times a_t \qquad \forall j \in \mathcal{J}, t \in \{1, ..., T\} \tag{4.21}$$

Processing capacity constraints: The processing capacity constraints establish an upper and an lower limit on the amount of ore tonnes to be processed in each period. The lower limit can be used to adjust the extraction if the processing plant requires a minimum tonnage to be processed.

$$\sum_{i=1}^{N} \left(T_i^O \times z_{it}\right) \geq Pl_t \qquad \forall t \in \{1, ..., T\} \tag{4.22}$$

$$\sum_{i=1}^{N} \left(T_i^O \times z_{it}\right) \leq Pu_t \qquad \forall t \in \{1, ..., T\} \tag{4.23}$$

Blending constraints: For each block characteristic(grade, benches, etc.), the blending constraints establish a lower and an upper limit in each period. Here, only the grade will be considered. Also, the importance of these constraints lies in the fact that the grade of each block controls the value of the ore sold.

$$\sum_{i=1}^{N} T_i^O \times (Gl_t - g_i) \times z_{it} \leq 0 \qquad \forall t \in \{1, ..., T\} \tag{4.24}$$

$$\sum_{i=1}^{N} T_i^O \times (g_i - Gu_t) \times z_{it} \leq 0 \qquad \forall t \in \{1, ..., T\} \tag{4.25}$$

Block extraction precedence constraints: The block precedence constraints guarantee that a block can be extracted only if a determined set of blocks has been entirely mined in a previous period or in the same period.

$$\sum_{\tau=1}^{t} y_{i\tau} \leq \sum_{\tau=1}^{t} y_{i'\tau} \qquad \forall i \in \mathcal{N}, \forall i' \in \Gamma_i^+, t \in \{1, ..., T\} \tag{4.26}$$

Extraction-processing constraints: The extraction and processing constraints enforce that each block needs to be extracted before processing.

$$z_{it} \leq \sum_{\alpha=1}^{t} y_{i\alpha} \qquad \forall i \in \mathcal{N}, t \in \{1, ..., T\} \tag{4.27}$$

Reserve constraints: The reserve constraints ensure that all blocks are extracted only once.

$$\sum_{t=1}^{T} y_{it} = 1 \qquad \forall i \in \mathcal{N} \tag{4.28}$$

Over and under production constraints: The over and under production constrains control the over and under production variables, and the stockpile notion emerges in these constraints. The first set of constraints control the under production. The second set of constraints control the overproduced ore, including the overproduction of previous period that has been transferred from the stockpile. Also, any overproduced ore from a previous period in the stockpile is conveyed to the current production period.

$$\sum_{i=1}^{N}(T_i^O \times z_{it}) + (T_{(t-1)}^{op} + T_t^{up}) \geq Pl_t \qquad \forall t \in \{1, ..., T\} \tag{4.29}$$

$$\sum_{i=1}^{N}(T_i^O \times z_{it}) + (T_{(t-1)}^{op} - T_t^{op}) \leq Pu_t \qquad \forall t \in \{1, ..., T\} \tag{4.30}$$

Upper and lower limit for the stockpile: The upper and lower limit for the stockpile in each period.

$$T_t^{op} \leq Su_t \qquad \forall t \in \{1, ..., T\} \tag{4.31}$$

$$T_t^{op} \geq Sl_t \qquad \forall t \in \{1, ..., T\} \tag{4.32}$$

Unique block extraction constraints: The unique block extraction constraints ensure only one transportation destination for each mined block.

$$\sum_{j=1}^{J} x_{ijt} = y_{it} \qquad \forall i \in \mathcal{N}, t \in \{1, ..., T\} \tag{4.33}$$

Dumping locations per period constraints: The set of constraints enables as many ore dump locations D_t as needed, including the waste dump location.

$$\sum_{j=2}^{J} q_{jt} \leq D_t \qquad \forall t \in \{1, ..., T\} \tag{4.34}$$

One time installation constraints: The one time installation constraint ensure that the installation expense is accounted only once.

$$\sum_{t=1}^{T} s_{jt} \leq 1 \qquad \forall j \in \mathcal{J} \tag{4.35}$$

Installation before operation constraints: The installation before operation set of constraints enforces the logical step by installing the infrastructure in a previous period or in the same operation period for any location.

$$\sum_{\tau=1}^{t} s_{j\tau} \geq q_{jt} \qquad \forall j \in \mathcal{J}, t \in \{1, ..., T\} \tag{4.36}$$

Location status constraints: The location status constraints guarantee if the state of any location is receiving material or not.

$$q_{jt} = q_{j(t-1)} + s_{jt} + f^r_{jt} - f^d_{jt} \qquad \forall j \in \mathcal{J}, t \in \{1, ..., T\} \tag{4.37}$$

Reinstalling a disassembled location constraints: The reinstalling of a disassembled location constraints guarantee that the reinstalling takes place only in the same or after the disassembling period.

$$\sum_{\tau=1}^{t} f^r_{j\tau} \leq \sum_{\tau=1}^{t} f^d_{j\tau} \qquad \forall j \in \mathcal{J}, t \in \{1, ..., T\} \tag{4.38}$$

Belt conveyor excavation precedence constraints: The belt conveyor excavation precedence constraints guarantee that a belt conveyor segment cannot be installed before if a determined set of blocks is entirely mined in a previous period or in the same period.

$$\sum_{\tau=1}^{t} s_{jt} \leq \sum_{\tau=1}^{t} y_{i\tau} \qquad \forall j \in \Gamma^+_j, i \in \Gamma^+_j, t \in \{1, ..., T\} \tag{4.39}$$

Belt conveyor construction order constraints: The belt conveyor construction order constraints enforces that the predetermined set of belt conveyor segments is installed in a previous period or in the same period.

$$\sum_{\tau=1}^{t} s_{jt} \leq \sum_{\tau=1}^{t} s_{j'\tau} \qquad \forall j \in \mathcal{J}, j' \in \mathcal{J} | (j, j') \in \mathcal{B} \tag{4.40}$$

Crusher Capacity constraints: The crusher capacity constraints limit the ore quantity to be crushed for each crusher j in period t.

$$\sum_{i=1}^{N} (T_i^O \times x_{ijt}) \leq q_{jt} \times U_j \qquad \forall j \in \mathcal{J}, t \in \{1, ..., T\} \tag{4.41}$$

Trucks required constraints: The trucks required constraints guarantee the number of trucks required to reach the production target in each period.

$$\sum_{i=1}^{N} x_{i1t} \times H_i^W + \sum_{i=1}^{N} \sum_{j=2}^{J} x_{ijt} \times H_{ij}^O \leq OP \times tw_t \qquad \forall t \in \{1, ..., T\} \tag{4.42}$$

Trucks balance constraints: The trucks balance constraints guarantee the purchase of the trucks needed in order to reach the production target in each period.

$$\sum_{t'=1}^{t} tp_{t'} = tw_t + td_t \qquad \forall t \in \{1, ..., T\} \tag{4.43}$$

4.4 Clustering

A cluster can be defined as a set of blocks grouped in the same bench. Given the large dimensions of geological block models and the formulation presented previously, it is necessary to implement simplifications that allow to obtain results in a manageable time. The application of clustering is recommended for optimizing large problems; clustering allows the aggregation in groups and assigns a single new variable for all aggregated data.

Clustering tools in SPSS®, version 24 (by IBM) has been used to perform k-means algorithm. The software can perform tasks as follows:

- Load and display the data by attributes.
- Clustering the data using k-means algorithm.
- Use different control parameters such as maximum cluster. number, maximum iterations and maximum number of data per cluster.
- Save cluster center coordinates which indicates which cluster number it belongs. The centers can be used as input data to estimate haulage distance.

K-means algorithm will be used as the clustering method. The clustering should be done for each bench separately. Also, blocks at different levels should not be aggregated in one cluster. The k-means clustering algorithm creates clusters based on the number of clusters desired and the input properties as similarity factors. The aggregation is based on blocks spatial location and material type:

1. X, Y coordinates of the blocks: it is important that the blocks within each cluster are not spatially separated. The coordinates of the blocks are the main input properties for the clustering algorithm.
2. A binary indicator is needed to show if a block is ore or waste. It prevents mixing the ore and waste blocks in one mining cut.

The following figures show an example of the proposed methodology. Figure 4.2 shows a bench from the block model, the colors indicate different metal grade content.

Figure 4.3 shows the final result after applying clustering using X, Y and cut-off grade as similarity factors. The black line shows that there was no aggregation of ore and waste blocks. The created clusters contains aggregated blocks in a continuous geometric shape

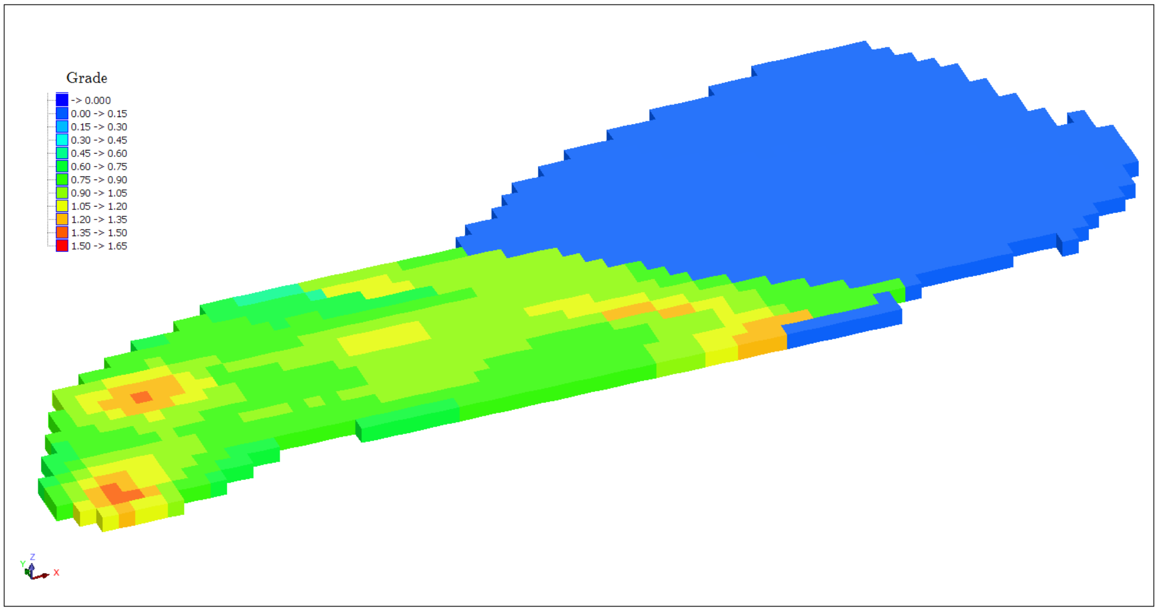

Figure 4.2 Bench displaying ore and waste blocks

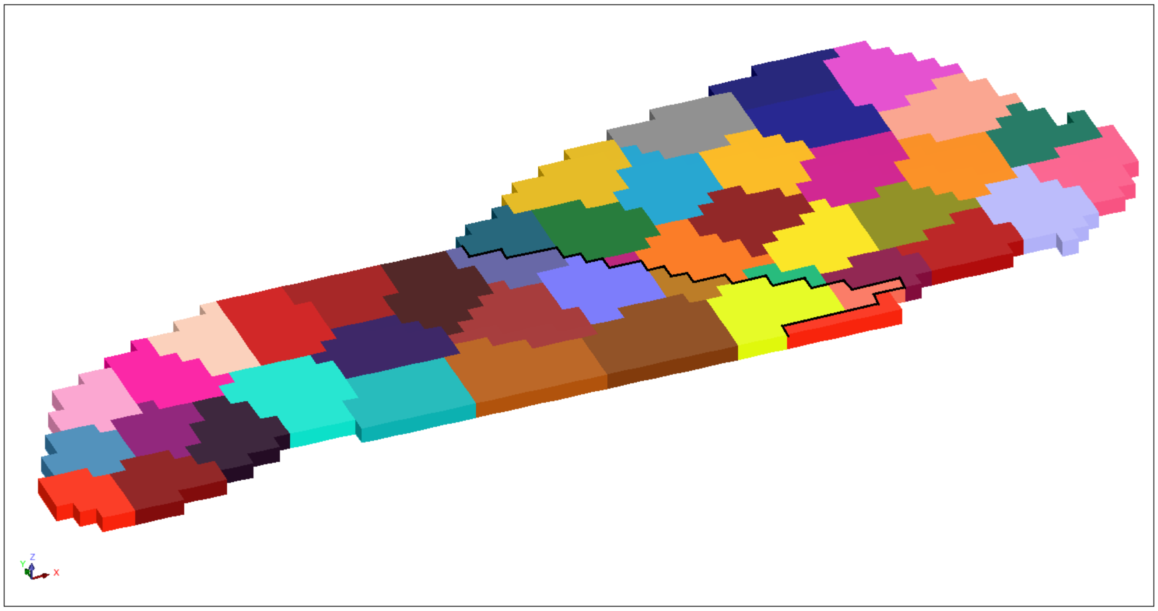

Figure 4.3 Bench displaying clusters

CHAPTER 5 STUDY CASE

5.1 LabMag project

All the presented information in this section was extracted from NI 43-101 Technical Report on the Feasibility Study on LabMag Taconite Project prepared for New Millennium Iron Corp in March 2014.

5.1.1 Geology

The project is owned by New Millennium Iron Corp in a 100 km wide belt of taconite between Quebec and Labrador. The deposit is situated about 30 km to the town of Schefferville, Quebec.

The LabMag iron deposit consists of magnetite Banded Iron Formation ("BIF") of the Lake Superior type. The formation has been broken down into individual stratigraphic units on the basis of facies; variable amounts of gangue minerals, mostly silicates, carbonates and sulphides are present. Faults are rare and where present with wrench faults parallel (Met-Chem Canada, 2014). The deposit is composed of a series of sub-horizontal layers (or seams) and approximately 120 m thick and all the sub-member units show variation in thickness. The floor of the deposit dips at approximately 6° which is a grade of 10.5% to the northeast without any significant deformation.

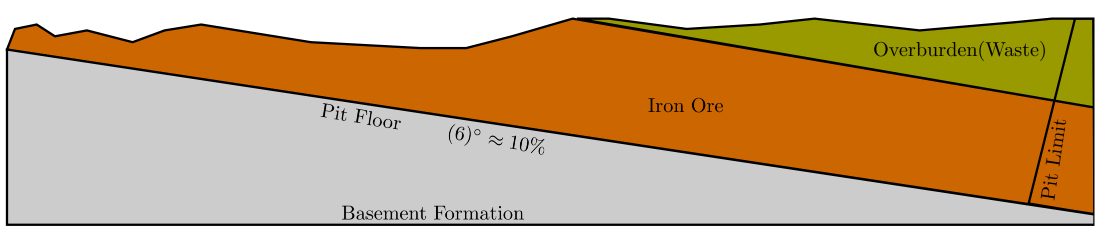

Figure 5.1 Iron formation geology

The taconite in LabMag is mostly seams of chert or jasper and magnetite. Magnetite is the dominant iron oxide mineral, hematite and martite appear in secondary amounts. The two waste materials for the LabMag deposit are overburden and Menihek Shale. Overburden covers the entire deposit but is minimal because the subjacent rock is exposed at surface. The Menihek Shale seam is existing on the north east of the deposit over the iron seams.

The average in-situ dry density for iron ore is 3.41 t/m^3. A density of 2.0 t/m^3 for the overburden and 2.94 t/m^3 for the Menihek Shale. Swell factor and moisture content affects the estimation of transportation requirements. A swell factor of 30% of material is expected from its in-situ state to after it is blasted and loaded into the trucks. The moisture content of 2% reflects the amount of water within the rock formation. Those value are typical from similar projects in the area.

To process the taconite extracting iron, fine crushing is required because taconite is a hard rock. Magnetic separation is used to separate the ore from the waste posteriorly.

5.1.2 Blending lithologies

For a productive IPCC mining operation, it is typical for contamination to occur between the ore and waste at the contact boundary. This is due to the nature of the large size of shovels and the fact that the rock requires blasting along operational constraints(Met-Chem Canada, 2014). The following method was used to estimate mining dilution and ore loss:

- Blending of lithological layers within each block in the resource model;
- Dilution and ore losses at the ore/waste contacts.

Since the open pit was designed with 15 m high benches that will be drilled, blasted and mined in one pass, it will be difficult to effectively separate the different lithological layers at the shovel face. These layers are often only several meters thick.

To account for this in the mine plan, an average grade was calculated for each block in the resource model based on the individual grades for each lithology within the block. The cut-off grade criteria for Davis Tube Weight Recovery (DTWR) and SiO_2 were then applied on the average grade items for each block to classify it as an ore or waste block.

The blending of lithologies at the shovel face induces a certain amount of dilution since thin zones that would not meet the cut-off criteria on their own are blended with the rest of the block and sent to the crusher as ore. The net result of this type of mining dilution is an increase in ore tonnage with decreased average weight recovery. The second area where mining dilution will occur is at the ore/waste contacts. Due to the fact that the mining operation will incorporate drilling and blasting and that the loading equipment is considerably large, it will be very difficult to perfectly separate the ore and waste at the geological contact. The two main areas where the orebody follows waste contacts are at the top (Overburden and Interburden) and the bottom (Lower Iron Formation).

Figure 5.2 presents a section of a typical block in the geological model. The Figure illustrates how the three seams in the block are diluted to arrive at a weighted average weight recovery of 23%. Prior to dilution, only 85% of the block meets the cut-off criteria. After dilution is accounted for, the entire block (100%) is considered as ore.

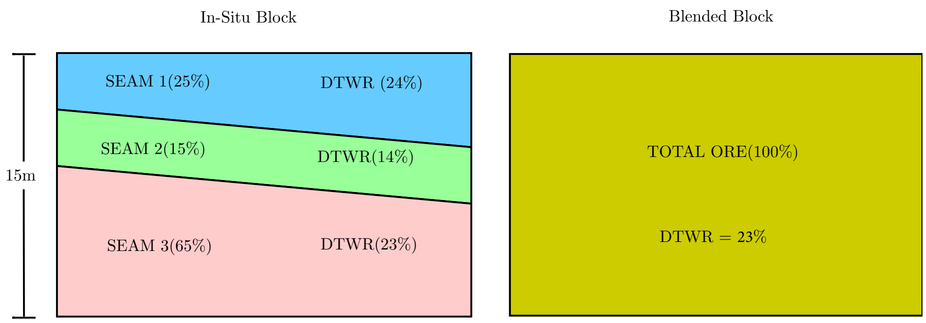

Figure 5.2 Blending lithology

5.1.3 Mine design

The designed pit is approximately 10 km long and 1.2 km wide; the height of the final wall along the east side of the deposit ranges from a minimum of 120 m to a maximum depth of 173 m. The pit floor follows the bottom of the lower iron formation; this floor reflects mineralization that does not meet the cut-off criteria. Therefore, it is not worth mining. The pit will be mined at 15 m height benches which is well suited for equipment size (shovels and drills) that are planned to be used.

Since the ore body outcrops, the final pit does not require the design of a permanent access ramp to the pit bottom. The benches will be mined flat and the pit access will be developed along the floor as the pit wall advances. The floor of the deposit dips at approximately a grade of 10%. Although this grade is not optimal for the haul trucks, it is manageable according to manufacturers like Caterpillar (Caterpillar, 2015). Alos, temporary ramps will be required in order to maintain access to the benches in the advancing wall. These temporary ramps will be built to have a maximum grade of 10%.

The total surface area of the pit is roughly 20 km^2, plant site, tailings facilities, dumps and stockpiles were intentionally placed outside the iron formation limit.

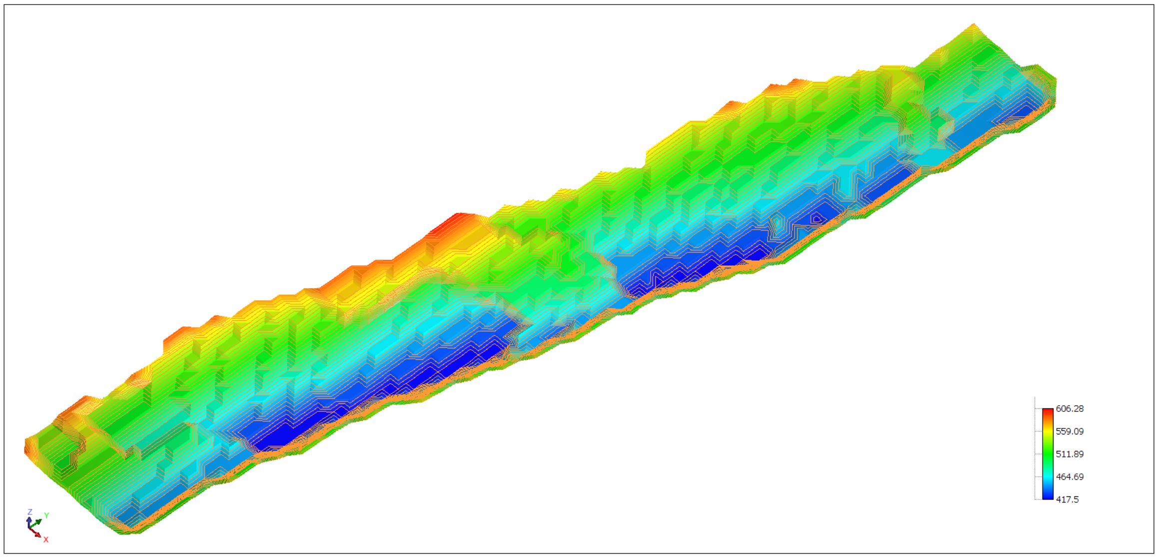

Figure 5.3 Final pit design

The mineral reserves were derived from the geological resource block model and have been identified as being economically extractable and can incorporate mining loses and addition of waste dilution. Mineral reserves are categorized in two classes of materials based on the following cut-off grade parameters:

- Ore: Davis Tube Weight Recovery (DTWR) $\geq$ 21.5% and SiO_2 grade $\leq$ 4.0%.
- Waste: Remaining ore material, overburden and Menihek shale.

Ore can be processed for the concentrate production or can be stockpiled near to the benefit plant for future processing. Also, the waste materials like overburden and Menihek Shale are going to be transported to the waste dump. Mineral reserves estimated from the final pit includes: 2,580 Mt of ore at an average DTWR of 26.1% and SiO_2 of 2.2%, 82 Mt of waste materials.

Table 5.1 LabMag mineral reserves

Category	**Tonnage (Mt)**	**DTWR (%)**	**Total Fe (%)**	**Concentrate**	
				Fe (%)	**SiO_2(%)**
Ore	2,580	26.1	29.7	70	2.2
Waste	82				
Total Material	**2,662**				

The annual expected production of ore is up to 175 Mt and it is to be accomplished in a life project ranging from 15 to 25 years.

It is possible to evaluate the implementation of an IPCC system in this project due to the following conditions:

- The in-pit crusher system can be used on a semi-continuous arrangement because the final wall of the ultimate pit can be reached in early periods.
- The geological/geotechnical conditions of the mine show a layered deposit with low dips. Temporary ramps will be required in order to maintain access to the benches in the advancing wall which is favorable for the location of in-pit crusher and for a belt conveyor system towards the plant.
- Since the pit has a considerable length and depth, it is probable that a system of in-pit crushers can shorten transport distances by trucks and reduce equipment purchase to reduce capital investment.

5.1.4 Process plant

The process plant is designed to treat approximately 175 Mt per year of taconite ore (dry basis) with a feed size up to 1 m for the primary crusher. Operating 358 days per year and 24 hours per day(at a nominal rate of 20,000 t/h), the process plant will recover a nominal of 40 Mt per year of concentrate.

Supplying companies will install lines at different voltages for each project locations. The mine and the processing plant will be powered by a 3.15 kV aerial power lines.

5.1.5 Mine equipment

IPCC system with trucks and shovels is the chosen mining method. Topsoil will be removed using a contractor and will be carried out with a fleet of dozers, small excavators and articulated haul trucks ahead of the mining operation. Additionally, mining dilution will take place at the ore-waste contacts.

The mining operation will include drilling and loading equipment considerably large. Due to the high tonnages expected to be mined and long haul distances, trucks and shovels will be the largest available on the market, and will be very difficult for them to differentiate the ore from the waste in geological contacts. It is assumed that the tonnage increases as a result of mining dilution; it will be balanced with the tonnage reduction as a result of ore losses.

The production fleet is composed by cable shovels with a 100 tonne bucket payload and the haul truck suggested by the company is a mining truck with a payload of 340 tonnes. This truck has been selected because its fleet size is manageable given the significant quantities of material and haul distances expected. The payload of 340 tonnes takes into account the liners on the truck hoppers required to work in this particular project. The trucks will haul the material either to the semi-mobile primary crushers or to the waste dump.

The company suggests the following parameters to estimate the operating hours per year:

- Mechanical availability: 85%
- Utilization: 90%
- Shift schedule: Two, 12 hour shifts per day, seven days per week
- Operational delays: 80 min/shift
- Job efficiency: 90%

Table 5.2 summarizes the yearly working hours for mine equipment.

Table 5.2 Yearly operating hours: adapted from Met-Chem Canada (2014)

Description	**Hours**	**Details**
Total Hours	8,760	7 days per week, 24 hours per day, 52 weeks per year
Scheduled Hours	8,592	Hours available accounting for weather delays
Down Mechanically	1,289	15 % of total hours
Available	7,303	Total hours minus hours down mechanically
Standby	730	10% of available hours (represents 90 % utilization)
Operating	6,573	Available hours minus standby hours
Operating Delays	730	80 min/shift
Net Operating Hours	5,483	Operating hours minus operating delays
Working Hours	**5,258**	**90% of net operating hours (reflects job efficiency)**

5.2 Trucks and shovels case

Consider the problem of locating a crusher in the pit rim to reduce haul trucks transportation costs and increase the cash flow. Traditionally, the proposed location is based on pit geometry and the distance from the processing plant to the exit ramp.

The goal of the presented case is to choose an optimal crusher location for trucks and shovels system, which will be denoted as TS case in the rest of the document. Eight (8) crusher candidate locations are selected with their respective belt conveyor design, operational costs and capital expenditure. The criteria to select each location is based on a constant distance of 1 km from the plant through the final pit boundary. Figure 5.4 shows the proposed locations for the stationary crusher.

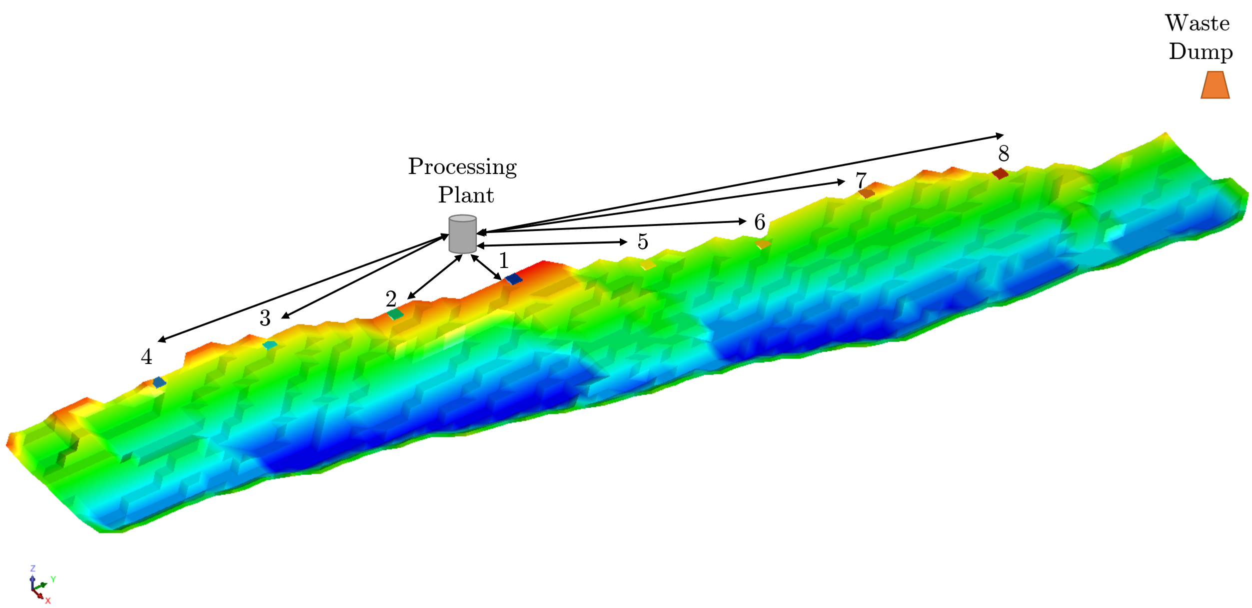

Figure 5.4 Crusher candidate locations - truck and shovel case

In this system, trucks haul both ore and waste out of the pit to the crusher at the pit rim or at the waste dump. Then, the ore is dumped to the hopper of a stationary crusher locate at the pit rim where the material is transferred to a conveyor connected to the processing plant or to the stockpile, both located at the same place. Both the ore and the waste haul trucks are of the same type.

It is assumed that all conveyors are steel cord belt type. The belt width was assumed to be of 2.4 m for all conveyors and the maximum belt speed of 6.0 m/s was calculated in accordance with the Standards of the Equipment Manufacturers Association (CEMA). Besides, the design capacity for the annual production of 175 Mt is 22,000 t/h. The cross sectional area of load vary between 45-85% under CEMA standards at 18° of surcharge angle.

5.3 IPCC 8 systems case

The IPCC 8 systems case will be denoted as IPCC 8 case in the rest of the document. It consists of eight(8) potential systems where each system is composed by a set of conveyor segments. For example, system 1 in Table 5.4(see also Figure 5.5), where the trucks haul ore to the in-pit crusher and the waste is also hauled by trucks to the waste dump. Both the ore and the waste haul trucks are of the same type as those mentioned in Section 5.2.

As shown in Figure 5.5, twenty-four(24) potential locations have been identified with the arrangement of the belt conveyor in-pit across the benches. The main conveyor transports ore to the surface where the material is transferred to a conveyor connected either to the processing plant or the stockpile, both located at the same place. In this case, eight(8) possible crusher locations are available and near to the plant and can be used without installation costs. Subsequently, belt conveyor segments enable any of the twenty-four(24) crusher possible locations to be operational.

It is preferred to minimize crusher relocation and belt conveyor installation for operating benches. Mine equipment performance is better when working on either parallel or radial bench conveyor setups (Li, 2014). Moreover, the parallel setup is shown in column three of Table 5.4. However, flexibility is included by allowing this case to run on a parallel belt conveyor system, totalizing as follows:

- Belt conveyor systems: 2×8
- Crusher candidate locations: 2×24

The system consists of two (2) primary semi-mobile crushers each rated at 11,000 t/h and will process 22,000 t/h of ROM using their respective belt conveyor segments. Also, the crushed ore will be transported on parallel belt conveyors systems.

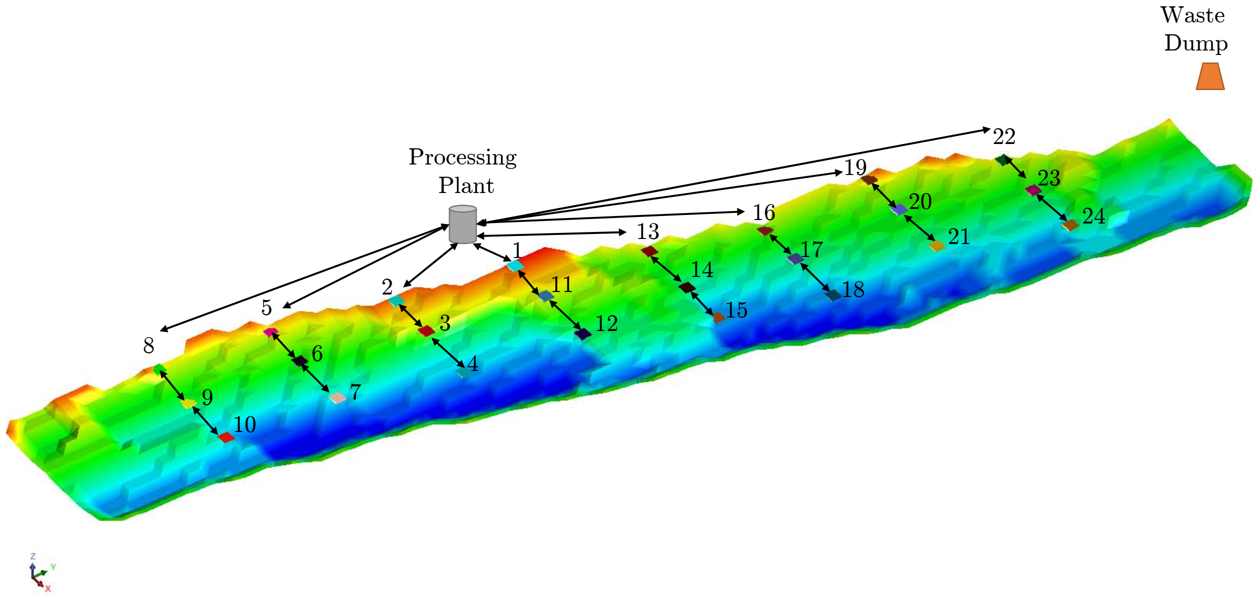

Figure 5.5 Crusher candidate locations - IPCC 8 systems case

5.4 IPCC 5 systems case

The IPCC 5 systems case, denoted as IPCC 5 case in the rest of the document. It consists of five(5) potential systems where each system is composed by a set of conveyor segments. For example, system 1 in Table 5.5(see also Figure 5.7). For this case the deposit was divided into four (4) zones and the criterion to delimit these zones was that each zone must contain the same total tonnage (around 665 Mt). Figure 5.6 shows this zones:

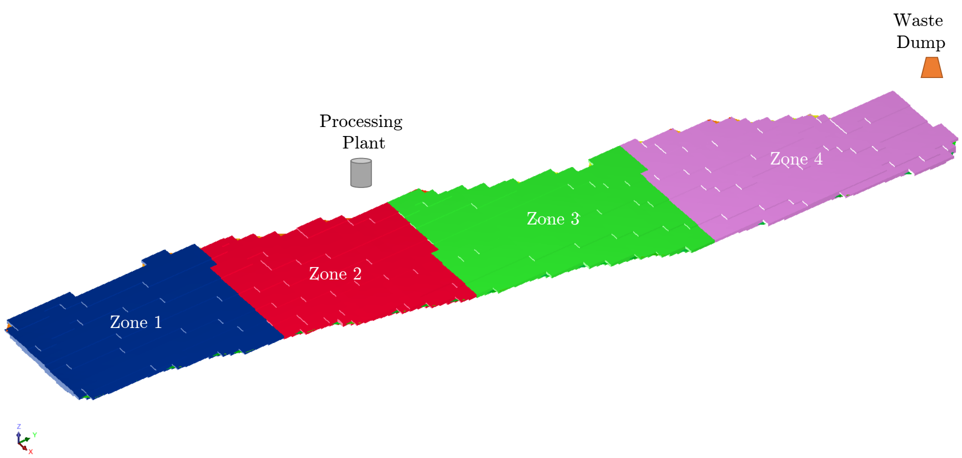

Figure 5.6 Pit zones

The proposed locations for crushers and belt conveyors were placed at the border between each zone to be used as a starting point of the belt conveyor system from the processing plant to the pit bottom. As presented in the previous IPCC 8 case, this system works with conveyors operating in a parallel setup. Although, the crusher location next to the plant can be used without installation costs, it can reproduce a situation where a crusher is installed from the beginning of the project. Figure 5.7 shows the described case.

- Belt conveyor systems: 2×5
- Crusher candidate locations: 2×14

It is assumed that all conveyors are steel cord belt type. The belt width was assumed to be of 1.5 m for all conveyors and the maximum belt speed of 6.0 m/s was calculated in accordance with the Standards of the Equipment Manufacturers Association (CEMA). Besides, the design capacity for the annual production of 175 Mt is 22,000 t/h. The cross sectional area of load vary between 45-85% under CEMA standards at 18° of surcharge angle.

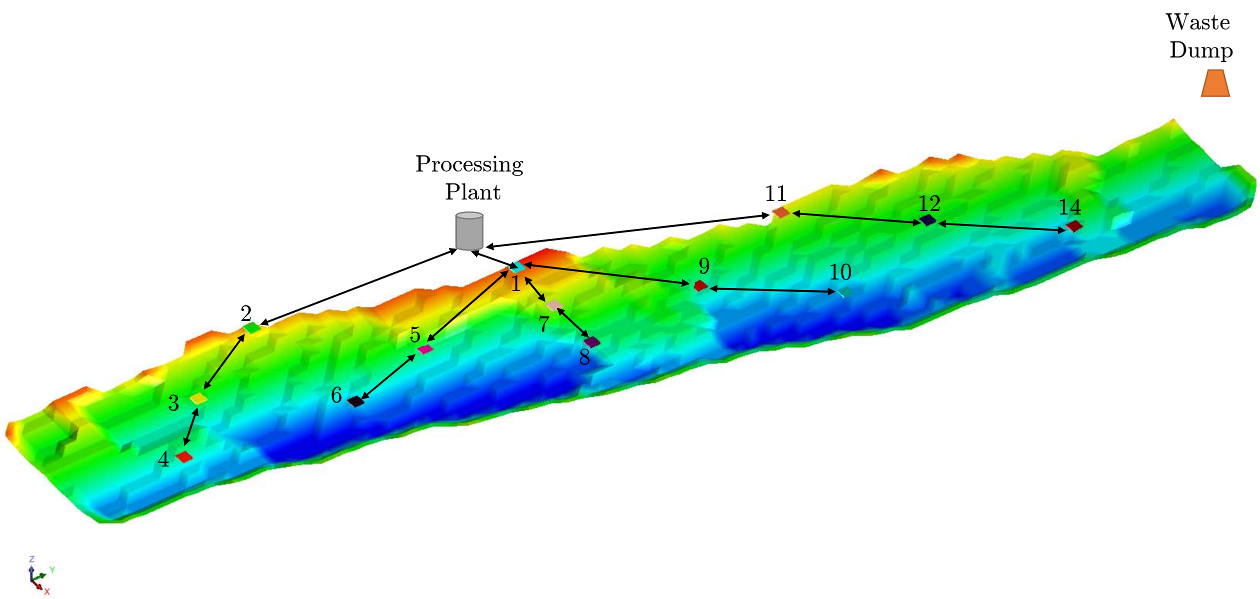

Figure 5.7 Crusher candidate locations - IPCC 5 systems case

In the first scheduling period, both IPCC cases are allowed to install the crushers next to the plant without any installation cost. As the pit deepens, the crushers can be relocated independently to a candidate location if it is available and follows excavation and IPCC constraints.

5.5 Input data

The information provided by the company for the realization of this thesis is presented as follows:

`Block model`

id	number of blocks: 8223 total
d_x	block dimension x axis: 100 m
d_y	block dimension y axis: 100 m
d_z	block dimension z axis: 15 m
$\%UPIT$	% block contained on ultimate pit
$density$	density block
t_{per}	% block under topography
t_{ob}	% block containing overburden
t_{ms}	% block containing Melik shale
$density_{ob}$	% density overburden
$density_{ms}$	% density Melik shale
Fe	Fe grade
SiO_2	SiO_2 grade
$DTWR$	Davis Tube Weight Recovery value
$EFWR$	Effective Weight Recovery value

$$T^O = \left(\frac{d_x \times d_y \times d_z \times \%UPIT \times density}{100}\right) \times \left(\frac{t_{per} - t_{ob} - t_{ms}}{t_{per}}\right) \tag{5.1}$$

$$T^W = \left(\frac{d_x \times d_y \times d_z \times \%UPIT}{100}\right) \times \left(\frac{(t_{ob} \times density_{ob}) + (t_{ms} \times density_{ms})}{t_{per}}\right) \tag{5.2}$$

$$\text{CONCENTRATED tonnes or } T^C = \frac{T^O \times EFWR}{100} \tag{5.3}$$

Mining parameters

g_{Fe}	Fe cut-off grade: 21.5%
$\overline{g}$	average input Fe grade to the mill: 26.1%
I	interest rate: 8%
P	commodity price: 113.31 $/t
R^p	processing recovery value: $EFWR$
c^{ODB}	drilling and blasting cost: 2.49 $/t
c^{OL}	loading cost: 1.51 $/t
c^{ORH}	rehandling of stockpiled ore cost: 1.0 $/t
c^{OP}	processing cost: 38 $/t
c^{WDB}	drilling and blasting cost: 2.49 $/
c^{WL}	waste loading cost: 1.51 $/t

Trucks parameters

PT	purchase price of a truck $6,000,000
IT	truck idle cost: 744,600$/period
OP	Available operating hours: 5,258h/period

Belt conveyor parameters

Belt conveyor system availability is 90%(Media et al., 2015). Because of the low and independent number of individual conveyor segments, it is not possible in practice to assume that failures will occur simultaneously.

Table 5.3 Belt conveyor parameters for the TS case

Conveyor	**Vertical lift(m)**	**Belt length(m)**	**Power (kW)**	**$ kW/h**	**$ kW/t**	**Installation Cost**
System 1	45	403	2,294	- $	- $	- $
System 2	15	1,020	1,780	$ 92.553	$ 0.0075	$ 10,869,359
System 3	45	2,062	4,118	$ 214.136	$ 0.0174	$ 12,720,139
System 4	30	3,060	4,669	$ 242.771	$ 0.0197	$ 14,491,686
System 5	60	1,079	3,631	$ 188.803	$ 0.0153	$ 10,973,765
System 6	75	2,026	5,265	$ 273.772	$ 0.0223	$ 12,656,543
System 7	30	2,983	4,527	$ 235.429	$ 0.0191	$ 14,356,498
System 8	45	4,052	6,580	$ 342.170	$ 0.0278	$ 16,254,950

Table 5.4 Belt conveyor parameters for the IPCC 8 case

System (parallel)	Segment	Parallel segment	Vertical lift(m)	Belt length(m)	Power (kW)	$ kW/h	Segment ($ kW/t)	System ($ kW/t)	Installation Cost
1(9)	plant - 1	plant - 25	45	403	2,294	$ -	$ -	$ -	$ -
	1 -11	25 -35	45	405	2,294	$ 119.30	$ 0.0097	$ 0.0097	$ 9,772,897
	11 - 12	35 - 36	60	504	3,000	$ 156.02	$ 0.0127	$ 0.0224	$ 9,952,383
2(10)	plant - 2	plant - 26	45	2,062	4,118	$ 214.14	$ 0.0174	$ 0.0174	$ 12,720,139
	2 - 3	26 - 27	45	403	2,292	$ 119.19	$ 0.0097	$ 0.0271	$ 9,772,897
	3 - 4	27 - 28	45	502	2,403	$ 124.97	$ 0.0102	$ 0.0373	$ 9,949,601
3(11)	plant - 5	plant - 29	30	3,060	4,669	$ 242.77	$ 0.0197	$ 0.0197	$ 14,491,686
	5 - 6	29 - 30	75	407	3,490	$ 181.48	$ 0.0148	$ 0.0345	$ 9,780,795
	6 - 7	30 - 31	30	501	1,805	$ 93.87	$ 0.0076	$ 0.0421	$ 9,947,609
4(12)	plant - 8	plant - 32	15	1,020	1,780	$ 92.55	$ 0.0075	$ 0.0075	$ 10,869,359
	8 - 9	32 - 33	45	403	2,294	$ 119.30	$ 0.0097	$ 0.0172	$ 9,772,897
	9 - 10	33 - 34	90	508	4,199	$ 218.35	$ 0.0178	$ 0.0350	$ 9,960,283

Table 5.4 Belt conveyor parameters for the IPCC 8 case (*Continuation*)

System (parallel)	**Segment**	**Parallel segment**	**Vertical lift(m)**	**Belt length(m)**	**Power (kW)**	**$ kW/h**	**Segment ($ kW/t)**	**System ($ kW/t)**	**Installation Cost**
5(13)	plant- 13	plant- 37	60	1,079	3,631	$ 188.80	$ 0.0153	$ 0.0153	$ 10,973,765
	13 - 14	37 - 38	45	502	2,403	$ 124.97	$ 0.0102	$ 0.0255	$ 9,949,601
	14 - 15	38 - 39	45	403	2,294	$ 119.30	$ 0.0097	$ 0.0352	$ 9,772,897
6(14)	plant - 16	plant - 40	75	2,026	5,265	$ 273.77	$ 0.0223	$ 0.0223	$ 12,656,543
	16 - 17	40 - 41	30	401	1,697	$ 88.25	$ 0.0072	$ 0.0294	$ 9,770,411
	17 - 18	41 - 42	45	502	2,403	$ 124.97	$ 0.0102	$ 0.0396	$ 9,949,601
7(15)	plant - 19	plant - 43	30	2,983	4,527	$ 235.43	$ 0.0191	$ 0.0191	$ 14,356,498
	19- 20	43- 44	45	403	2,294	$ 119.30	$ 0.0097	$ 0.0288	$ 9,772,897
	20 - 21	44 - 45	45	502	2,403	$ 124.97	$ 0.0102	$ 0.0390	$ 9,949,601
8(16)	plant - 22	plant - 46	45	4,052	6,580	$ 342.17	$ 0.0278	$ 0.0278	$ 16,254,950
	22 - 23	46 - 47	45	403	2,294	$ 119.30	$ 0.0097	$ 0.0375	$ 9,772,897
	23 - 24	47 - 48	30	501	1,807	$ 93.94	$ 0.0076	$ 0.0452	$ 9,947,609

Table 5.5 Belt conveyor parameters for the IPCC 5 case

System (parallel)	Segment	Parallel segment	Vertical lift(m)	Belt length(m)	Power (kW)	$ kW/h	Segment ($ kW/t)	System ($ kW/t)	Installation Cost
1(6)	plant - 2	plant - 15	15	2,184	3,056	$ 158.90	$ 0.0129	$ 0.0129	$ 12,936,879
	2 - 3	14 - 15	75	898	4,039	$ 210.02	$ 0.0171	$ 0.0300	$ 10,652,074
	3 - 4	16 - 17	30	708	2,386	$ 124.09	$ 0.0101	$ 0.0401	$ 10,314,955
2(7)	plant - 1	plant - 14	45	403	2,294	$ -	$ -	$ -	$ -
	1 - 5	14 - 18	105	1,145	5,971	$ 310.52	$ 0.0252	$ 0.0252	$ 11,091,506
	5 - 6	18 - 19	30	825	2,386	$ 124.09	$ 0.0101	$ 0.0353	$ 10,523,495
3(8)	plant - 1	plant - 14	45	403	2,294	$ -	$ -	$ -	$ -
	1 - 7	14 - 20	60	504	3,062	$ 159.24	$ 0.0129	$ 0.0129	$ 9,952,383
	7 - 8	20 - 21	45	502	2,450	$ 127.42	$ 0.0104	$ 0.0233	$ 9,949,601
4(9)	plant - 1	plant - 14	45	403	2,294	$ -	$ -	$ -	$ -
	1 - 9	14 - 22	120	1,426	7,361	$ 382.77	$ 0.0311	$ 0.0311	$ 11,591,126
	9 - 10	22 - 23	30	1,141	3,044	$ 158.29	$ 0.0129	$ 0.0440	$ 11,083,639
5(10)	plant - 11	plant - 24	45	2,184	4,933	$ 256.52	$ 0.0209	$ 0.0209	$ 12,937,610
	11 - 12	24 - 25	45	1,141	3,656	$ 190.11	$ 0.0155	$ 0.0363	$ 11,084,514
	12 - 13	25 - 26	30	1,141	3,044	$ 158.29	$ 0.0129	$ 0.0492	$ 11,083,639

5.6 Results

5.6.1 Overview

The cases were programmed in A Mathematical Programming Language (AMPL) and solved using CPLEX 12.7 in a computer with eight(8) i7 processors at 3.2 Ghz and 24 GB memory RAM. Table 5.6 contains the computational tests results for each case. The Tolerance Gap measures optimality between the best integer objective value and the objective value of the best node of the evaluated branches.

Table 5.6 Summary of results

Case	NPV	Locations used	Tolerance Gap	Run time
TS	$20,402,054,864	1	2.00%	6 h
IPCC 8	$21,128,467,106	13	3.44%	24 h
IPCC 5	$21,263,339,903	15	2.29%	12 h

The overall results indicate a representative increase in the NPV between the different cases where IPCC was implemented. The TS case using a single stationary crusher as ore destination obtained the worst NPV compared to the cases where IPCC was used. As a result the cases where semi-mobile crushers on a parallel belt conveyor setup were implemented obtain a higher NPV compared with the TS case: the IPCC 8 case is +4.5% while the IPCC 5 case is +5.0%.

Figure 5.8 presents ore production profiles between the different cases. During the mine life, the annual production remained approximately even in all cases, but in the last period the TS case had a decrease in production which is the result of extracting more material in previous periods compared to the IPCC cases in the same periods. The stockpile use for each case is presented in Figure 5.9. It is remarkable the minimum use of this option because in all cases the total ore annual material that went to the stockpile is below 600,000 tonnes, which represents less than 0.3% of ore annual production of 175 Mt.

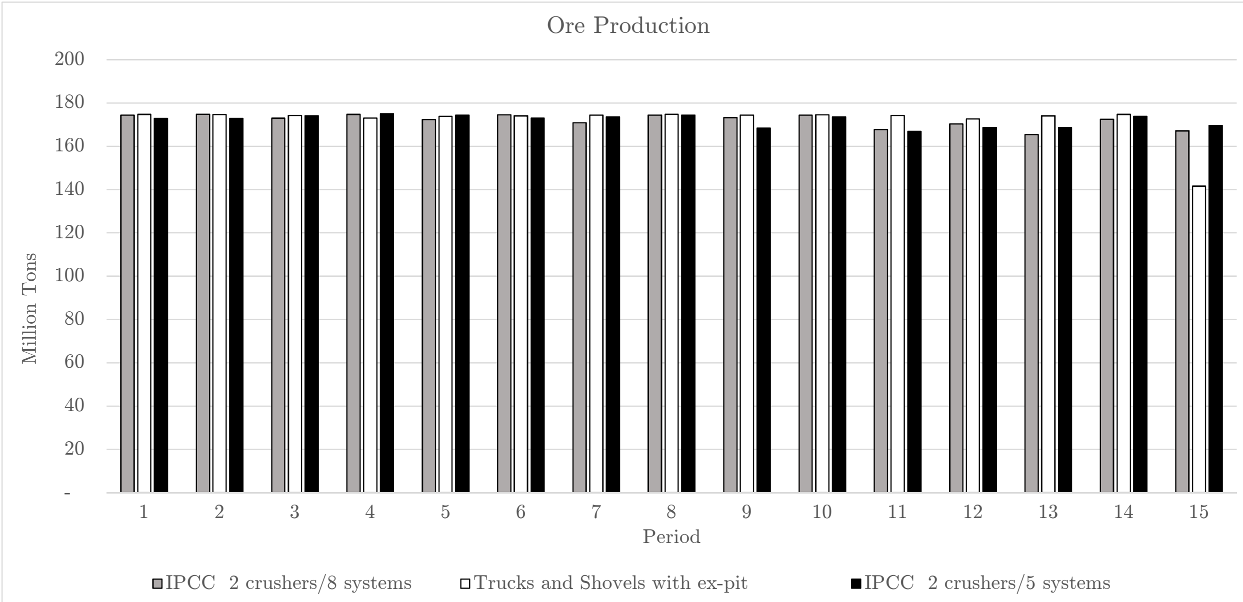

Figure 5.8 Ore production per period

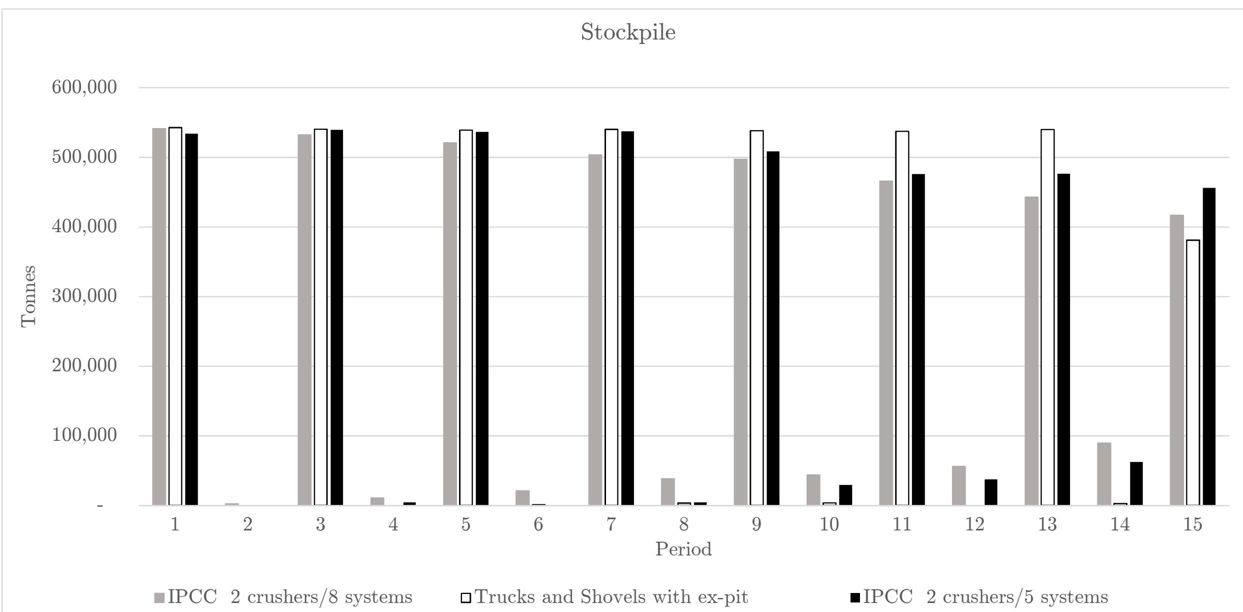

Figure 5.9 Tonnes in stockpile in each year

The waste extraction profiles between cases are presented in Figure 5.10. During periods 1 to 5, for all cases, the large quantity was extracted. Waste extraction in the IPCC 8 case is optimum in handling stripping ratio due to a descending profile compared to the other cases.

Since waste material represents around 3% of the total material to be excavated, it does not have a great impact on the solution.

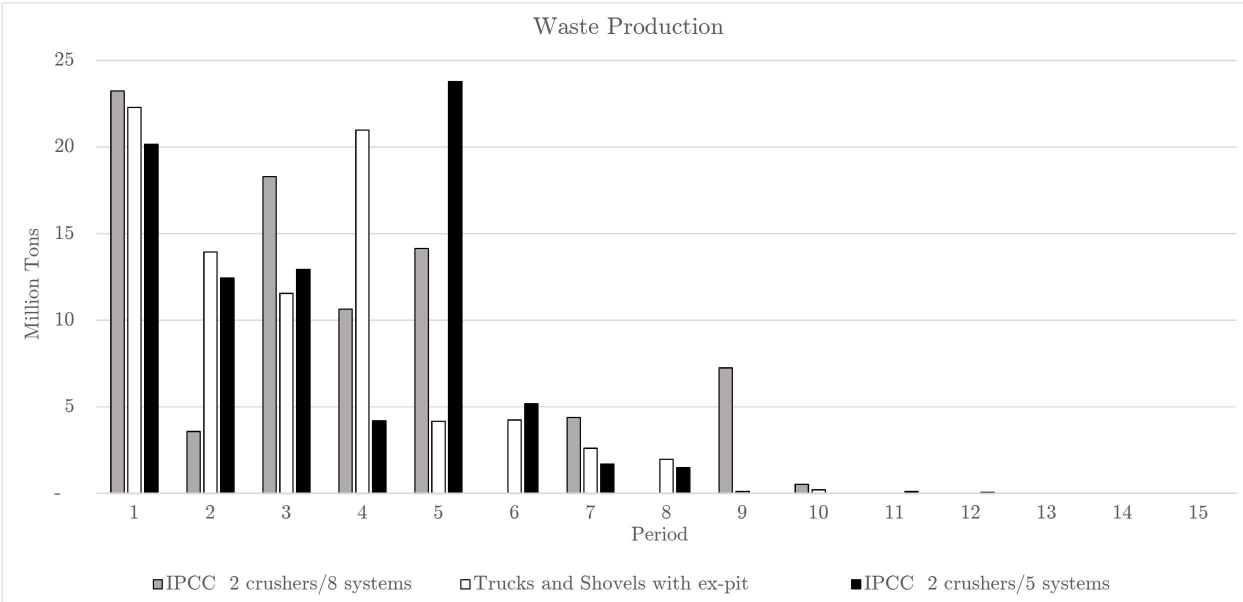

Figure 5.10 Waste production per period

5.6.2 Mine transportation

In the TS case according to Figure 5.11, a crusher is located at 1 km far from the plant, contrary to the criterion which is often locate the crusher close to the plant. This result indicates that it is better to invest in a 1 km belt conveyor and place it in a non-symmetrical position in respect to the pit geometry than to locate it next to the plant without investing in a belt conveyor system.

However, the pit has a symmetrical geometry in respect to the plant location and the material spatial distribution as is shown in Figure 5.6. The excavation sequence and haulage distances allocated the optimal position in the same direction where the pit expands. The position selected can be used as a starting point to place an exit ramp because all the haulage cycles calculated have that point as the final destination.

For IPCC cases, the solution supports the use of different positions for the crushers. For this effect, a visual review of these results are presented below:

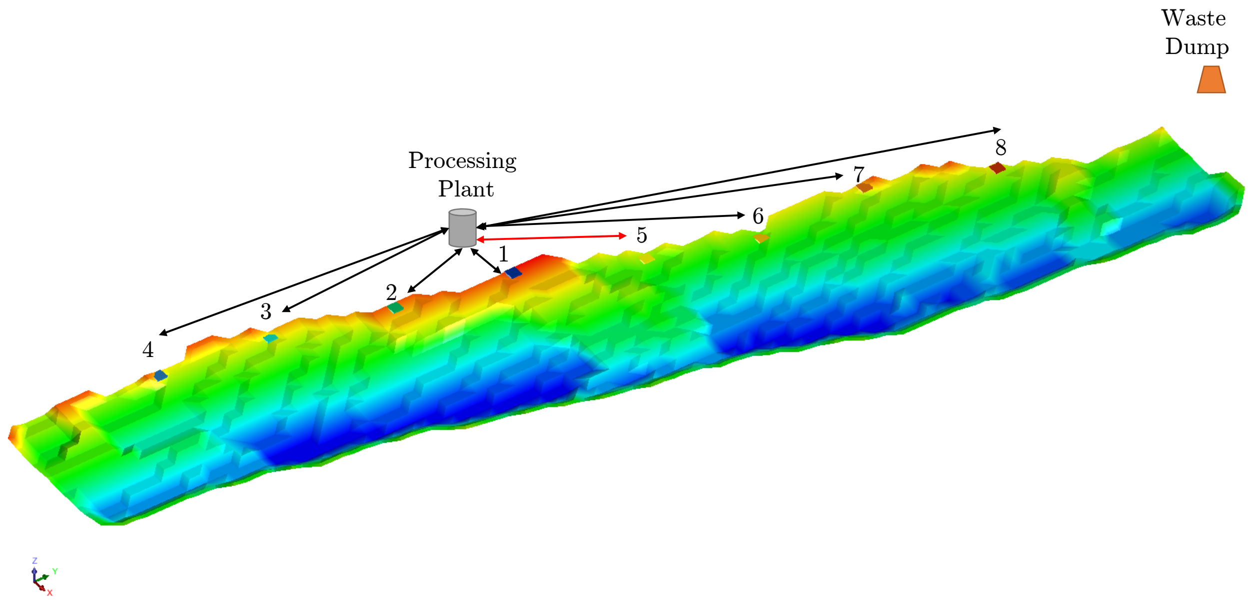

Figure 5.11 Crusher location - TS case

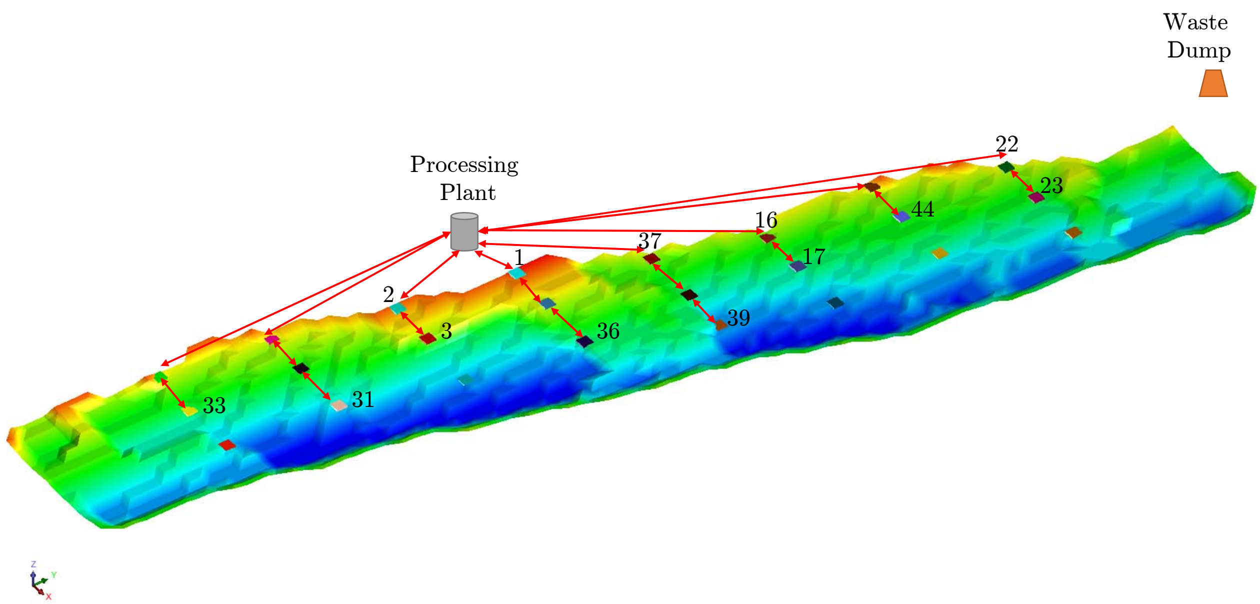

Figure 5.12 Systems installed and crusher locations used - IPCC 8 case

For the IPCC 8 case shown in Figure 5.12, the solution using partially all the proposed systems is clearly displayed. Table 5.7 details until which segments are necessary to be

Table 5.7 Conveyor segments and crushers installation - IPCC 8 case

Period	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**
Segment installed	1 44	16				36		31	33	22 37	39		2	23 3	17
Crushers location	1 44	16 44	1 16	1 44	16 44	36 44	1 36	31 36	1 33	22 37	37 39	31 37	2 31	23 3	17 37

installed in each system to become operational. For example, in period 1, positions 1 and 44 are used as crusher locations. To use position 1, it is needed to install the segment plant - 1. And to use the position 44, it is needed to install the segments plant - 43 and 43 - 44. In this way, the model allows a crusher to work in location 44. The use of 4 out of 8 positions at the pit bottom indicates that IPCC system is partially implemented in the pit.

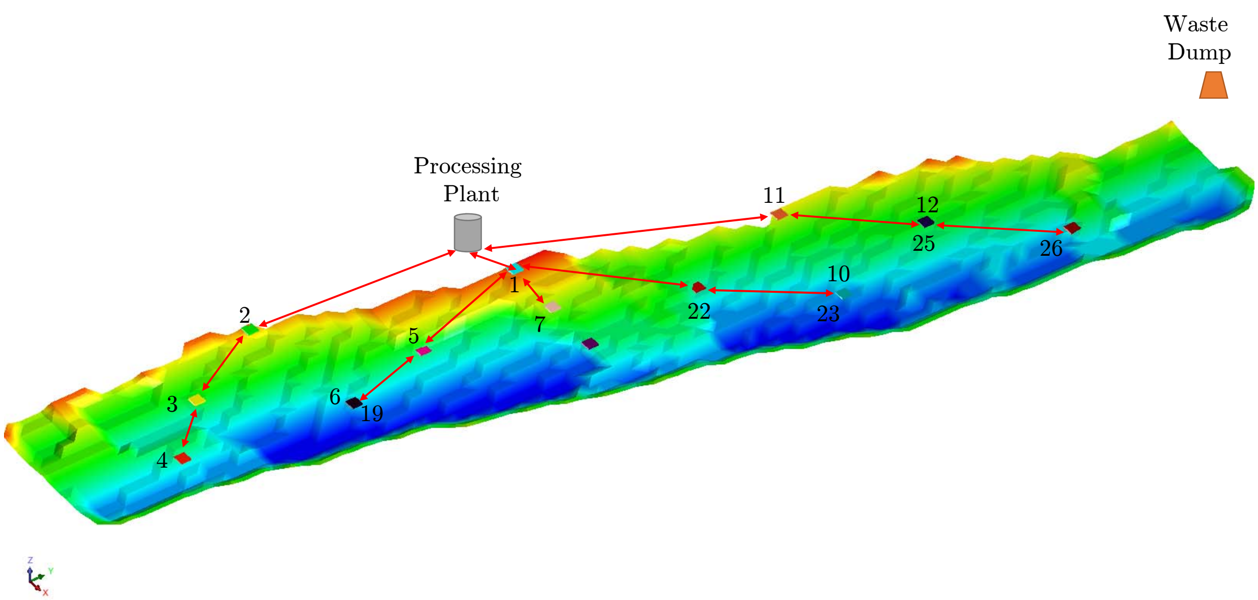

Figure 5.13 Systems installed and crusher locations used - IPCC 5 case

The IPCC 5 case shows similar results to the previous case with respect to the belt conveyors systems installation. Only one position at the bottom was not used and it had the lowest depth in the pit, and it is possible because either the previous or nearby locations can receive the material from this bottom area. The belt conveyor segments installation order and use of each position over time have a similar v to the IPCC 8 case. In the first periods, the plant right side positions were used, then the left side and finally the ones located at the pit bottom.

Table 5.8 Conveyor segments and crushers installation - IPCC 5 case

Period	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**
Segment installed	1 11		12 25		3	7	2		23	10	5 19		4 6	22 26	
Crushers location	1 11	1 11	12 25	11 12	3 25	7 25	2 3	2 3	3 23	3 10	5 19	10 12	4 6	22 26	6 23

An analysis of truck haulage distances for ore is presented in Figure 5.14. TS case displays an expected behavior increasing haulage distances as the pit deepens through the mine plan. For IPCC cases, haulage distances do not increase with time because the in-pit crushers relocation shortens these distances. In the IPCC 8 case, the average distance increases in two periods compared with the IPCC 5 case. In this case, the haulage distance tends to decrease with time indicating a lower use of trucks due to the crushers relocation shown in Table 5.8.

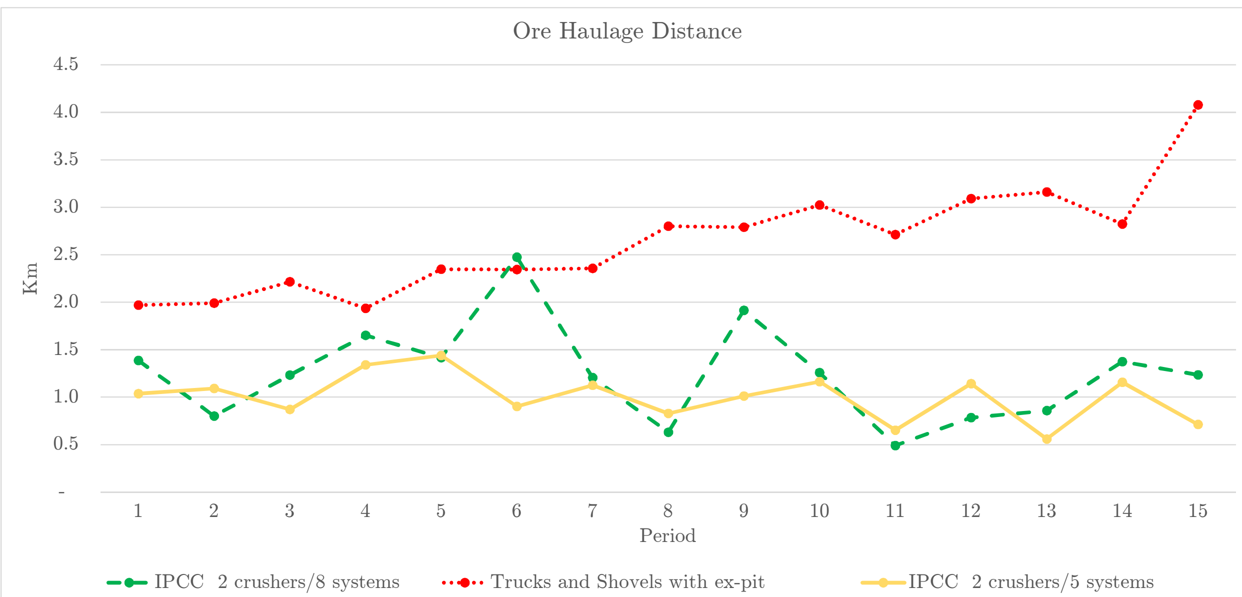

Figure 5.14 Ore haulage distance

The number of trucks required profile is similar to haulage distance in each case. In the TS case, the number of trucks required in the last periods are acceptable under financial terms because the purchases of new trucks were made three periods before the end of the project. For the IPCC 8 case, noticeable variations are observed in two periods, including these extra trucks required for only one period, the total number of trucks to purchases is less than TS case. Finally, the IPCC 5 case shows the best truck requirement profile because it decreases the number needed in each period in accordance to the haulage distances discussed before.

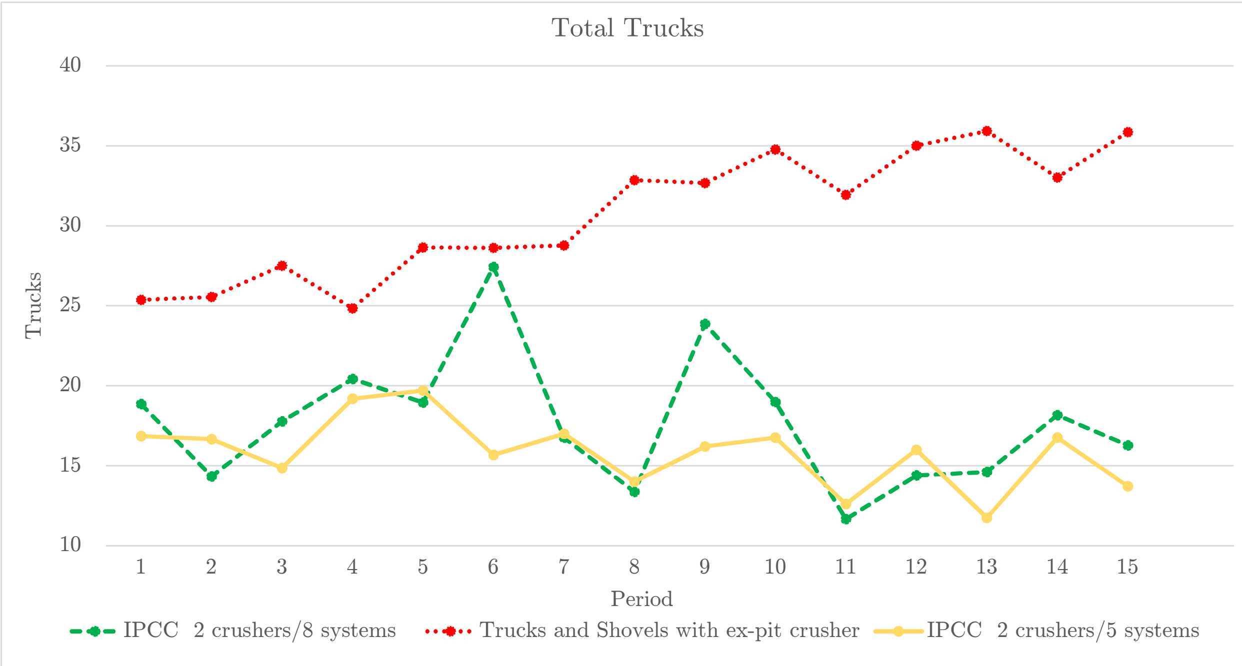

Figure 5.15 Total trucks required

Total transportation costs between cases are presented in Figure 5.16. In the TS case annual costs increase slightly over the duration of the project in accordance with the haulage distances increase. For IPCC cases, the annual costs have a tendency to decrease due to crusher repositioning shortening haulage distances; consequently IPCC 5 systems is the best case with lower and decreasing annual costs.

Costs per tonne transported is presented in Figure 5.17 and corroborated the total transportation cost analysis. The average cost per tonne transported for each case is as follows:

- Trucks and Shovels: 1.65 $/t
- IPCC 8 systems: 1.02 $/t
- IPCC 5 systems: 0.93 $/t

Assuming that the TS case as the base case, then both IPCC systems have a considerable cost reduction per tonne transported: IPCC 8 case reduction is -30% and IPCC 5 case reduction is -43%.

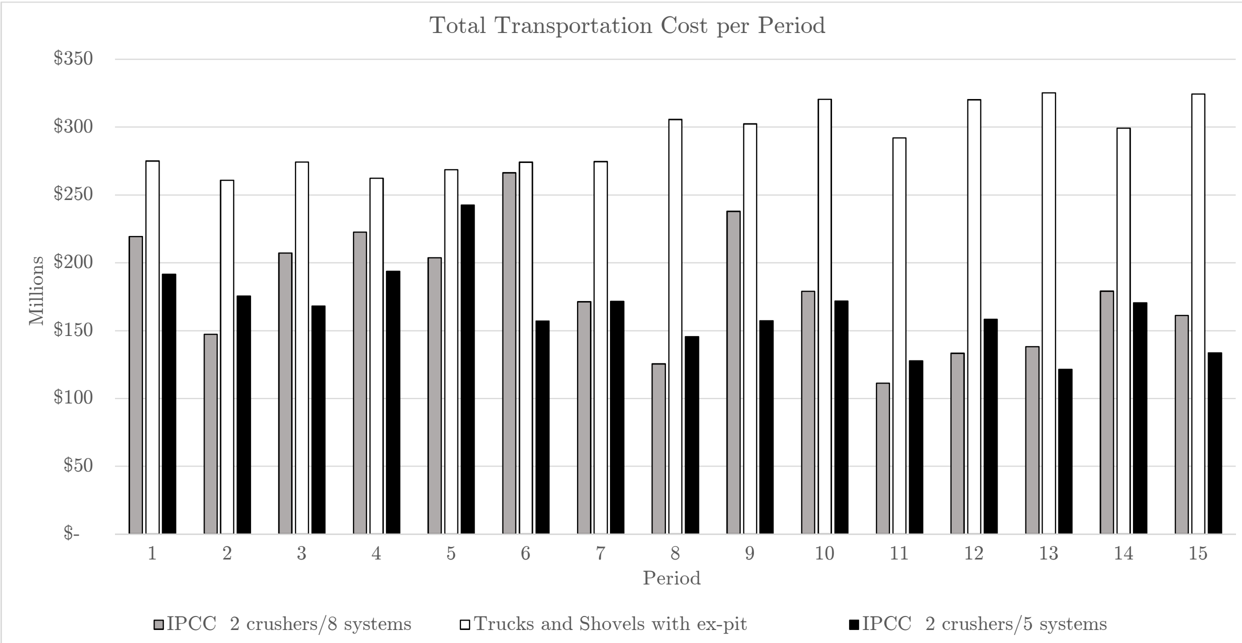

Figure 5.16 Transportation cost per period

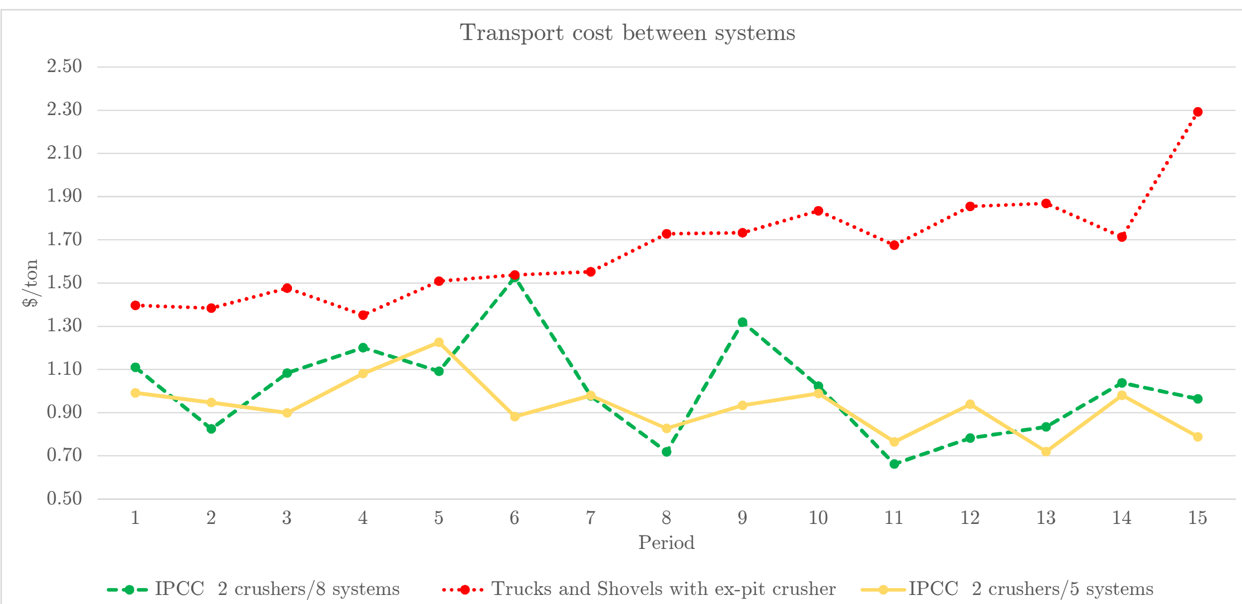

Figure 5.17 Transportation cost per ton

Cumulative annual operating costs and capital investments required for each case are shown in Figure 5.18 based on the results presented before. Capital costs include the required the trucks and belt conveyor segments to transport the extracted material. Operational costs for the TS case are approximately +38% higher compared to IPCC systems which means that IPCC implementation can reduce significantly the transportation costs. Table 5.9 displays the cost reduction between the traditional trucks and shovels system and the IPCC systems.

Table 5.9 Cumulative transportation cost and capital investment

Case	Truck and shovel	IPCC 8 systems	IPCC 5 systems
Transportation cost	$4,379,910,203	$2,702,964,776	$2,486,920,030
Capital investment	$189,449,847	$193,397,451	$173,041,204

Although, the capital investment expenses between the cases are similar; the project will not have a significant reduction in this investment due to the required number of trucks which demand the purchase of belt conveyor segments in the IPCC cases.

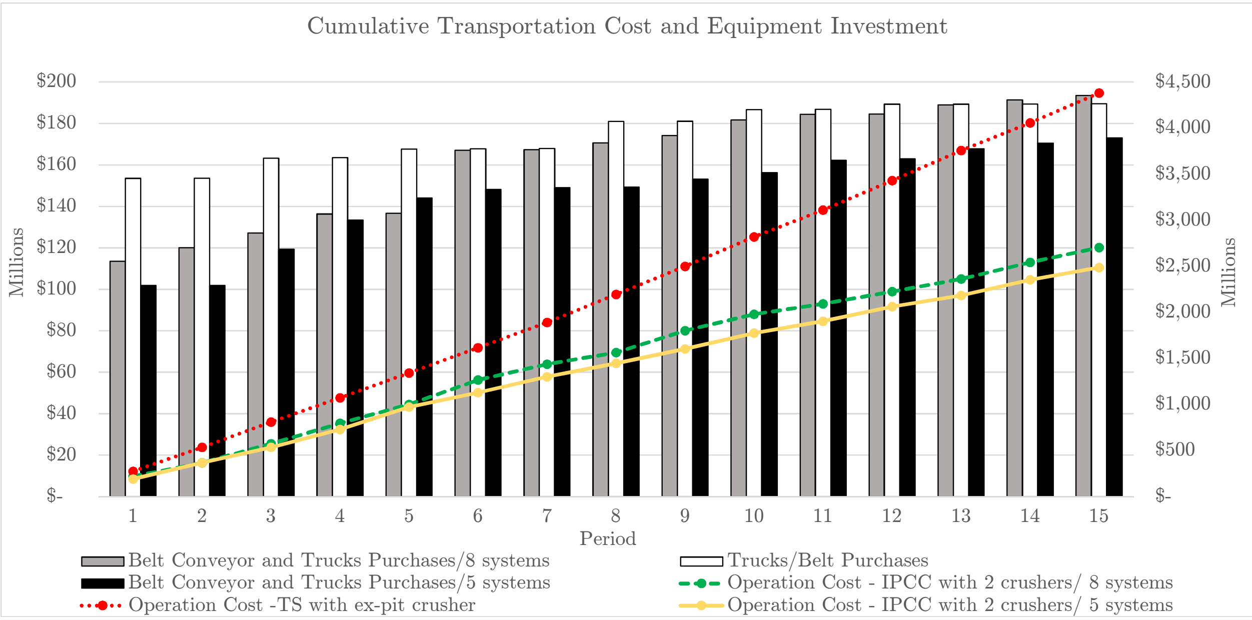

Figure 5.18 Cumulative capital and operational cost

As a conclusion, the IPCC 5 systems case is a suitable transportation system option for the project, considering the technical and economical results reviewed in this chapter.

5.6.3 Discount Rate Sensitivity Analysis

The discount rate, which is used to discount the revenue of the operation over time, was considered as an input parameter to the model presented. The study case IPCC 5 systems was solved using 8% as discount rate. However, the NPV is sensitive to the discount rate. Increasing the interest rate reduces the NPV comparing to a lower discount rate, because the revenue gained is discounted in later periods under a higher interest in equation 4.5. Table 5.10 provides the NPV values for a range of discount rates between six to ten percent.

Table 5.10 Discount Rate Sensitivity Analysis

Discount Rate	**NPV**	**Difference (%)**
6%	24,075,004,491	13.2%
7%	22,601,663,826	6.3%
8%	21,263,339,903	-
9%	20,045,008,872	-5.7%
10%	18,933,539,043	-11.0%

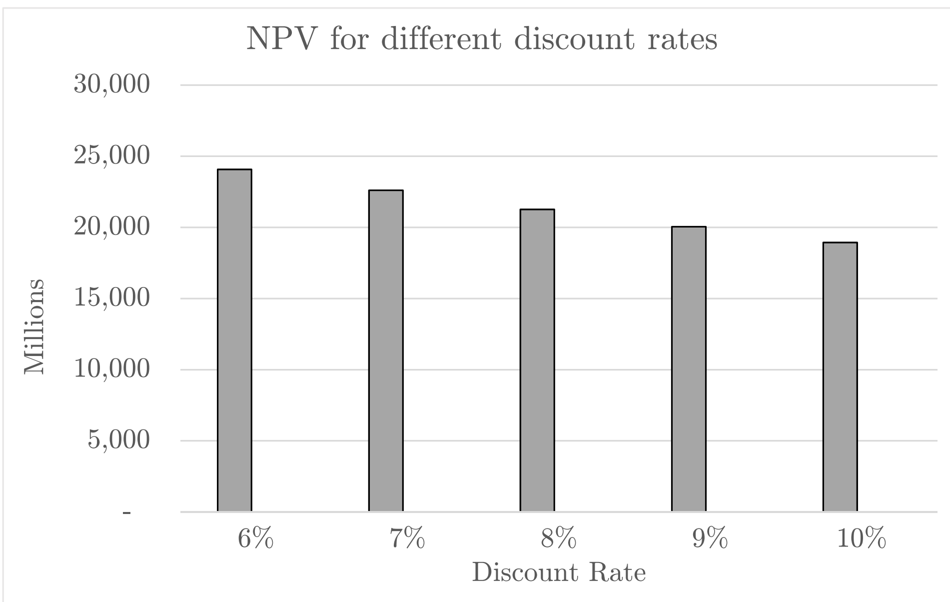

Figure 5.19 Discount Rate Sensitivity Analysis

The recalculated NPVs are displayed in Figure 5.19. The NPV under a 6% discount rate is $24,075,004M, and decrease gradually to $18,933,539M for a 10% discount rate. This shows the value of choosing a realistic rate, because the model presented compare different transportation systems implementation in a prefeasibility study stage.

However, since in this thesis goal was develop a mathematical tool to be used in decision making for a transportation system, the discount rate value was given by the sponsor.

5.6.4 Risk profile

The IPCC 5 case mine production scheduling is based on a single estimated model. In order to evaluate the geological uncertainty involved, this schedule was evaluated on ten (10) simulated orebody models provided by the company. Using the given data, the risk profiles were generated.

For the discounted cash flow, the risk is shown in Figure 5.20. The difference between the expected cash flow is negligible in practice (less than 1.0%). Therefore, it is possible to conclude that the generated schedule using the deterministic model is robust when it is compared to the different orebody simulations.

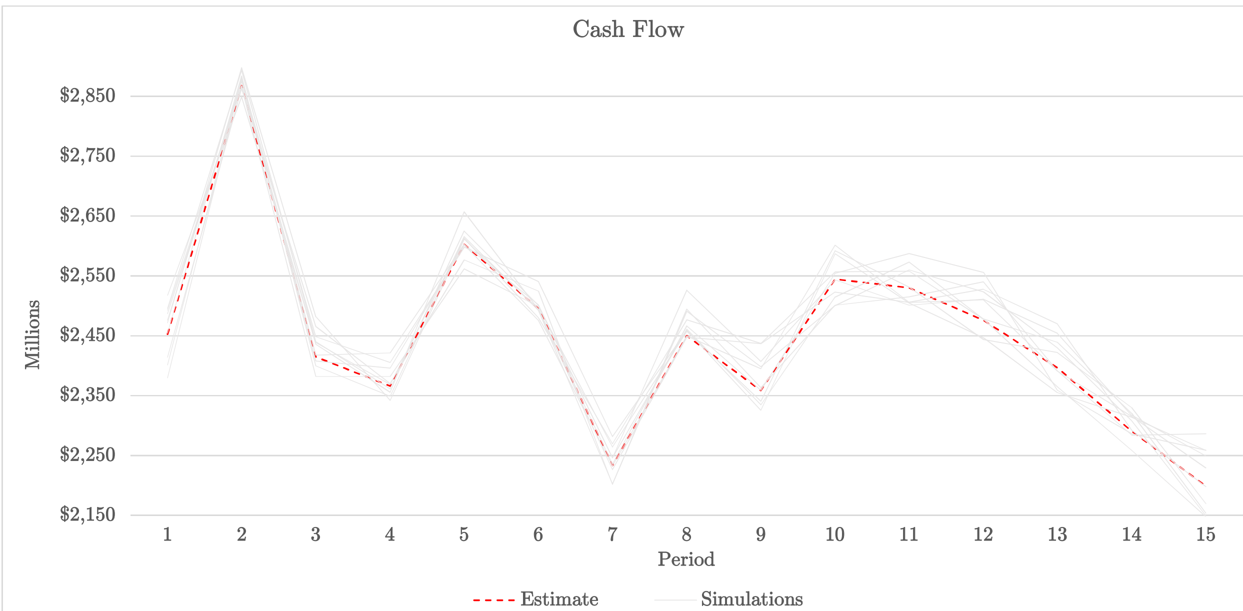

Figure 5.20 Cash flow risk profile

In terms of constraints, the risk profiles are all acceptable. Figure 5.21, 5.23 and 5.24 represent respectively the ore tonnage, the average SiO_2 grade, and the average Fe grade for all the simulations and their averages. The average ore production is close to 175 Mt target with a expected deviation of 3% in periods 5 and 9. And, the SiO_2 and Fe grade are within the range of tolerance. Figure 5.22 shows that there is no ore being misclassified as waste material.

All profiles illustrate that the solution provided by the deterministic approach are possible to be achieved in practice when evaluated over probable scenarios.

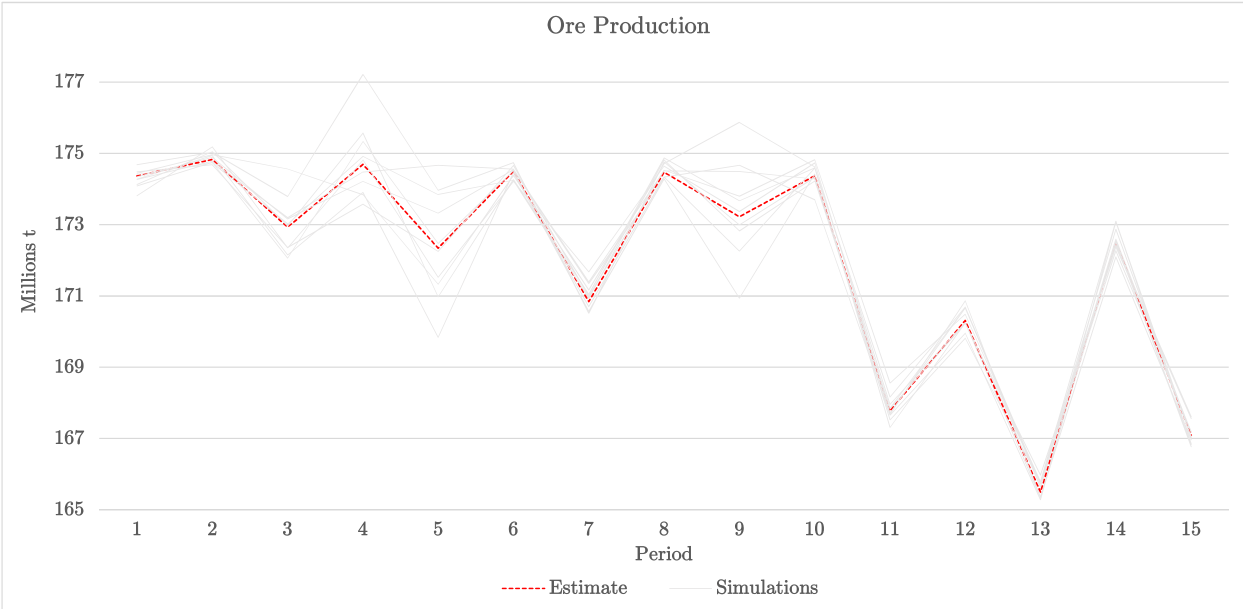

Figure 5.21 Ore production risk profile

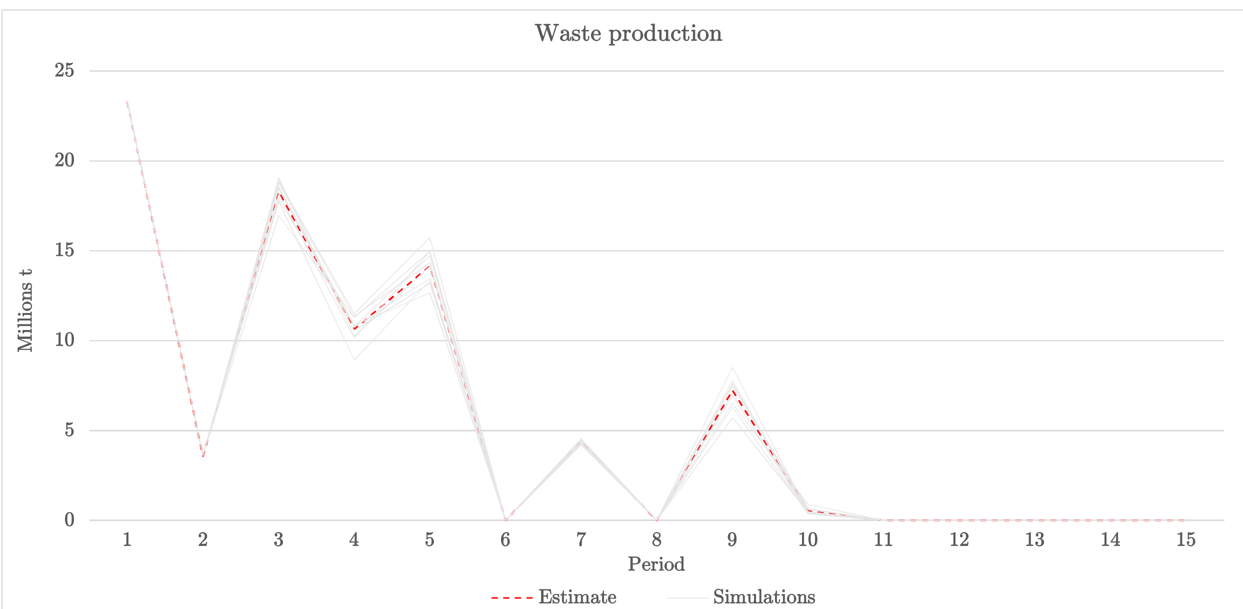

Figure 5.22 Waste production risk profile

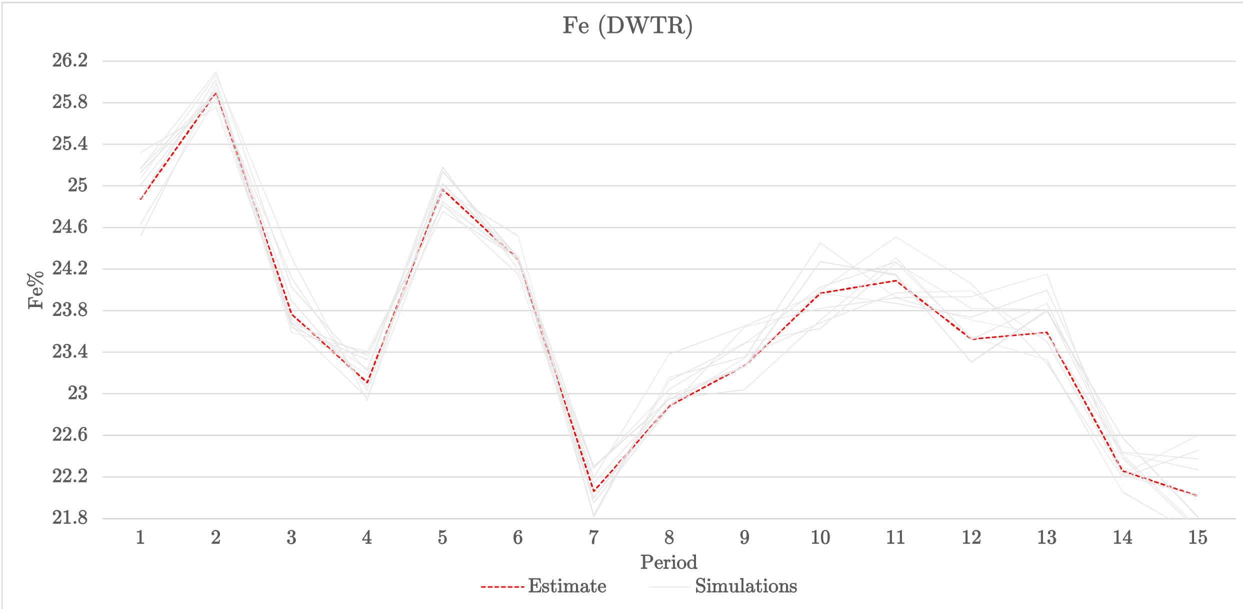

Figure 5.23 Fe% risk profile

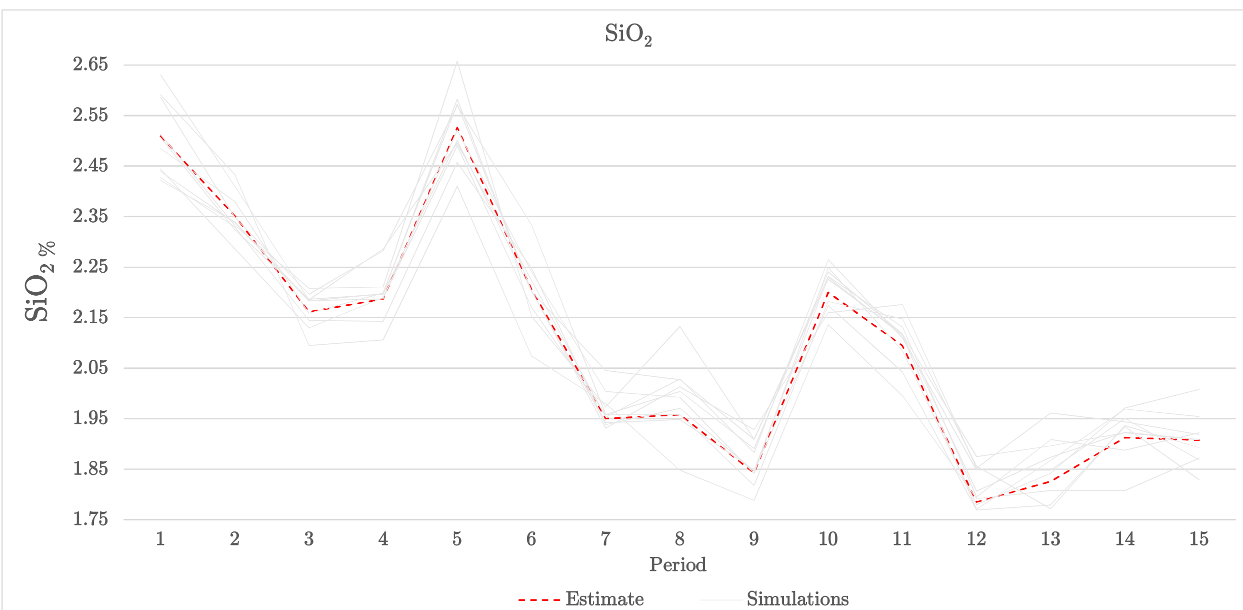

Figure 5.24 SiO_2% risk profile

CHAPTER 6 CONCLUSION

Some components of the presented model are innovative for Strategic Mine Scheduling subject. For example the integration of an IPCC system investment, installation, use and relocation over all periods providing a dynamic and optimal solution compared with the traditional static schedule methods. Additionally, the incorporation of decision variables related to capital investment for IPCC and TS systems provides a reliable solution. As a result, the model is both a new and an improved tool to study IPCC system implementation as well as TS efficiency.

- The proposed model determines which candidate location has the minimum capital investment and operating costs when mine scheduling is considered.
- The IPCC system can be flexible as trucks and shovels system and reduces haulage costs (savings in fuel, tires, auxiliary equipment). Also, it reduces OPEX (operating cost), increasing the NPV and generating higher profitability.
- The larger and deeper the open pit mine is, the more profitable the application of an IPCC system technology will be. Implementation of the IPCC system significantly reduces the number of trucks required.
- Capital costs will be lower if production requirements are large, then operating costs will be lower for the IPCC system than the conventional trucks and shovels system.
- The implementation of the transportation system by belt conveyors and crushers inside the pit can have a significant impact on its geometry, generating a readjustment in the haul roads.
- Using clusters generates feasible schedules and the solution computational time is manageable for industrial applications.

6.1 Summary

The model presented optimizes capital investment and transportation costs as a function of distance between mining work area and in-pit crusher locations and material quantity.

Case study at an iron mine evaluated interactions between IPCC and the conventional trucks and shovels system. The results show that the model is suitable generate and analyze long-term mine plans. When creating the appropriate set of belt conveyors and crusher locations, it can produce alternative mine plans that include truck equipment.

6.2 Future works

The model developed in this thesis has provided a contribution for IPCC system on strategic mine planning. Besides, there is still a need to investigate the IPCC systems implementation using optimization to be integrated on mine scheduling. The next recommendations could add value to the knowledge about this topic.

- Integrate uncertainty variability for grades and material types.
- Include temporal belt conveyor segments that can be reinstalled if needed.
- Formulate a heuristic solution for the presented model, according to the number of variables and restrictions involved in the problem.
- Investigate the IPCC implementation if the number of potential crusher locations increases.
- Add crushers capacities as variables to model the purchases of different crushers if required.

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ANNEX A AHP systems pair-wise comparison tables

Annex A.

Table A.1 Production rate comparison matrix

	SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC
Production rate	**Rail-veyor**	**1**	1/9	1/7
	Semi-IPCC	9	**1**	2
	Full-IPCC	7	1/2	**1**
	TOTAL	17.00	1.61	3.14

Table A.2 Production rate normalized matrix

SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC	Weight
Rail-veyor	0.1	0.1	0.0	6%
Semi-IPCC	0.5	0.6	0.6	60%
Full-IPCC	0.4	0.3	0.3	35%
TOTAL	1.0	1.0	1.0	1

Table A.3 Production comparison indicators

CI	**0.015**
CR	**0.900**
C Ratio	**0.017**

Table A.4 Flexibility comparison matrix

	SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC
Flexibility	Rail-veyor	**1**	1/7	1/2
	Semi-IPCC	7	**1**	5
	Full-IPCC	2	1/5	**1**
	TOTAL	10.00	1.34	6.50

Table A.5 Flexibility normalized matrix

SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC	Weight
Rail-veyor	0.1	0.1	0.1	9%
Semi-IPCC	0.7	0.7	0.8	74%
Full-IPCC	0.2	0.1	0.2	17%
TOTAL	1.0	1.0	1.0	1

Table A.6 Flexibility comparison indicators

CI	**0.012**
CR	**0.900**
C Ratio	**0.014**

Table A.7 Relocation time comparison matrix

	SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC
Relocation time	Rail-veyor	**1**	1/9	1/4
	Semi-IPCC	9	**1**	7
	Full-IPCC	4	1/7	**1**
	TOTAL	14.00	1.25	8.25

Table A.8 Relocation time normalized matrix

SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC	Weight
Rail-veyor	0.1	0.1	0.0	6%
Semi-IPCC	0.6	0.8	0.8	76%
Full-IPCC	0.3	0.1	0.1	17%
TOTAL	1.0	1.0	1.0	1

Table A.9 Relocation time comparison indicators

CI	**0.139**
CR	**0.900**
C Ratio	**0.154**

Table A.10 CAPEX comparison matrix

	SYSTEM	**Rail-veyor**	**Semi-IPCC**	**Full-IPCC**
CAPEX	**Rail-veyor**	**1**	1/6	2
	Semi-IPCC	6	**1**	5
	Full-IPCC	1/2	1/5	**1**
	TOTAL	7.50	1.37	8.00

Table A.11 CAPEX normalized matrix

SYSTEM	**Rail-veyor**	**Semi-IPCC**	**Full-IPCC**	**Weight**
Rail-veyor	0.1	0.1	0.3	17%
Semi-IPCC	0.8	0.7	0.6	72%
Full-IPCC	0.1	0.1	0.1	11%
TOTAL	1.0	1.0	1.0	1

Table A.12 CAPEX comparison indicators

CI	**0.074**
CR	**0.900**
C Ratio	**0.082**

Table A.13 OPEX comparison matrix

	SYSTEM	**Rail-veyor**	**Semi-IPCC**	**Full-IPCC**
OPEX	**Rail-veyor**	**1**	4	1/4
	Semi-IPCC	1/4	**1**	1/6
	Full-IPCC	4	6	**1**
	TOTAL	5.25	11.00	1.42

Table A.14 OPEX normalized matrix

SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC	Weight
Rail-veyor	0.2	0.4	0.2	24%
Semi-IPCC	0.0	0.1	0.1	9%
Full-IPCC	0.8	0.5	0.7	67%
TOTAL	1.0	1.0	1.0	1

Table A.15 OPEX comparison indicators

CI	**0.084**
CR	**0.900**
C Ratio	**0.094**

Table A.16 Mine geometry comparison matrix

	SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC
Mine geometry	**Rail-veyor**	**1**	1/2	1
	Semi-IPCC	2	**1**	5
	Full-IPCC	1	1/5	**1**
	TOTAL	4.00	1.70	7.00

Table A.17 Mine geometry normalized matrix

SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC	Weight
Rail-veyor	0.3	0.3	0.1	23%
Semi-IPCC	0.5	0.6	0.7	60%
Full-IPCC	0.3	0.1	0.1	17%
TOTAL	1.0	1.0	1.0	1

Table A.18 Mine geometry comparison indicators

CI	**0.064**
CR	**0.900**
C Ratio	**0.071**

Table A.19 Material movement comparison matrix

	SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC
Material movement	Rail-veyor	**1**	1/3	1
	Semi-IPCC	3	**1**	1
	Full-IPCC	1	1	**1**
	TOTAL	5.00	2.33	3.00

Table A.20 Material movement normalized matrix

SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC	Weight
Rail-veyor	0.2	0.1	0.3	23%
Semi-IPCC	0.6	0.4	0.3	45%
Full-IPCC	0.2	0.4	0.3	32%
TOTAL	1.0	1.0	1.0	1

Table A.21 Material movement comparison indicators

CI	0.074
CR	0.900
C Ratio	0.082

Table A.22 Material properties comparison matrix

	SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC
Material properties	Rail-veyor	**1**	1/8	2
	Semi-IPCC	8	**1**	8
	Full-IPCC	1/2	1/8	**1**
	TOTAL	9.50	1.25	11.00

Table A.23 Material properties normalized matrix

SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC	Weight
Rail-veyor	0.1	0.1	0.2	13%
Semi-IPCC	0.8	0.8	0.7	79%
Full-IPCC	0.1	0.1	0.1	8%
TOTAL	1.0	1.0	1.0	1

Table A.24 Material properties comparison indicators

CI	**0.053**
CR	**0.900**
C Ratio	**0.059**

Table A.25 System Properties comparison matrix

	SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC
System Properties	**Rail-veyor**	**1**	1/8	2
	Semi-IPCC	8	**1**	6
	Full-IPCC	1/2	1/6	**1**
	TOTAL	9.50	1.29	9.00

Table A.26 System Properties normalized matrix

SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC	Weight
Rail-veyor	0.1	0.1	0.2	14%
Semi-IPCC	0.8	0.8	0.7	76%
Full-IPCC	0.1	0.1	0.1	10%
TOTAL	1.0	1.0	1.0	1

Table A.27 System Properties comparison indicators

CI	**0.102**
CR	**0.900**
C Ratio	**0.114**

Table A.28 Enviroment & Safety comparison matrix

	SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC
Enviroment & Safety	**Rail-veyor**	**1**	5	1
	Semi-IPCC	1/5	**1**	1/6
	Full-IPCC	1	6	**1**
	TOTAL	2.20	12.00	2.17

Table A.29 Enviroment & Safety normalized matrix

SYSTEM	Rail-veyor	Semi-IPCC	Full-IPCC		Weight
Rail-veyor	0.5	0.4	0.5		44%
Semi-IPCC	0.1	0.1	0.1		8%
Full-IPCC	0.5	0.5	0.5		47%
TOTAL	1.0	1.0	1.0		1

Table A.30 Enviroment & Safety comparison indicators

CI	**0.002**
CR	**0.900**
C Ratio	**0.003**