THE UNIVERSITY OF QUEENSLAND

BACHELOR OF ENGINEERING THESIS

The Implications of Improved Conveyor Technology on In-Pit Crusher Conveyor Systems

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ABSTRACT

Open pit mines are beginning to reach depths previously considered impossible, due to modern technology, geotechnical advances and fluctuating commodity prices. Open pit haulage costs currently account for up to 60% of the total mining operational costs, and with increasing depth, the haulage distance and hence the number of trucks required to satisfy production has increased. This creates opportunities for alternative material transportation methods to replace the conventional TS system. This research project aims to investigate the performance of a semi-mobile IPCC system as replacement to TS operations in an open pit copper-gold mine.

An extensive review of literature regarding open pit mining systems and mine planning concluded that there is a gap of understanding in the industry for determining the optimum system for open pit metalliferous mining. A review of literature highlighted a shared ideology that whilst IPCC systems present the benefit of much lower operating costs, increased safety, and lower dust and GHG emissions, the system represents a significant up-front capital investment which the mine is only able to depreciate over a sufficient mine life. IPCC implementation requires extensive mine planning and scheduling, often requiring large waste production early on in the mine life, deterring mine planners away from the advantages that the system is capable of providing.

A copper-gold deposit block model underwent mine planning using two separate mining systems, semi-mobile IPCC as well as TS. Two IPCC pits were designed, IPCC 26B and 36B at depths of 390 m and 540 m deep respectively. A conventional TS pit was designed, with an optimum depth of 525 m. IPCC yielded the greatest total undiscounted pit revenue, however applying production scheduling and a discount rate to the revenues found that TS had the greatest total discounted value. IPCC presented lower operational costs than TS, with savings of up to $1.43 per tonne of material moved, and up to $36 M annually in OPEX.

Applying the capital costs on a yearly basis found that TS had the lowest CAPEX requirement, due to truck implementation over time compared to the large upfront CAPEX that was necessary for IPCC operations. Using the yearly cash flows, OPEX and CAPEX, an FTM was generated. The TS pit yielded the greatest NPV, with $1.325 B, including an IRR of 63%. Making the TS mining method the most economically viable mining method for the given deposit, and the most ideal mining system in countries where NPV is regarded as the highest priority in mine valuation. Whereas, the large recovery presented by IPCC 36B indicates that

IPCC systems can be selected as the most ideal mining system in countries that value recovery over NPV in mine valuation, such as Post-Soviet countries.

Further work consisted of investigating similar comparisons between IPCC and TS systems with varying deposits, pit exit strategies and crusher portability modes. It was also recommended that an existing IPCC mine be compared with a theoretical TS mine simulation, using real life cash flows, OPEX and CAPEX to reduce the chance of error in the investigation.

GLOSSARY

AUD	Australian dollars.
D&B	Drill and blast.
CAPEX	Capital expenditure.
FMEA	Failure mode and effects analysis.
FTM	Financial technical model.
GHG	Greenhouse gasses.
Heavy Vehicle	An oversize or overmass heavy vehicle (i.e haul truck).
IPCC	In-pit crushing and conveying.
IRR	Internal rate of return.
Mining Plant	Machinery, equipment, appliance or tool.
NPV	Net present value.
Pass	One full shovel load that is dumped into a haul truck.
TS	Truck and shovel.
USD	United States dollars.
WRD	Waste Rock Dump.

CONTENTS

LIST OF FIGURES

LIST OF TABLES

1. INTRODUCTION

1.1. BACKGROUND INFORMATION

Surface mining is the predominant mining method world-wide, resulting in approximately 90% of total mineral production (Wetherelt, 2011). However, easily accessible deposits have been depleted, causing companies to mine at greater depths and with larger scales of operations (Paricheh, Osanloo and Rahmanpour, 2016). Due to modern technology, geotechnical advances and fluctuating commodity prices, open pit mines are beginning to reach depths previously considered unviable, such as Chuquicamata mine in Chile (850 metres deep). Historically, TS systems have been the preferred mining method for open pit mines, however with increasing depth, the haulage distance and the number of trucks required to satisfy production has increased, creating opportunities for alternative material transportation methods.

In-Pit Crushing and Conveying (IPCC) is a mode of material transportation that utilises conveyors and stackers in conjunction with fully mobile, semi-mobile or fixed in-pit crushers. Developed in Germany in the 1950s, IPCC systems have since been installed in over 200 mines world-wide (Koehler, 2003). For large open pit mines, the benefits of IPCC over TS are summarised below:

- Continuous material haulage;
- Increased mine and road safety due to fewer moving vehicles on mine roads;
- Reduced road maintenance requirements (water trucks etc.);
- Reduced dust generation;
- Lower labour requirements;
- Electrical power (IPCC) is more cost effective than diesel; and
- Reduced GHG emissions and noise generation (Jeric and Hrebar, 2008).

IPCC systems are currently used at greater pit depths where the effort of IPCC design, scheduling and mine planning can be justified (Ko, 1993). Currently the upfront capital expenditure required to establish crushers and conveyors ensure IPCC is unsuitable for small to medium sized operations.

1.2. PROBLEM DEFINITION

During mine planning, mining method selection is critical for pit design, as it directly affects the success of the mine. As metalliferous open pit mining advances to new depths, companies search for cost effective alternatives to the conventional TS mining method. Increased haulage routes, and associated fleet size and operational costs, result in significant TS material transportation costs. Due to the lower operational costs, IPCC is considered a viable competitor to conventional TS operations. No quantitative assessments currently exist on the comparison between IPCC and TS performance in open pit metalliferous mining. A comprehensive comparison between the two methods on an identical metalliferous orebody will provide a basis for companies to make an informed decision to optimise mine design.

1.3. AIMS AND OBJECTIVES

This research project aims to investigate the performance of a semi-mobile IPCC system as replacement to TS operations in an open pit copper-gold mine. The key objectives of this research project are listed below:

- Undertake background research into the current state of IPCC systems, including mine planning requirements, economics and comparison to TS haulage.
- Develop a conceptual porphyry copper-gold block model.
- Undertake mine planning and scheduling on the proposed deposit utilising a) a semi-mobile IPCC system and b) a TS system.
- Evaluate the economics of both systems, including cash flows, operating and capital costs.
- Compare the economics between IPCC and conventional TS methods based on results.
- Validate of the conceptual deposit, mine plan, economic evaluation, results and data.

1.4. SCOPE

This research project is a comparative analysis of the IPCC mining method and TS based on the different financial results of extracting a conceptual porphyry copper-gold deposit. The following is a list of all factors in the scope of this project:

- Open pit design is limited to copper-gold deposit;
- IPCC system under consideration is semi mobile;
- Economical evaluation conducted will consider both mining methods; and
- Sequence and schedule for both mine designs uses a constant production rate.

The following factors will not be considered in the research project:

- Legal, social and political ramifications of the mine;
- IPCC performance in different deposit types;
- Waste rock dump (WRD) design and spreader scheduling;
- Underground mining considerations; and
- Geotechnical analysis of TS open pit.

1.5. SIGNIFICANCE AND RELEVANCE TO INDUSTRY

The benefits of this research project to the industry include an improved understanding of the performance of IPCC systems in an open pit copper mine scenario. Mine planners and schedulers are under constant pressure to optimise production throughout the mine life. With a better understanding of the IPCC performance in varying conditions, planners will be able to confidently select material transportation methods. In establishing the advantages and disadvantages of IPCC implementation, mine planners can select the best method of material transportation to suit pit conditions, assisting the mining company to achieve the best possible economic, productivity and safety outcomes.

1.6. METHODOLOGY

A number of tasks were conducted in order to achieve the objectives of the research project. An initial review of current literature was conducted to investigate the current state of IPCC systems in open pit metalliferous mines. This review of literature includes IPCC technology, mine planning requirements, economics and comparison to TS haulage.

Following the literature review, a conceptual block model for a porphyry copper-gold deposit was created. This deposit was investigated and detailed mine planning was undertaken on the deposits for both semi mobile IPCC systems as well as conventional TS method. An optimum open pit design and schedule was determined for both haulage methods, utilising the advantages of each mining method.

The two mine designs differ in final pit limit, pit geometry, life of mine, distance to dump site and more due to the respected limitations and requirements. The variables that were left constant in this investigation to accurately compare both methods are as follows:

- Production rate;
- Distance from pit to WRD/mill;
- Metal selling prices;
- Processing cost;
- Infrastructure CAPEX; and
- Financial technical model (FTM) discount rate.

The two mines underwent economic evaluation and financial modelling, considering the net present value (NPV), revenue, CAPEX, OPEX, tax considerations etc. The results of the simulations were analysed and any necessary refinements and iterations were made to the mining parameters and mine designs in order to improve accuracy of the results. This lead to an in-depth comparison between the two mines' economic results, investigating the cash flow differences at various stages, the rate of return and NPV.

Ultimately, recommendations were made in regards to the preferred material haulage method. The details of the investigation, including all research, conceptual mine, mine plan and schedule, economic results and analysis were outlined in a comprehensive project report.

2. RISK ASSESSMENT

2.1. OVERVIEW

Risk analyses are a critical part of any mining operation and should be performed as a part of the risk management process for each project. A risk analysis helps identify and manage potential problems that could undermine key business initiatives or projects (MindTools, 2016). Identification and consideration of any risks that may affect this research project is critical in ensuring its completion and quality. Recognizing the risks prior to occurrence will minimise their impact upon the project. In order to identify key risks, a failure mode and effects analysis (FMEA) was developed for the research project. Following the FMEA plans will ensure the project is completed on time and to the highest standard. The FMEA table can be found in Appendix A.

2.2. TECHNICAL RISKS

The primary project objectives of this research task are:

1. Submit all project tasks on time;
2. Submit project tasks to a high standard; and
3. Maintains significant relevance in industry.

The inability to accomplish these objectives will result in functional failures. A functional failure can be defined as *the inability of an asset to fulfil an intended task to a standard performance that is acceptable to the user of the asset* (Weibull, 2009). These failures will occur if:

- Research project is not completed;
- The standard of the research project is lower than desired;
- Data used in the research proposal is incorrect;
- The research project does not hold relevance to industry;
- Research project strays too far from intended objective; and
- Research project conclusions and recommendations cannot be applied to industry.

The following is examples of how functional failures may occur:

- Failure to submit research project;
- Personal illness or absence due to injury;
- Poor time management;
- Lost report and/or corrupt files;
- Failure to research literature regarding research topic;
- Data is insufficient to use for research project;
- Failure to meet deadlines for milestones;
- Failure to peer review and edit report; and
- Conclusions are not relevant.

2.3. Risk Ranking

Risk ranking is a common method used in risk management to provide a sharper focus to the critical risks within a system. The risk matrix can be used to *define the level of risk by considering the category of probability or likelihood against the category of consequence of severity* (Hubbard, 2016). The risk-ranking matrix developed for the completion of the project is illustrated in Table 1.

Table 1.
Risk ranking matrix.

		Consequence				
Likelihood		**Insignificant (1)**	**Minor (2)**	**Moderate (3)**	**Major (4)**	**Severe (5)**
	Almost Certain (5)	Medium (5)	High (10)	Very High (15)	Extreme (20)	Extreme (25)
	Likely (4)	Medium (4)	Medium (8)	High (12)	Very High (16)	Extreme (20)
	Possible (3)	Low (3)	Medium (6)	Medium (9)	High (12)	Very High (15)
	Unlikely (2)	Very Low (2)	Low (4)	Medium (6)	Medium (8)	High (10)
	Rare (1)	Very Low (1)	Very Low (2)	Low (3)	Medium (4)	Medium (5)

2.4. CORE RISKS

By applying the severity and likelihood ratings to the various risks, as shown in Appendix A, all risks involved in this research project were ranked. The top five risks were recognised as having the greatest damage on the project should they occur. The top five risks are:

1. Failure to submit research project;
2. Personal illness or absence due to injury;
3. Data is insufficient to use for research project;
4. Lost report and/or corrupt files; and
5. Failure to peer review and edit report.

2.5. CONTINGENCY PLANS

A contingency plan is a course of action designed to help respond to a form of failure, ensuring the project can be completed even if a key failure occurs. A contingency plan was developed for the research project and is detailed in Table 2.

Table 2.
Contingency plan.

Risk	*Contingency*	*Consequence*
Failure to submit research project	Set milestones, complete project ahead of time and work during holidays and breaks	Research project cannot be credited to university studies, hence failure
Personal injury or absence due to illness	Familiarise self with extension application process, maintain healthy lifestyle	Research cannot be submitted or extended time for project completion is required
Data is insufficient to use for research project	Follow project schedule to allow time to assess data, re-run model, alter variables and parameters	Collected data is incomplete or unreliable, providing insufficient conclusions
Lost report and/or corrupt files	Backing up data in multiple locations, for example online and external hard-drives	Extended time for project completion, failure to submit or quality is reduced
Failure to peer review and edit report	Follow project schedule, edit sections of the project report along the way, make sure peer review is able and willing to assist	Jeopardise the quality of work by not editing and checking report

3. PROJECT MANAGEMENT

3.1. OVERVIEW

This research project was conducted during the course of the 2017 academic year. The Examiner's thesis was completed by October, concluding all research, data analysis and reporting. During the first semester the project research proposal was created to focus and define the research plans. This was followed by an annotated bibliography of a wide range of resources, leading to a thorough review of existing literature relative to the project. The second semester included data analysis, discussion of the results and conclusions to be used in the industry.

3.2. TASKS AND ACTIVITIES

A number of tasks have been completed in accordance with the project schedule. The critical ones being:

- A research proposal detailing the background of the study, as well as the aims, objectives, scope, methodology and significance to the industry;
- A detailed annotated bibliography of prior literature related to IPCC, their applications and comparing them to other modes of haulage;
- A comprehensive review of existing literature on hard rock open pit mining, open pit planning, modes of haulage and comparison of IPCC to T&S;
- Block model development;
- Typical parameters and costs associated with IPCC and T&S methods to form the basis for the model;
- A risk assessment of project, with the failure modes and core risks identified, as well contingency plans to reduce the severity of the associated risks;
- Open pit mine design on the deposit utilising a) a fully mobile IPCC system and b) a T&S system using *Deswik* sofware;
- Economic evaluation of both systems, including cash flows, operating and capital costs;
- Interpretation and analysis of model results, comparing the performance of both systems; and
- Examiners copy thesis, summarising all findings and conclusions.

3.3. REQUIRED RESOURCES AND BUDGET

During the course of this research project, numerous resources were necessary for the project to be completed to schedule and to the desired standard. The resources that were used in the research project are as follows:

- Access to university computer with licences for *Deswik*, *Whittle*, *Talpak* and *Microsoft Excel*;
- Access to university library and databases;
- Access to university academic supervisor; and
- Access to printing.

A budget was created to provide an estimate for the cost of conducting this research proposal. Numerous costs were estimated.

Table 3.
Project milestones with completion dates.

Item	*Quantity*	*Cost ($)*	*Total Cost ($)*
University PC	1	1500	1500
University Deswik Licence	1	2500	2500
University Microsoft Excel Licence	1	200	200
Academic Supervisor	20 hours	200/hr	4000
Undergraduate Researcher	800 hours	45/hr	36 000
Printing (<50 pages)	4	15	60
Printing (>50 pages)	3	60	180
		Total	44 440

3.4. CRITICAL PATH

A Gantt chart was created to present the timeline of the project, and the critical path to completion is indicated. The Gantt chart can be found in Appendix B.

4. LITERATURE REVIEW

4.1. OPEN PIT HARD ROCK MINING

An open pit mine can be defined as *an excavation or cut made at the surface of the ground for the purpose of extracting ore and which is open to the surface for the duration of the mine's life* (Mine-Engineer, 2007). The objective of any mining operation is to extract the mineral deposit at the lowest possible cost in order to maximise profits. The method in which the mineral is extracted is a critical factor in the mine's economic and safety success. Open pit mining is a popular technique to exploit hard rock minerals (minerals that exhibit compressive strengths greater than 80 Mpa) that lie near-surface, including metal ores such as copper, gold, silver and iron. If the deposit is not exploited using combined open cut and underground methods, then the pit floor represents the lowest level of the deposit that is exploited.

The pit is formed in benches, which are ledges that form single levels of operations to which minerals are mined. Hard rock deposits require multiple benches dug at an angle to minimise structural weaknesses, creating an inverted cone like mine design. The ultimate pit depth is often the most economically viable shape and depth. The final pit design is the result of multiple pushbacks, in which the pit is expanded vertically and horizontally to optimise the NPV of the mine. Figure 1 depicts a pit cross section, identifying characteristics.

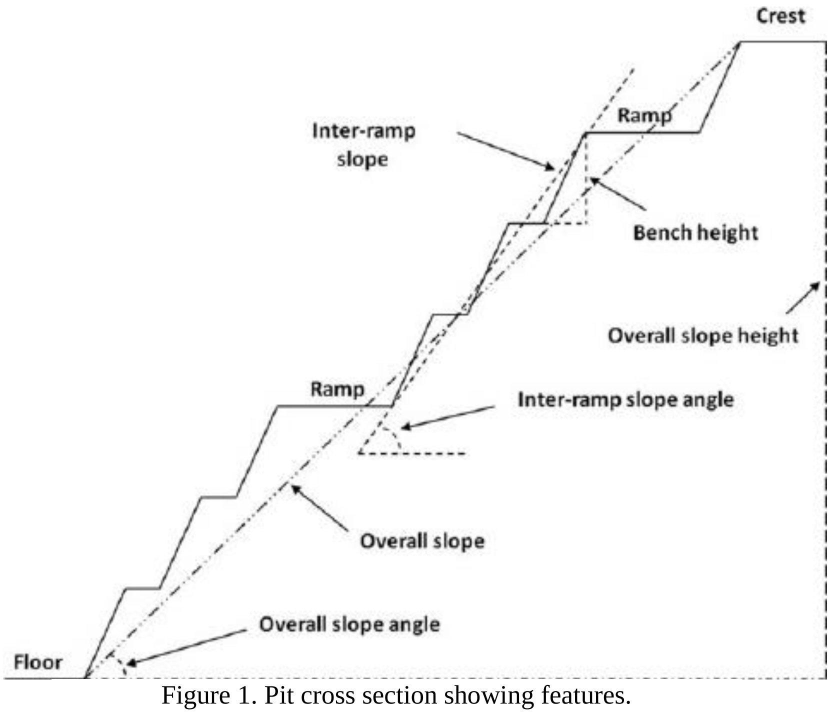

Figure 1. Pit cross section showing features.

The bench height is typically determined by using the reach of the bucket height on the excavator being used in the mine, while the bench slope is dependent upon the stability criteria (Hem, 2012). The bench width also functions to catch rolling rocks off the bench slope, however in pushback production the bench width must also be big enough to accommodate the mining plant (i.e. an excavator and a truck).

In comparison to underground mines, surface mining achieves quicker access to deposits, lower capital costs, higher production rates, and greater flexibility and workplace safety (Mine-Engineer, 2007). Whilst underground mines target minerals that are of higher grade, they are more time consuming, capital intensive and less efficient. With so many considerations and operations, open pit mining requires extensive planning in the design phase to maximise the profitability, efficiency and productivity.

4.2. OPEN PIT PLANNING

Open pit planning can be defined as the decision making process that leads to a realistic and coherent plan to extract mineral resources (Whittle, 2011). Open pit planning considers the pit design, mine sequence, ore selection, haulage method and mining method. Pushbacks are a useful tool in open pit planning, assisting in early ore access hence increasing the NPV of the project. However, implementing excessive pushbacks can lead to unnecessary expense to maintain multiple working slopes, as well as operational problems. In recent times there have been many attempts at determining the most appropriate technique to define the pit limits. Two notable techniques are the Lerchs Grossman and the Floating Cone technique.

In 1964 Helmut Lerchs and Ingo Grossmann developed an algorithm to find the optimum design for an open pit mine. In their words, the objective of their algorithm is *to design the contour of a pit so as to maximize the difference between total mine value of the ore extracted and the total extraction cost of ore and waste* (Lerch & Grossman, 1964). The solution, called the Lerchs Grossman method, uses the economic block model to create a directed graph, consisting of nodes and a set of connecting arcs. This directed graph dictates the order in which the blocks should be removed (Khalokakaie, Dowd & Fowell, 2013). The Lerchs-Grossmann can be relied on to yield the optimum pit limits in regards to orebody removal. The Lerchs-Grossman method is used in mining optimisation software as an industry standard.

A less complex method for determining final pit limits was described by Carlson *et al* in 1966. The Floating Cone method defines the pit limits on what is economically feasible to remove. Cones are created pointing towards a positive block, and includes the five blocks in the row above that must be removed to access the positive block and so forth. The value of the cone is simply the sum of each block value contained in the cone, if the sum of the cone is positive, it is deemed economical (Khalokakaie, 2011). The floating cone method is a simplistic approach to determine the ultimate pit, and contains numerous limitations. The floating cone method does not take into account neighbouring block values and has the potential to miss combinations of profitable blocks.

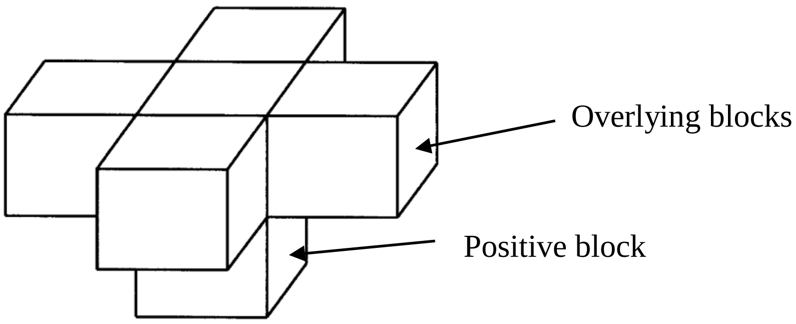

Figure 2. Diagram of the floating cone method principal, identifying the six blocks needed to be extracted for the one block below.

5. OPEN PIT HAULAGE METHODS

5.1. INTRODUCTION

The mining industry worldwide is facing considerable economic and social constraints due to rising labour costs, falling mineral prices and stricter environmental regulations. Transportation of blasted rock and waste to the primary crusher and waste rock dump (WRD) can account for up to 60% of the total operational costs for a mine (May, 2012). Removing waste to uncover the ore deposit in an open pit mine creates a negative return in operations, albeit is a necessary expense item. As a result, considerable attention is being paid into improving current transportation technology or seeking alternate transport technologies that can reduce the cost whilst upholding mine productivity. Currently the two major haulage methods in open pit mines are TS and IPCC.

5.2. TRUCK AND SHOVEL HAULAGE

Large-scale open pit mining has featured prominently in the Australian hard rock industry since the early 20th century (Humphrey and Wagner, 2012). A steadily increasing demand for materials in conjunction with deeper pits is forcing companies to utilise larger equipment to meet production targets. The preferred method of open pit mining is TS, utilising haul trucks as the method of haulage. Haul trucks are low investment cost, high production earth moving equipment. TS operations move more material than all other mining systems combined (Humphrey and Wagner, 2012).

5.2.1. *System Overview*

Truck and shovel mining refers to any load-haul-dump system employing a loading unit and haul trucks to move material. Hard rock mining requires drill and blasting (D&B) prior to the loading and haulage to loosen the material. Loaders such as electronic mining shovels, front end loaders and hydraulic shovels load and dump the loose material into haul trucks over a number of passes. Haul trucks then drive back and forth from the loader to the waste rock dump in fleets. When selecting the operating equipment to be used in a TS system, both the trucks and shovels need to be matched based on their characteristics. The choice of loader is often based on the bench height and production of the open pit mine. The choice of truck is determined based on the bucket capacity of the loader, three to five loading passes to fill a haulage truck has long been accepted as industry standard practice (Humphrey and Wagner, 2012).

TS methods are popular in many situations. When deposits are geologically complex, the flexibility of the trucks allows for simplistic mine planning, allowing for irregular pit shapes. When no clear life of mine is established, truck shovel operations are the preferred option, as incremental implementation of additional trucks is relatively seamless. TS operations are best suited for small deposits, where the capital costs of more expensive mining methods are not justified, whilst the small investment for a haul truck results in faster payback. In addition, truck downtime is relatively minimal due to the interdependency of operating units. However, with the current trends of global mining industry, such as lower commodity prices, high operating costs and longer haulage routes, TS efficiency has come under the spotlight.

Haulage costs account for as much as 60% of the mining cost for open pit mines, and it is essential to maintain an efficient haulage system (May, 2012). Figure 3 illustrates the TS process, identifying some of the possible system bottlenecks. A larger fleet size often results in increased loader productivity but decreased truck productivity; therefore the fleet size must be optimised (May, 2012). When the number of trucks outbalances the loader, queuing may occur, whereby the trucks sit idle at the loader or waste dump while another truck is loading or dumping. Spot time is the time from the last load for a full truck to the first bucket for an empty truck. Both queuing and spot time represent significant wastes of energy and labour, showcasing the negative effects of discontinuous labour. Table 4 summarises the typical cost distribution of TS operations.

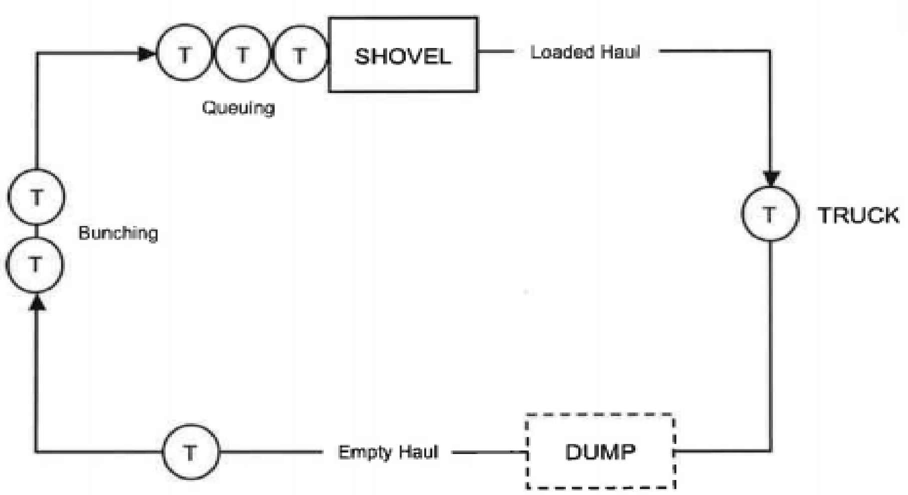

Figure 3. Schematic of truck-shovel operation (Carmichael, 1986).

Table 4.
Typical cost distribution of TS operations (Ko, 1993).

Mining Process	*% Total Cost*
Drilling	5.00
Blasting	14.00
Loading	9.00
Hauling (diesel)	30.00
Hauling (other costs)	27.00
General Mining Services	15.00
Total Mining Cost	100.00

The operational costs of large haul truck fleet sizes in deep open pit mines can often outweigh the benefits of low capital expenditure. The main sources of operational costs are:

- Energy consumption;
- Labour costs; and
- Equipment costs.

Increased production demands results in an increase in energy consumption. Haul trucks can be powered using diesel or electricity, however the latter is less popular in mining operations due to in-pit power line requirements. Energy consumption occurs in truck shovel operations in four ways; shovel loading and idle, and truck loading and idle. Truck fuel consumption changes from truck to truck, with the smaller truck sizes averaging approximately 40 l/hr and the larger trucks averaging approximately 90 l/hr (ClassicMachinery, 2009). Applying these consumption rates to large truck fleets (20 to 60 trucks is not uncommon) operating on 24 hour rosters with fluctuating fuel prices amounts to immense operational costs.

Truck haulage methods have high labour requirements, with a single truck consisting of up to seven operating and maintenance personnel 24 hours of the day (Knights, 2015). Additionally, the consumables associated with truck operations such as tyre replacements increase with larger production. The use of diesel as a source of power for machinery produces significant greenhouse gas emissions (GHG). CO_2 is produced at a rate of approximately 2.73 kg/litre of diesel, therefore a fully operating mine with a large fleet of diesel powered trucks will contribute heavily to the mines carbon footprint (Williams, Ackerman, & Nati, 2009).

5.3. Conveyor Systems

5.3.1. *Overview*

A conveyor system is a mechanical handling system which moves materials from one location to another (MidlandConveyors, 2012). Especially useful for transporting heavy or large materials, conveyor systems allow for continuous operations, making them very popular in the material haulage industry. Conveyor systems range in shape and characteristics, made to suit the conditions upon it operates. For example, a conveyor system created to move boxes in a packaging factory may use a line-shaft roller conveyor, whereas loose materials need to be transported on a medium that will prevent spilling.

5.3.2. *Conveyor Belts*

A conveyor belt is a type of conveyor system which employs a carrying medium, often called the belt, which loops around two or more pulleys to transport materials. The pulleys are powered so that they rotate the belt, allowing for an endless loop of carrying medium, as illustrated in Figure 4. It is not uncommon for only one pulley to be powered, rotating the whole system, this is called the driver pulley, and the unpowered pulleys are called idle pulleys. The conveyor belt system is extremely popular in bulk material handling industries, whereby dry materials such as ore, coal, salt, sand and grains are transported in loose bulk form (Ko, 1993). Belt systems can range up to multiple kilometres, with the longest belt conveyor system in the world sitting at 98 km long in Western Sahara (BBC, 2011).

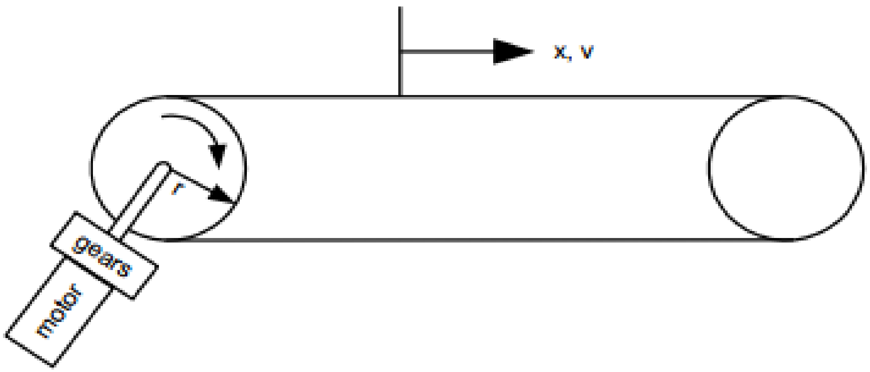

Figure 4. Diagram of the conveyor belt system.

In an 1868 paper submitted to the British Engineering Society, Lyster described his work on conveying bulk materials using endless belts. Lyster was responsible for creating many of the conveyor technology that is still used in present day conveyors. In the late 1800's, Webster made significant progress on conveyor belts to handle grains, which was further improved on by Edison, who in the 1890's refined the belts to be able to handle ore material types (Ko, 1993). Thomas Robins, another pioneer of the conveyor belt, introduced the three roll idler set, increasing ore throughput capacity.

In 1905, Richard Sutcliffe combined the idea of cheap material haulage with the production hungry coal industry, and invented the first conveyor belt haulage method for coal mines, revolutionising the mining industry (Ko, 1993). By reducing the need for human labour transporting coal in underground coal mines, Sutcliffe cut costs and increased production in coal mines across Ireland and the UK. Further refinements to these original conveyors have been made by conveyor manufacturers across the world in the last 120 years (Ko, 1993).

The modern day conveyor belt system consists of the following four elements:

1. Carrying belt;
2. Driving unit;
3. Idlers and supported structures; and
4. Associated accessories, including; devices for belt tension, loading, unloading and cleaning.

The conveyor belt is the most important feature in the system (Ko, 1993). In the material haulage industry, a belt must satisfy a number of qualities. It must be flexible enough to bend round pulleys, creating the endless belt feature. The belt must have the mobility to trough under its own weight to conform to the angle of the idler pullies. Finally, the belt must be strong enough to transmit the necessary tension required for the belt to hug closely to the idlers. To satisfy these requirements, conveyor belt systems often use rubber, canvas, chains or metal aprons to form the belt.

The method in which belt conveyors can receive material from loaders is very flexible, being able to receive material from one or more stackers at a time. Material can be unloaded simply by rolling over the head end, or during the length of its travel using plows or travel trippers.

The carrying and return idlers are almost always a three pulley type, including a flat idler in the centre, with an idler either side often inclined 20° to the horizontal. The spacing between the idlers needs to be optimised to reduce shock, increasing belt life (Alspaugh and Bailey, 2005). Diameter of the idler is also a critical design feature of the conveyor, whereby increasing the diameter can reduce the belt wear. The supporting frame should be rigid. The density, angle of internal friction, lump size, shape and effective angle all dictate the incline angle which material can be conveyed. Conveyor Equipment and Manufacturers Association (CEMA) standards state that conventional trough conveyor incline should range between 10 – 30°, depending on the material properties (Alspaugh and Bailey, 2005).

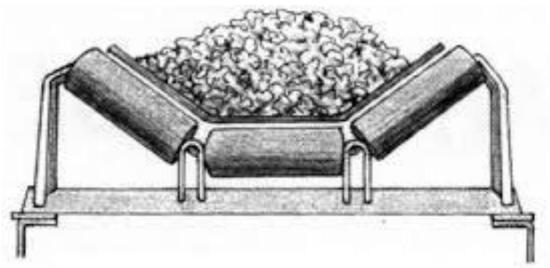

Figure 5. Conveyor belt idlers showing material positioning (Transmission Products, 2000).

In almost every industry, conveyor belts have proven to be a reliable haulage method. Individual conveyors can have a total availability of up to 95%, whilst more intricate conveyor systems incorporating more than one conveyor belt operates at approximately 90% availability (CEMA, 2005). Conveyors are designed to operate for long periods on end, and can be effective in industries where continuous haulage is critical for success. The system is controlled and operated via computer technology. The availability of a conveyor system is also dependent on operator efficiency, making operator training essential for success (Alspaugh, 2005).

Belt conveyors are more environmentally friendly and operate at a higher degree of safety than other modes of haulage. Conveyors are electrically powered, allowing for a quiet and relatively pollutant free form of haulage. Enclosing the conveyor system controls dust, and prevents spilling unwanted materials into the environment. Conveyors can be blended into the landscape, raised or designed underground, reducing safety and environmental concerns. Compared to alternative haulage systems, conveyors operate with fewer personnel, reducing the exposure to hazards for workers. Modern day conveyors require safety devices such as emergency pull cord switches to provide maximum protection to the personnel as well as the equipment (CEMA, 2005).

5.3.3. *Conveyor Economics*

Conveyor systems have reduced labour costs, as the entire system can be monitored from a single control panel, hence requiring a minimum number of workers to be involved in system operations. These low labour costs in turn allow for lower material haulage operating costs, providing a higher return on investment. Maintenance related labour costs are also extremely low in comparison to alternative haulage methods. Maintenance outage durations for high capacity continuous conveyor belts rarely exceed eight hours a week, and major belts can be repaired or replaced in one week (CEMA, 2005). Conveyor belts can often convey over 100 million tonnes before needing replacing or maintenance.

Conveyor belts are powered by electric power and are therefore less affected by the burdens that come with liquid fuel, such as price fluctuations and shortages. Due to the system design, conveyors do not encounter empty return trips and idle time that alternative methods encounter; hence conveyors only consume power when transporting material. As petroleum based fuel prices continue to rise, the low cost of electrical power further separates the operating cost of conveying to other modes of haulage.

5.3.4. *Types of Conveyors in Mining Industry*

Within the mining industry, various types of conveyor belt systems exist to facilitate the different conditions of each mine. The most popular types of belt systems used in mining are belt conveyors, pipe conveyors, apron conveyors, chain conveyors and sandwich conveyors.

The pipe conveyor is an enclosed conveyor system that utilises the mobility of the belt to effectively create a moving pipe, allowing for vertical and horizontal curves. Material is placed on the belt as per usual, the rollers then form the belt into a tubular shape to allow for enclosed transportation of material, protecting the material from climatic conditions whilst avoiding material loss and spillage. At the discharging point, the rollers allow for the belt to open automatically, dumping the material at the desired location. Figure 6 visualises the pipe conveyor set up.

Figure 6. Pipe conveyor system showing the rolling phase.

Pipe conveyors are popular in the mining industry because they offer enclosed transportation, posing no risk to the environment or risk of spilling the material. Depending upon the diameter of the moulded pipe and the properties of the material being handled, throughput rates can range up to 10,000 t/h (Bridgestone, 2012). Whilst the enclosed shape allows the conveyor to operate with shorter vertical and horizontal curves than regular belt conveyors, the vertical slopes that it can offer are not as remarkable as the sandwich conveyor.

In 1979, Dos Santos studied the means of moving and elevating large quantities of bulk materials at the steepest possible inclines. By only using readily accessible hardware and materials, Dos Santos developed his own sandwich belt design. His design uses two rubber conveyor belts, face-to-face, firmly holding the material together and allowing the conveyor angle to increase whilst preventing fallback (Dos Santos, 2016). The material is held together by the hugging pressure supplied by the additional belt, ultimately creating enough frictional force to prevent the material sliding back. This principal is illustrated in Figure 7.

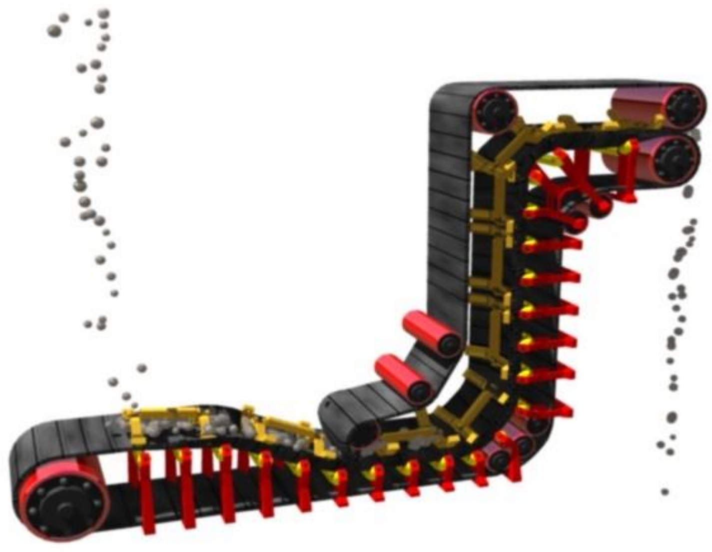

Figure 7. Dos Santos sandwich conveyor diagram.

Since then, over 150 systems have been implemented into material haulage operations worldwide, with throughputs ranging up to 6000 t/h (Dos Santos, 2016). The simplistic approach, high lifts and high conveying angles as well as impressive material throughputs have made the system extremely popular in open pit mining operations. The 'hugging' mechanism used by the conveyor system also reduces contamination and dust generation.

Similar to the tracks of an army tank, apron conveyors link individual apron plates together by overlapping each other to form a chain, creating a conveyor with a trough like carrying surface that can hold materials, as seen in Figure 8. Apron conveyors are often used in the metallurgical industry, widely being used to transport heavy rocks from chutes to primary crushers. The heavy-duty construction can withstand high impact from falling ore, making it a sound choice to be used under stockpiles and equipment where the ore may have a long fall before landing on the conveyor. By using high-grade steel or heavy-duty materials for the plates, the conveyor can be used for abrasive materials.

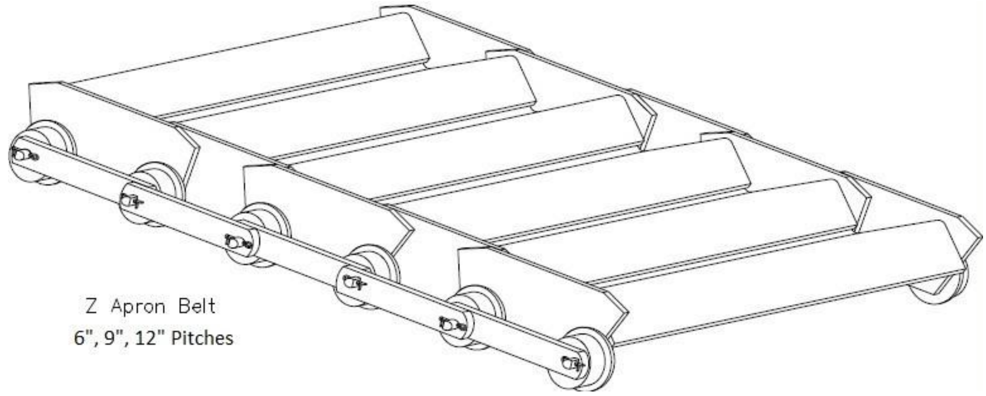

Figure 8. Apron conveyor system diagram.

Chain conveyor systems stray away from the conventional belt system, and instead employ a continuous chain arrangement that is driven by a motor, as seen in Figure 9. Predominantly used to transport heavy unit loads, chain conveyors have found use in underground mine operations in the form of armoured face conveyors (AFC). The AFC's main function is to remove cut coal form a long wall face after it is sheared by the long wall miner. The AFC has a large carrying capacity and is structurally very strong and low in body height. The system consists of the link chain which runs long ways with the conveyor set up, and the flight bars run perpendicular to the link chain and collect material. Both chains must be very strong and rigid as they are constantly exposed to heavy dynamic and static loading (Naruka, 2015).

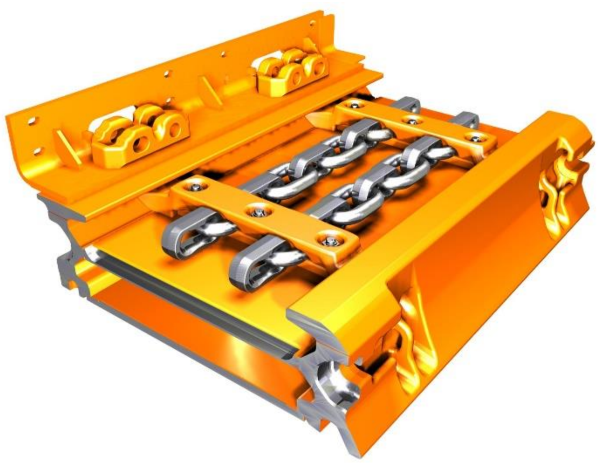

Figure 9. A single section of the armoured face conveyor system.

5.3.5. *Auxiliary Technology*

Although belt conveyors are generally employed to transport materials, they are often used in conjunction with auxiliary equipment. They can form the material transportation stage for mining methods such as IPCC or bucket wheel excavators, and offer a number of extra features that can contribute to the success of a mine (Sandvik, 2008). Accurate and continuous weighing can be undertaken on a conveyor belt to give live readings to mine managers. Automated monitoring systems that sense heat and noise within the conveyor system can be used to prevent imminent failures. Conveyor belt systems can be used as an effective blending procedure, whereby materials can be placed onto the belt at different positions, mixing up the feed. Conveyor feed can undergo magnetic separation by using an overhead magnetic force to separate the material (Alspaugh and Bailey, 2005).

5.4. IN-PIT CRUSHING AND CONVEYING

The In-Pit and Conveying (IPCC) system is a hybrid system incorporating a continuous haulage system, namely a conveyor, with a primary crusher that has been designed to fit into mobile frameworks and a continuous spreading system (Ko, 1993). Traditionally materials were loaded directly onto the conveyor belt without size reduction, however an in-pit crushing plant was introduced as a precondition for conveyance to overcome the conveyor limitations and ease the transportation.

The aim of the sizing system is to reduce the run of mine (ROM) to a size that is deemed reasonable to convey. Wyllie (1989) states that *the maximum size of material on the conveyor belt should not exceed approximately 300 mm, or a third of the belt width.* Conveying material that is too large in size can incur operational downtime. The crushing needs to be organised in such a way that it can handle both waste and ore, and manage throughputs that do not outweigh the crushing capacity, causing bottlenecks in the sizing process. As an excavator is only as productive as the trucks it is loading, an IPCC excavator can feed a continuously running crushing and conveying system, allowing the excavator to run at increased capacity.

The IPCC system has three main forms of crusher portability; fully mobile, semi mobile and fixed in-pit. Fully mobile IPCC systems use a crusher or sizer that is readily portable and follows the working face to be fed directly by loaders. This method eliminates the need for haul trucks. North Cement limestone quarry in Germany implemented the first fully mobile crusher in 1956, starting the trend for truckless IPCC systems in open pit mines (Ko, 1993). Modern day mobile crushers use tracks, similar to an army tank, and follows the shovel around the open pit. A portable conveyor attachment that can be connected to the mobile crusher forms the bridge from the crusher to the conveyor system. The mobility of the crushers restricts the machinery and crushing ability of the IPCC system, however technology advances has progressed fully mobile crushing throughputs to approximately 8000 t/h (see Figure 10).

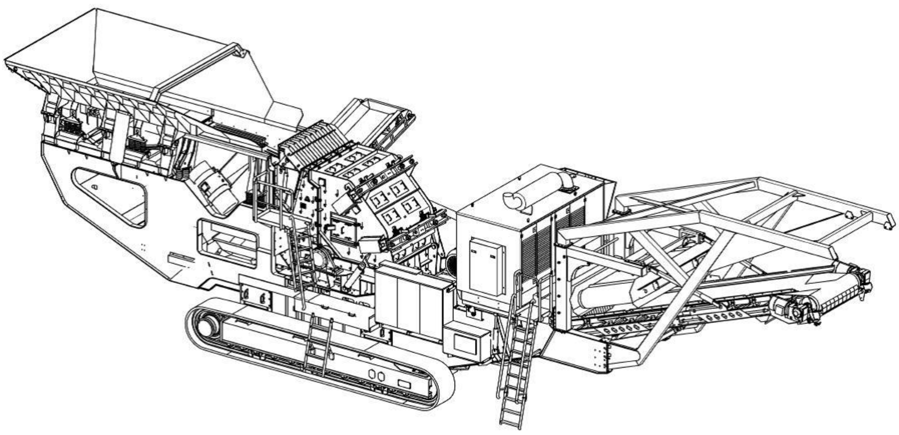

Figure 10. Fully mobile IPCC system diagram (Russell, 2015).

Semi mobile crushing uses both conventional TS operations and IPCC for material excavation and transportation. Instead of hauling the material out of the pit using trucks, the trucks dump the excavated material into semi mobile crushing systems, which feeds the material onto conveyors for onward haulage (Ko, 1993). The dump trucks travel in shorts hauls from excavator to crusher, minimising the truck related operational costs. This method combines the flexibility advantage of trucks and operating cost efficient material haulage of conveyor. Semi mobile crushers can facilitate larger capacities than fully mobile in-pit crushing units due to larger shape, for this reason the crushing plants are rarely relocated. Ground preparations for this method of crushing is minimal and foundations are not required due to the short periods of relocation. The crusher undergoes relocation once every 3 to 10 years, often using a transport crawler. This collaboration of continuous and discontinuous haulage has been found to be effective in open pit hard rock mines

Fixed IPCC systems do not move from where they are erected for the duration of the mine life. Mines can employ a singular fixed crusher, or they can implement a number of crushers depending on the mine size and productivity. The stationary crusher allows for larger, more productive crushing within the pit and hence higher ROM feed capacities, however the reduced flexibility of crusher positioning means the system is more haul truck intensive (see Figure 11). As discussed previously, higher dependency on haul trucks rather than conveyors reduces the operational efficiency of the material transportation. This method of crushing makes for difficult mine planning, as the mine has to revolve around the fixed crushing stations and mine expansion becomes difficult.

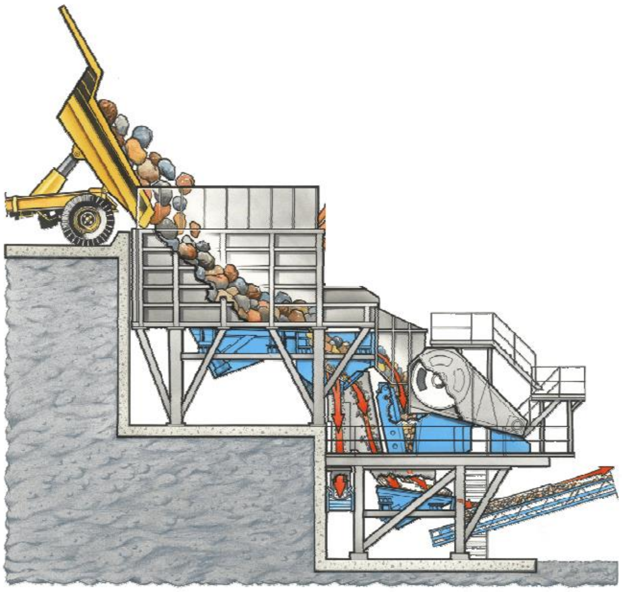

Figure 11. Fixed IPCC system diagram (Russell, 2015).

It is obvious that each IPCC type comes with a set of positives and negatives, and the correct choice is often dependent on the mine conditions. In 2008, Sandvik presented a summary of IPCC crushing options and their typical performances and requirements for each crusher type, this is shown in Table 5.

Table 5.
IPCC crushing options (Sandvik, 2008).

	Fully Mobile Crusher	*Semi Mobile Crusher*	*Fixed Crusher*
Throughput	<10,000 t/h	<12,000 t/h	<12,000 t/h
Truck Quantity	None	Low	Intermediate
Crusher Type	Sizers, jaw/double roll crushers	Any	Any
Unit Crushing Costs	Higher	Intermediate	Lower

The IPCC system offers a high availability, regardless of the systems intricate design. A single crusher can have an availability of up to 85%, accompanied with a conventionally conveyor system availability of approximately 95%, presenting an overall system availability of approximately 90% (Atchison & Morrison, 2011). However, utilisation of the IPCC system can be significantly lower as a result of system dependencies, going as low as 75%.

The final stage of the IPCC transportation system is the spreader stage, whereby large material moving equipment receive material from the conveyors and dump it in an orderly and efficient manner (Ko, 1993). Spreaders are used to spread the waste whilst stackers are used to stack the ore.

5.4.1. *IPCC Mine Planning Consideration*

IPCC implementation in open pit mines can cause significant headaches in the mine planning stage. One of the biggest challenges hindering the success of IPCC systems is the conveyor placement for out of pit transfer. The three main techniques of out of pit transportation for IPCC systems are; dedicated ramp, conveyor tunnel and conveyor on haul road.

The dedicated ramp, or conveying slot method, incorporates a ramp into the mine design that is exclusively used for conveying material. By providing a ramp which is designed entirely to provide an exit for conveyors, the angle can be raised up to the maximum slope angle of 18° for regular conveyors (Atchison & Morrison, 2011). Whilst this method can sometimes be costly and difficult to develop, it maximises the IPCC efficiency. The steep exit out of the mine allows for the minimum haulage distance from crusher to stockpile, reducing the haulage and operating costs. This method limits the final pit shape because the ramp cannot be moved in the typical pushback sequence, therefore the mine can only be expanded horizontally on sides that do not involve the conveying slot (Atchison & Morrison, 2011).

This pit exit method introduces new constraints into the production scheduling of the mine. As the direction of mining expands opposite to the conveyor ramp side, the production scheduling order is limited in optimisation. This is shown in Figure 12, with the resultant pit phasing in Figure 13. This incurs large waste production early in the mine life in order to achieve the required recovery.

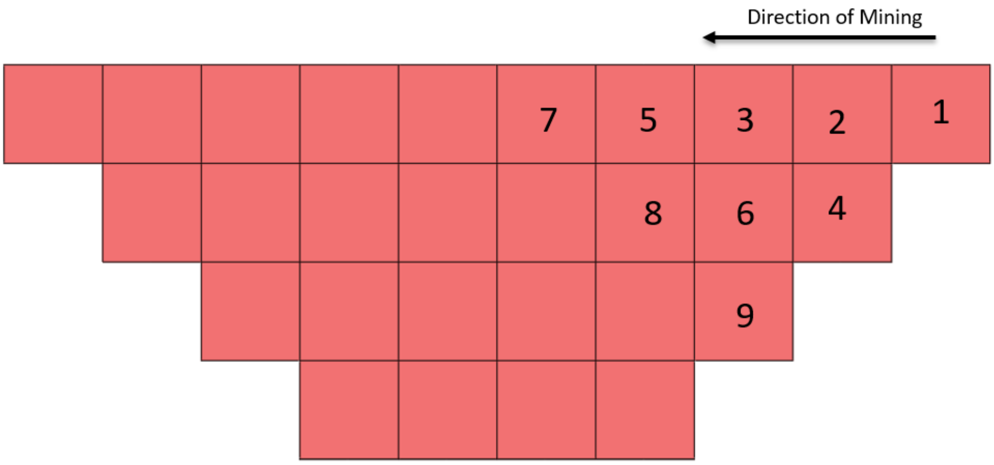

Figure 12. IPCC production scheduling example.

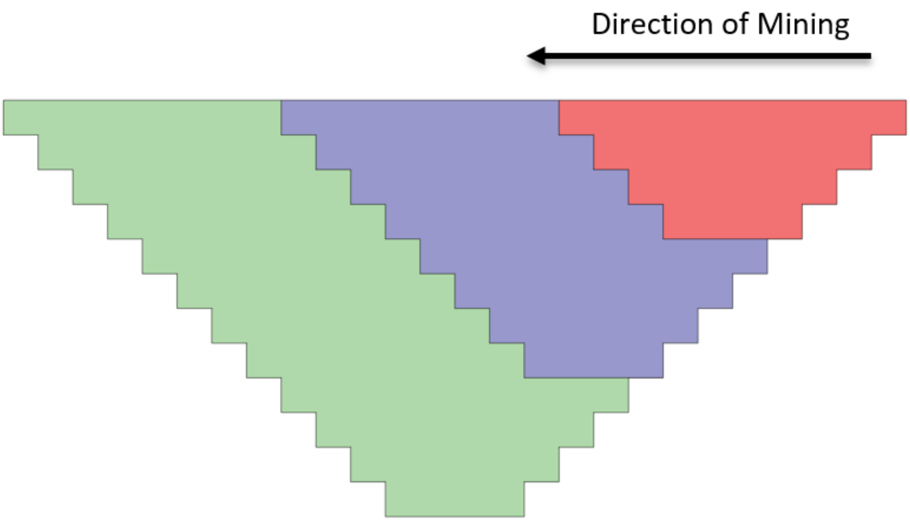

Figure 13. IPCC pit phasing.

If both trucks and an IPCC system are used in an open pit, such as semi or fixed IPCC systems, than the situation occurs where the conveyor system and haul road intersect (Mohammadi, Hashemi & Moosakazemi, 2011). To sustain traffic flow either a conveyor tunnel or a conveyor bridge can be implemented. A conveyor bridge is created by elevating the conveyor ramp to a height that is greater than a fully loaded truck allowing for vehicles to pass underneath, whilst conveyor tunnels premade concrete or plastic tunnels buried in the haul road to prevent the cross over. High angle conveyors, such as sandwich conveyors, can overcome awkward pit geometries caused by low angle belt conveyors but they are not developed for high capacity demands, i.e. >2500t/h (Oberrauner and Turnbull, 2013).

The conveyor tunnel is an exit strategy that involves creating an underground tunnel for conveyors to transport material to the surface. The conveyor tunnel allows for the material to be transported without consuming space within the pit shell and causes very few problems with truck-conveyor cross over. However, this method poses large geotechnical issues by creating a tunnel in an operating open pit, enormous capital costs and makes for difficult mine planning (Mohammadi, Hashemi and Moosakazemi, 2011).

The conveyor on haul road method is the most popular method, which is often used when open pit mines make the switch between TS and IPCC. The conveyor follows the haul road upon which the heavy vehicles use. As a result, material haulage distance increases significantly as it follows the same route and angle (<6°) as the truck haulage. Whilst this technique is simplistic and flexible, it is associated with increased capital costs due to the greater belt length. The conveyor corridor on the haul road requires an additional 8 m to the haul road width, resulting in increased waste production (Oberrauner and Turnbull, 2013).

When the top bench has been successfully pushed back, the hydraulic backhoe moves to the lower bench to mine, and the auxiliary equipment progressively drops to benches as well. The three-bench operation reduces bench widths in the open pit to 20 – 30 m, maintain high inter-ramp angles, and backhoe operation allows for increased operator visibility and reduced swing angles. However, this method requires conventional TS systems to construct the initial box cut suitable for ramp conveyor. The fully mobile IPCC system cannot fully replace TS operations, as sinking of the mine is undertaken using truck-shovel systems, while pit widening is accomplished using fully mobile IPCC systems.

5.4.2. *IPCC Case Studies*

Since the first IPCC systems were implemented in the late 1950's over 200 mobile and semi-mobile crushing systems have been installed worldwide (Koehler, 2003). IPCC systems have been employed in all types of weather conditions, such as the hot arid conditions in Western Australia (bauxite mines) to the tropical climate in Papua New Guinea (Bougainville copper mine) or even the snowy tundra of Canada (Highland Valley copper mine). The following are two examples of IPCC systems that have operated successfully.

5.4.2.1. Sierrita Copper Mine

Sierrita mine is an open pit copper mine in southern Arizona, USA. Since the beginning of operations in 1970, trucks were used at the method of haulage, however in 1982 two fixed in-pit crushers were constructed close to the pit perimeter accompanied by a conveying system to transport the waste to a waste dump (Ko, 1993). This IPCC system reduced the cycle times and haulage routes for haul trucks. More recently however, the two former ore crushers was replaced by a semi-portable gyratory crusher with a capacity of 3630 t/hr (Mohammadi, Hashemi and Moosakazemi, 2011). The sized material is transported to the waste dump using overland conveyors. Sierrita copper mine was the first open pit mine to utilise a semi-mobile IPCC system, and had a total capital cost of approximately $32 million (USD). The system allowed the fleet size to be reduced by 25%, providing the mine with an average mining cost saving of $0.32 a tonne (Mohammadi, Hashemi and Moosakazemi, 2011)

5.4.2.2. Island Copper Mine

Island copper mine is located near Port Hardy in British Columbia, Canada. In 1985, the mine commissioned a 5500 t/h semi mobile crusher to be used with a conveyor system as the prominent mining system in the open pit. The crusher works in conjunction with several truck fleets to produce 43,000 tonnes of copper ore per day. The trucks have an average travel distance of 1.2 km from the working face to the crusher, reducing the truck fleet from 25 to 14 trucks, resulting in a saving of $0.19/tonne and a payback period of 4 years (Mohammadi, Hashemi and Moosakazemi, 2011).

5.5. COMPARISON BETWEEN HAULAGE OPTIONS

The decision to which mining method should be implemented in an open pit mine has been debated in the air since IPCC systems where created. As every geological deposit in the world is unique and differs in-depth, grade, shape and geotechnical conditions, a clear and decisive answer on which mining method is better suited will never be possible. However, both approaches bring advantages tailored for certain deposits. A review on the advantages and disadvantages of IPCC systems in large-scale mining operations was conducted by Jeric and Hrebar in 1997. For large open pit mines, the benefits of IPCC mining over TS are summarised below:

- Continuous material haulage;

- Increased mine and road safety due to fewer moving vehicles on mine roads;
- Reduced road maintenance requirements (water trucks etc.);
- Reduced dust generation;
- Increased safety (reduced vehicle collisions);
- Electrical power (IPCC) is more cost effective than diesel; and
- Reduced emissions and noise generation (Jeric and Hrebar, 2008).

Their paper identified that whilst theoretically IPCC systems may be seen as the more beneficial method of material haulage at greater pit depths, it requires extensive mine planning and scheduling, deterring mine planners away from the advantages that IPCC can provide. Other detractors to IPCC implementation is the large upfront capital costs incurred due to the purchase and set up of crushers and conveyors. A shutdown of one belt can stop the entire production until revival, whereas a breakdown of one truck in an entire fleet will only slightly reduce the production of the mine. IPCC processes such as relocation of crushers and extension of conveyors can shut down the mining operation for a period of 2-3 days.

A comparative economic analysis of transportation systems in surface coal mines was conducted by Sevim and Sharma in 1991. By developing computerized design and cost models, they were able to evaluate typical U.S coal mine transportation systems such as TS, IPCC and coarse-coal slurry. The after tax costs (capital + operating) for each material transportation method for the 9 years that the theoretical coal mine was in operation was evaluated, and summarised in Table 6. As can be seen, the surface conveyor presents the lowest after tax costs. Whilst this paper provides insight into the material haulage in open pit coal mines, it does not delve into IPCC.

Table 6.
Surface haulage costs for modelled base case (in thousands of 1990 USD) (Sevim & Sharma, 1991).

Year	*1*	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9*
Surface Conveyor	763	762	761	760	759	758	3388	849	850
Coarse-coal slurry	1033	1290	1051	1061	4127	1278	2808	1297	1306
TS	1219	2414	1366	1366	1366	1366	5366	2561	2513

In 2003, Koehler showed that whilst IPCC systems present the benefit of much lower operating costs, the system represents a significant up-front capital investment to which the mine is only able to depreciate over a sufficient mine life (Koehler, 2003). His results showed that for short hauls, TS operational costs were approximately double the IPCC operational costs, and for long hauls TS tripled the operational costs for IPCC. This cumulative cost of both mining methods is illustrated in Figure 14.

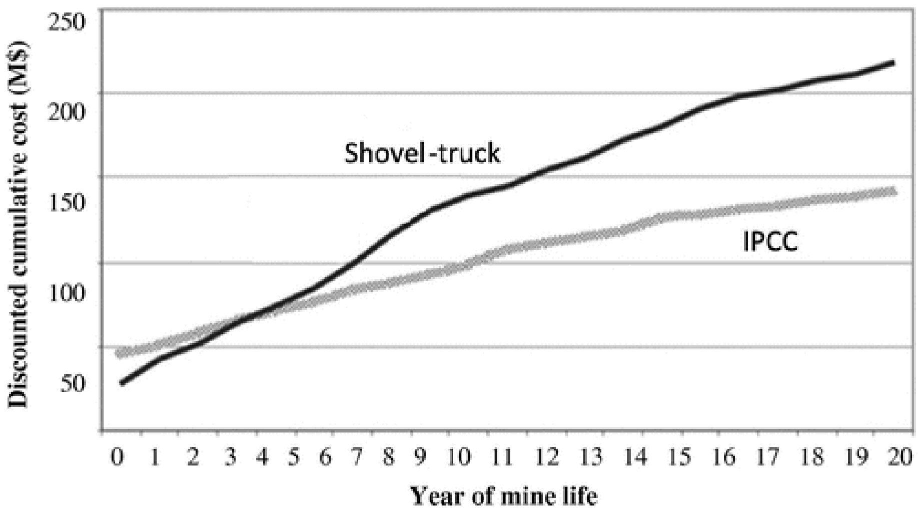

Figure 14. Cumulative cost of TS operations versus IPCC system.

Turnbull and Cooper (2009) evaluated IPCC as a partial or full substitute to TS methods. In 13 of the 15 studies they analysed, they found that IPCC generated operating savings ranging from US$0.18 to US$0.82 per tonne of material moved. By evaluating the various case studies, Turnbull and Cooper believed that if truck cycle times are greater than 25 minutes than IPCC systems are more economical. These 15 case studies are spread over both open pit hard rock mines and open cut coal mines between 2008 and 2009. They also noted that IPCC is ideally suited for new operations, or expansion of existing operations.

A comprehensive comparative study between valuable operating time of both TS operations and IPCC systems was undertaken by Dzakpata *et al* in 2016. The authors measured the performance of both systems on their respective utilised time, operating time and valuable operating time. The findings of the report concluded that although trucks have greater flexibility and lower capital costs, IPCC offers a better measure of performance on all three factors of equipment performance. Paricheh, Osanloo and Rahmanpour (2016) discussed the optimum time and location for applying a semi-mobile IPCC system into an already working TS mine using a mathematic programming approach. Their conclusion was that TS methods should be replaced with IPCC at approximately 490 metres deep into the pit, and 17 years into the mine life. The switch in mining systems only produced a 1% NPV improvement for the project.

6. METHODOLOGY

The methodology for this project followed conventional mine planning stages, including:

- Deposit & Block Model;
- Design optimisation;
- Pit design;
- Design scheduling; and
- Cost estimation.

6.1. DEPOSIT

A copper-gold block model was developed and used as the deposit for all mine scenarios. An oval shape mine plan is best suited for the tabular shape of the deposit, rather than the cone mine design for conventional cylindrical deposit shapes. This allows for longer pit walls, ideal for a designated ramp pit exit strategy. The shallow porphyry copper deposit has a relatively high copper/gold grade, with increasing grade with depth. The pit is shown in Figures 15 and 16. Table 7 summarises the block model characteristics.

Table 7.
Project block model characteristics.

Parameter	*Value*
Number of blocks (total)	322 877
Number of blocks (ore)	60 247
Max. Cu grade	1.76 %
Max. Au grade	2.70 g/t
Depth to top of deposit	5 m
Depth to bottom of deposit	770 m

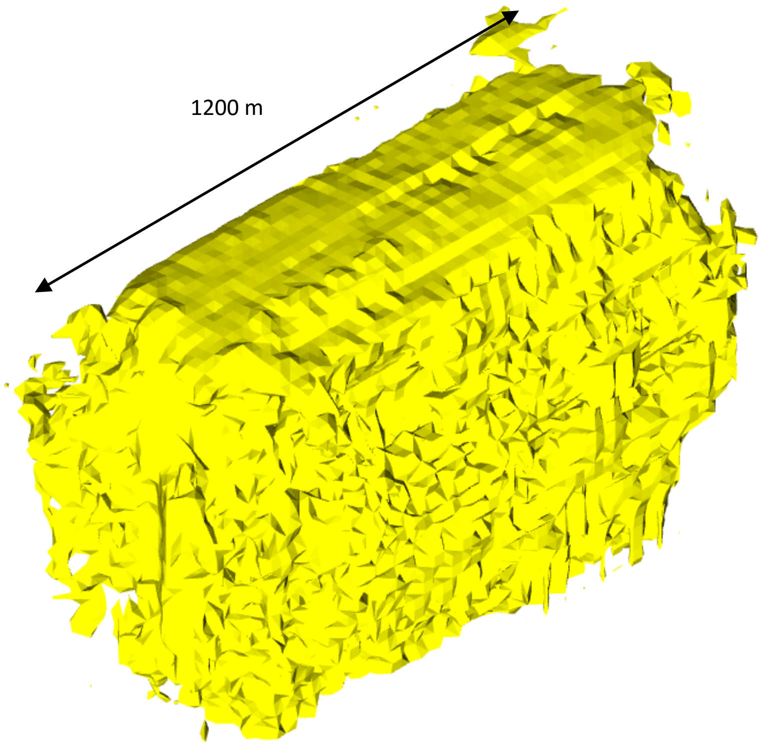

Figure 15. Isometric view of block model.

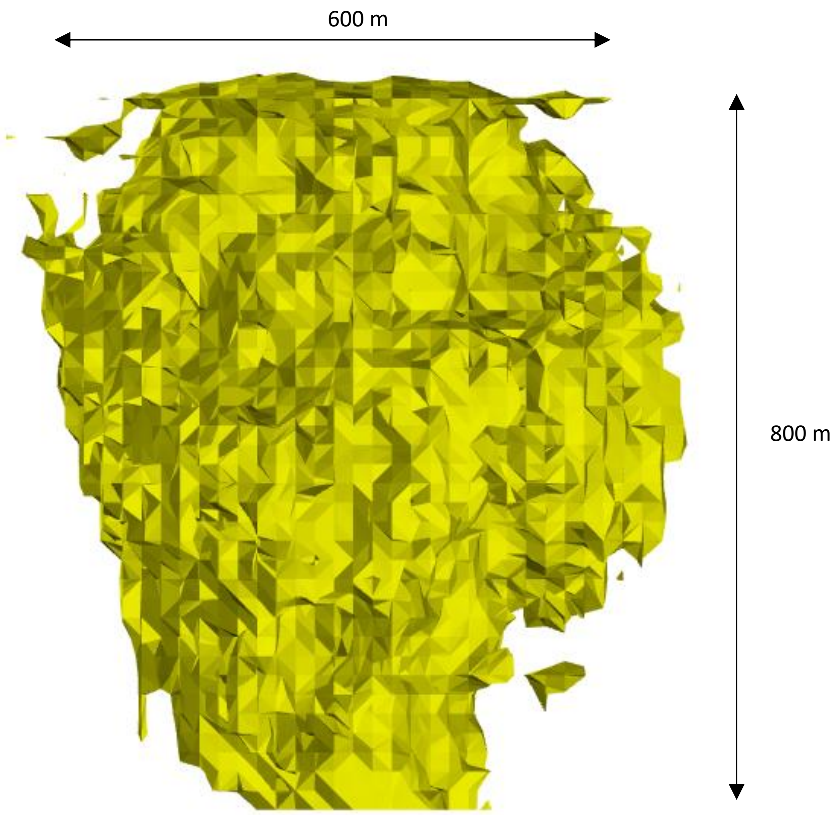

Figure 16. Cross sectional view of block model.

6.1.1. *Cut-Off Grade*

A grade-tonnage curve was graphed for the deposit, and is visible in Figure 17. The cut-off grade (COG) for both systems was determined, using mining and processing costs attained from literature, shown in Table 8. These mining costs are purely used to obtain a COG estimation, and will not be used further in the investigation. The copper COG was found to be 0.31% Cu for the TS system and 0.29% Cu for IPCC. This resulted in a resource of 322 Mt and 360 Mt for both the TS and IPCC systems respectively.

Table 8.
COG Resource Estimation.

System	*Mining Cost ($)*	*Processing Cost ($)*	*Copper Price ($)*	*Selling Cost ($)*	*Recovery (%)*	*COG (%)*	*Resource (Mt)*
TS	4	14	7000	500	90	0.31	322
IPCC	3	14	7000	500	90	0.29	360

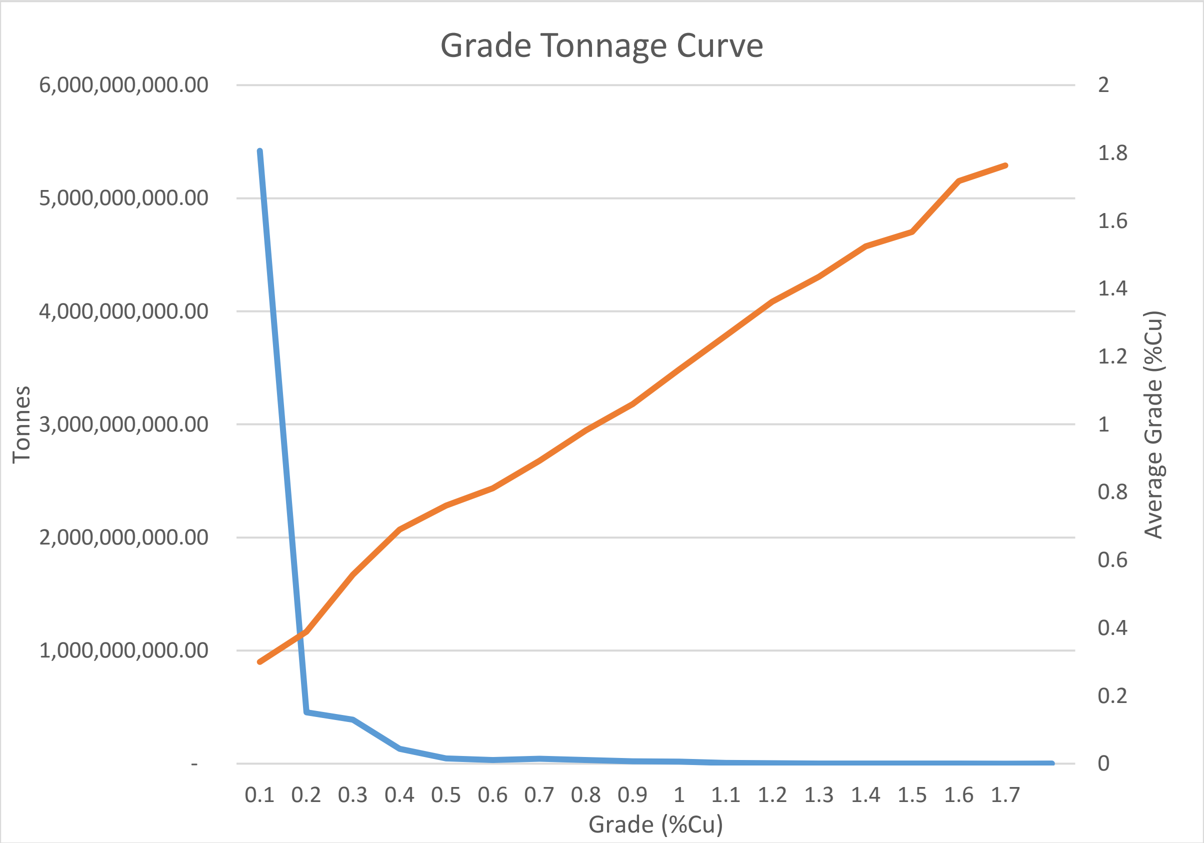

Figure 17. Grade Tonnage Curve.

6.2. PIT OPTIMISATION

GEOVIA Whittle software was used to determine the optimum mine plans, including pushback sequence, pit depth and plan size that maximises profitability based on predetermined mining constraints (Dassault systemes, 2009). The final pit shape and pit parameters were based on the following:

- Exploiting the most economically viable reserve within the lease boundary;
- Arrangement of grade values in the resource block model;
- Full utilisation of capital; and
- Maximising NPV.

However, *GEOVIA Whittle* is limited to truck and shovel mining operations only, and cannot provide pushback sequences for IPCC systems. Therefore, the IPCC pit design can only be based on the optimum pit depth that *GEOVIA Whittle* provides, rather than the pushback sequence.

6.2.1. *Mining/Processing Capacities*

A mining capacity of 25Mtpa was used for both TS and IPCC operations, under the assumption that a single Hitachi EX 5500-6 loader will be used, resulting in approximately 25 Mtpa of production. A processing capacity of 4 Mtpa was set for the initial year of production, ramping up to 5 Mtpa in the second year.

6.2.2. *Mining and Processing Cost*

To establish suitable mining and processing costs, several feasibility studies and current operations were evaluated. A mining cost of \$3/t was used for IPCC system and \$4/t for TS operation. A processing cost of \$14/t was assumed, based on values for similar mines that utilise heap leaching and flotation for the oxidised and sulphide domains (CostMine, 2016).

6.2.3. *Commodity Selling Costs*

It was assumed that this mine would be situated in Western Queensland, near Mt Isa. The selling cost used in Whittle was based on those of surrounding mine sites, which range between \$400/t to \$500/t. Therefore a copper selling price of \$500/t was used.

6.2.4. *Commodity Prices*

The commodity prices were determined from historical value graphs, where a reasonable judgement on the future prices of the metals was made. Figure 18 below shows the plot, were a value of $7000/t of Copper was chosen.

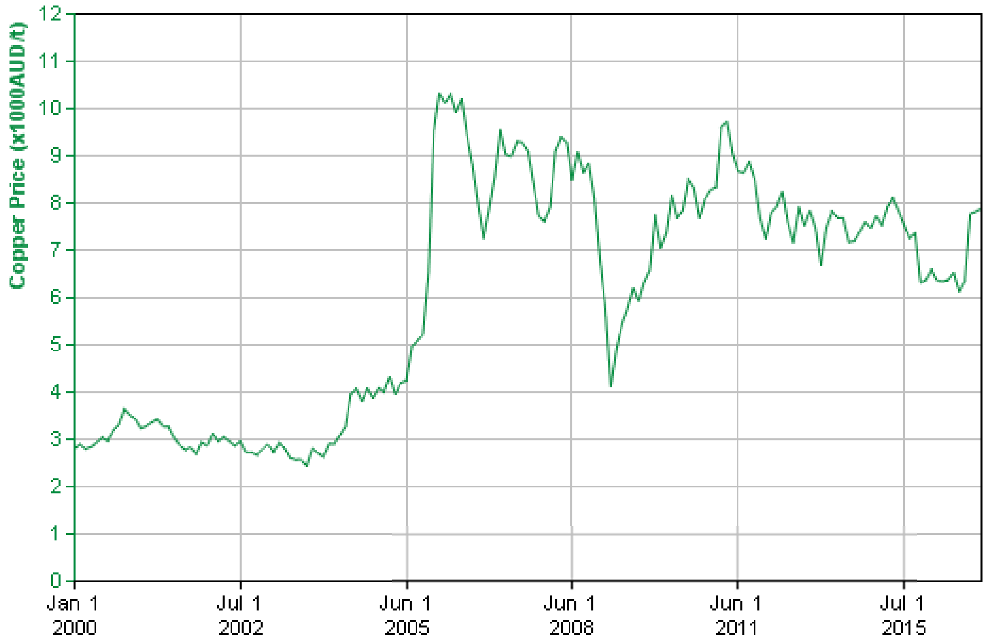

Figure 18. Commodity price graph (Cu).

6.2.5. *Metal Recoveries*

In determining the recovery rates, it was assumed that a conventional flotation process route would be the primary processing method, hence the recovery rates for this method were investigated. Recovery rates for similar processing plants ranged from 69% to 94%. Based on these plants, the values used in the whittle optimisation were the following:

- Recovery of Copper: 90%; and
- Recovery of Gold: 75%;

6.3. Pit Design

The pit shells and mine plans were exported from *GEOVIA Whittle* and input into *Deswik CAD* software to commence the pit design. *Deswik CAD* is a CAD engine that is used for open cut and underground mine designs. The block model was input into the software, and all constraints and necessary parameters were set.

6.3.1. *TS Design*

The TS mine design was far more simplistic than the IPCC design. The pit shells (shown in Figure 19) were used as an outline for the pit shape and the applying the design parameters summarised in Table 9 an optimum pit shape was established.

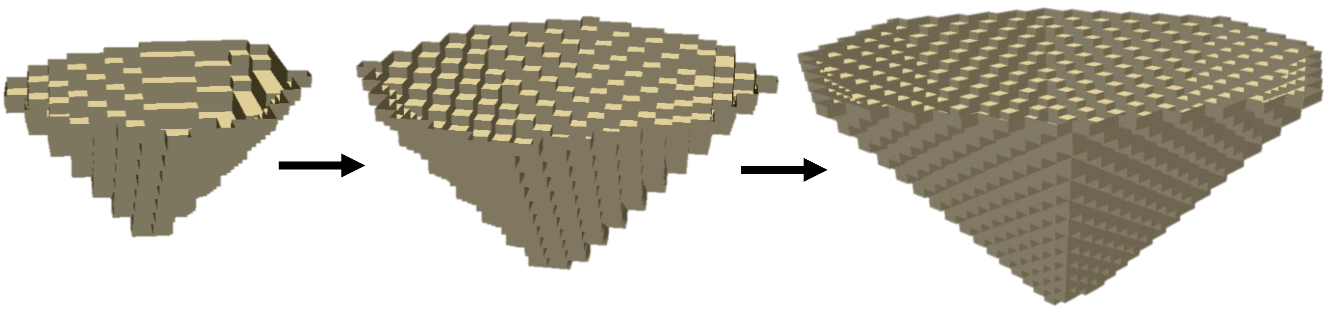

Figure 19. TS pit shells.

Table 9.
TS pit characteristics.

Description	*Value*
Bench Height	15 m
Face Angle	60°
Overall Angle	50°
Haul road width	25 m
Berm width	4 m
Pit Depth	525 m

The bench height was based off the maximum reach of the mining shovel, approximately 15 m in height. A 4 m berm width was applied to allow for sufficient room to catch falling material or failed wedge material from overhead benches whilst minimising waste (Mine-Engineer, 2007). Haul road width was designed to be 25 m to accommodate for the widest vehicle in use, which is the CAT 797. A slope angle of 60° was applied, resulting in an overall angle of approximately 50°. The optimum shape pit strings are visible in Figure 20 and Figure 21, and the pit solid is shown in Figure 22.

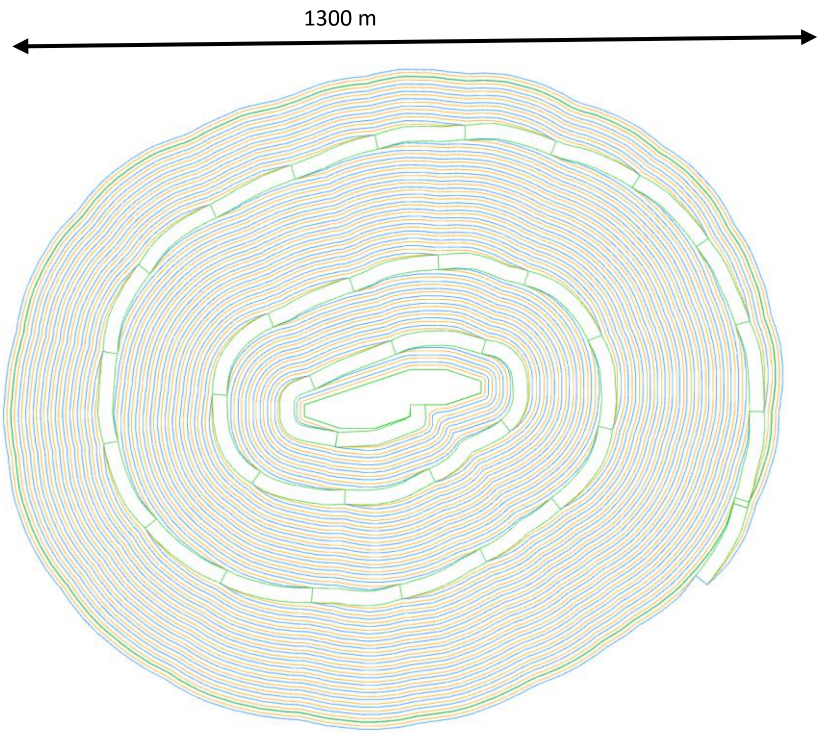

Figure 20. TS optimum pit shape plan.

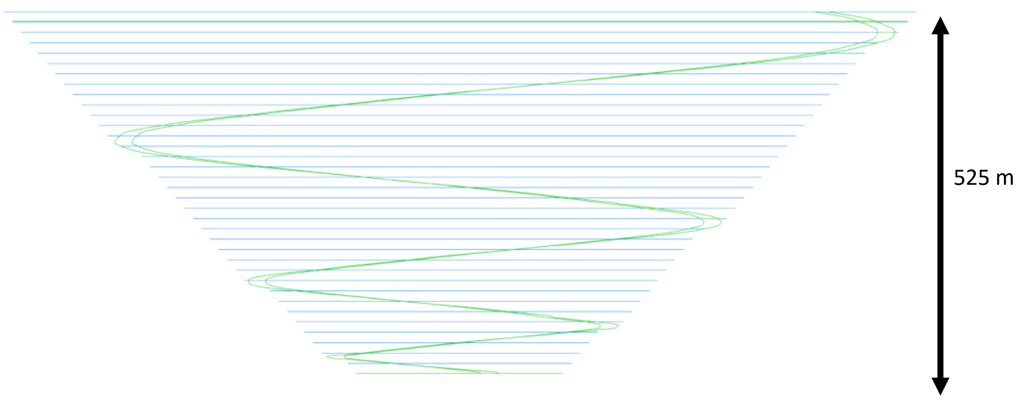

Figure 21. TS optimum pit side view.

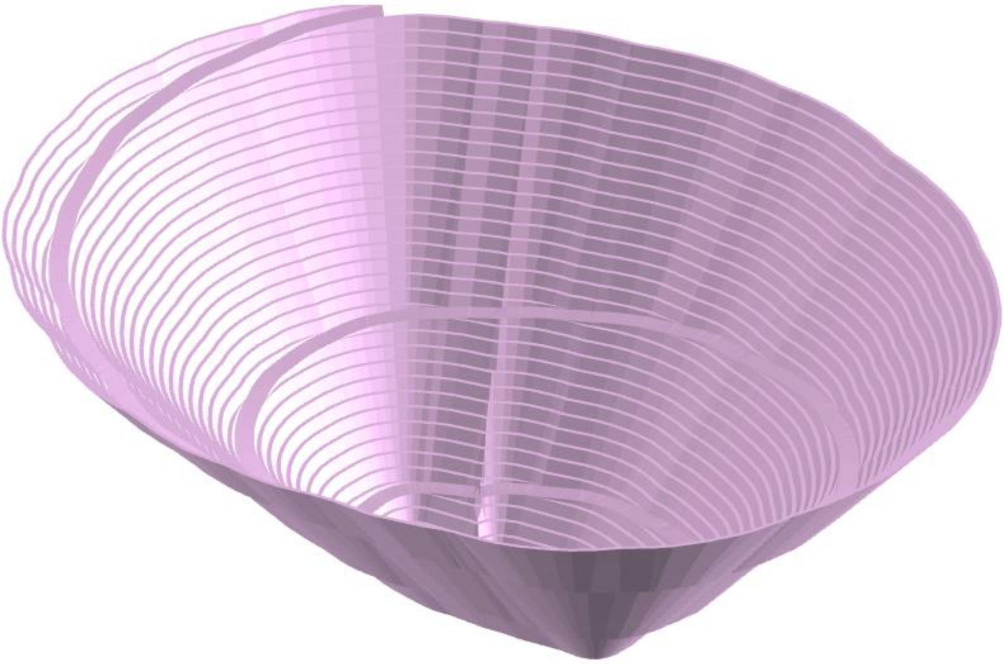

Figure 22. Isometric TS pit solid.

The optimum TS mine resulted in a probable mineable reserve of 190 Mt, achieving a 60% recovery, at a stripping ratio of 1 to 2.2. The resultant grade was determined to be 0.516%. The constant production rate of 25 Mtpa allowed for a 25 year life of mine, resulting in a total waste output of 426 Mt. The pit characteristics are summarised in Table 10.

Table 10.
TS pit values.

Description	*Result*
Total Material	620 Mt
Total Ore Tonnage	190 Mt
Total Waste Moved	426 Mt
Average Ore Grade	0.55% Cu
Recovery	60%
Life of Mine	25 Years

6.3.2. *IPCC Pit*

6.3.2.3. Overview

The IPCC pit required a more extensive pit design than the conventional TS method. The most critical design consideration was the pit exit strategy. The three pit exit strategies discussed in the background section were assessed to identify which was the most effective pit exit strategy for the given deposit. The tunnel method was deemed unsuitable for the deposit as it would require large working CAPEX. Additionally, the tunnel hinders the pushback sequence and limits nearby blasting operations. Whilst the haul road method would seem the most simplistic method, it often requires a greater CAPEX and OPEX than the designated ramp method due to a longer conveyor route. The haul road is limited to the gradient of the trucks (8-10%) rather than utilising the maximum slope possible for conveyor belts (33%), hence lengthening the conveyor and increasing power demand. Combining trucks and conveyors on the same haul road often expands the pit shape as the turning radius of conveyors far exceeds those of haul trucks, requiring a vastly wide pit to accommodate for the turning conveyors.

The designated ramp method utilises the shortest distance of conveyor belts to exit the pit, however it is often associated with a larger waste production. The designated ramp method was deemed the most appropriate pit exit strategy and was implemented into the design. This method introduces a number of constraints into the pit design process. The conveyor ramp method can rise at a maximum angle of 18° before fall back occurs on the conveyor, and was designed to be 5 metres in width to accommodate for the conveyor belt and allow ample room for maintenance. Minimising the angle the ramp makes with the flat side of the pit reduces the chance of the designated ramp leaving the extents of the pit and creating a conveyor slot, thus minimising waste production. This angle was determined by calculating the plan distance required to maintain a slope angle of 18° for a single bench, shown in Figure 23, and applying it to the pit bench design. A plan view of the pit bench design for three benches with the designated conveyor ramp implemented is shown in Figure 24. The bench face is highlighted yellow and the berm is in blue. The minimum angle was found to be 11° to the horizontal.

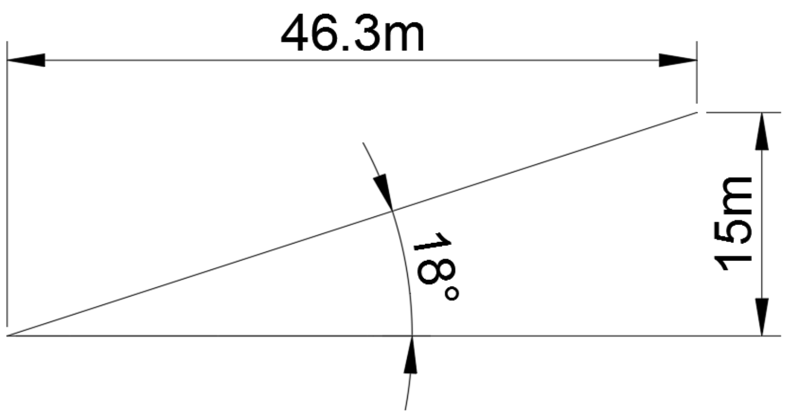

Figure 23. Designated ramp bench cross section.

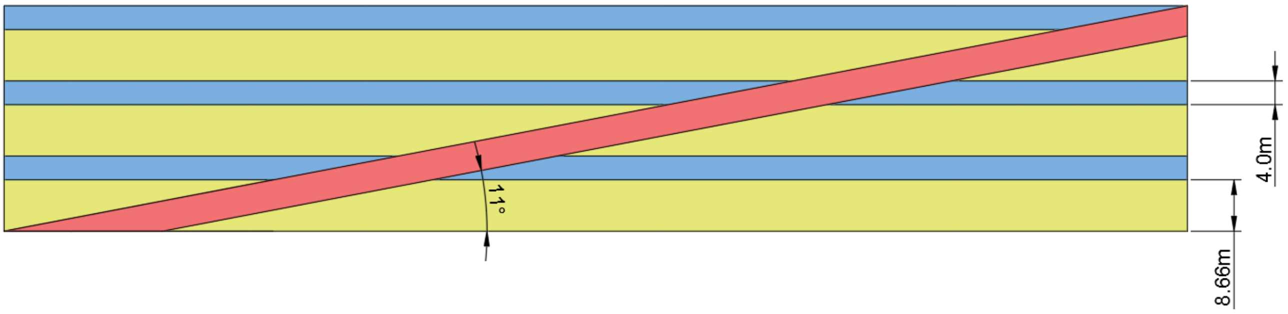

Figure 24. Pit wall plan view.

The optimum pit depth provided by *GEOVIA Whittle* was 540 m, and the optimum base length is 900 m. Applying a conveyor ramp into this design requires the ramp to cut out of the pit shape, this is called a conveyor slot or cut out slot, as shown in Figure 25. This conveyor slot requires large waste production early on in the mine life and should often be avoided if possible. A second IPCC pit depth was investigated to allow for a pit with no conveyor slot. This depth was determined to be 390 m, and is the maximum pit depth that contains the ramp within the regular pit extents, the cross section can be seen in Figure 26.

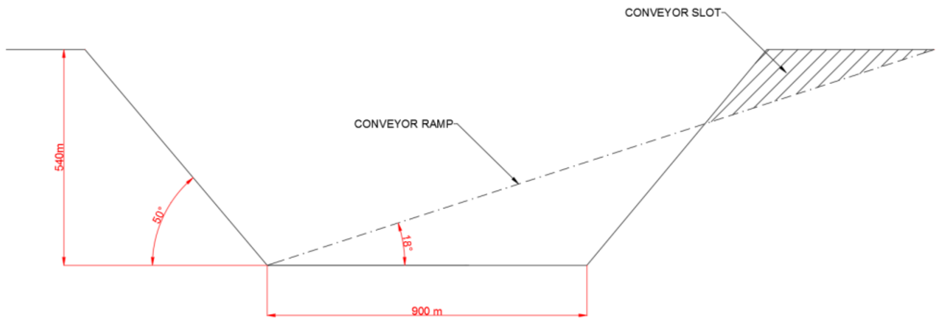

Figure 25. 540m pit cross section.

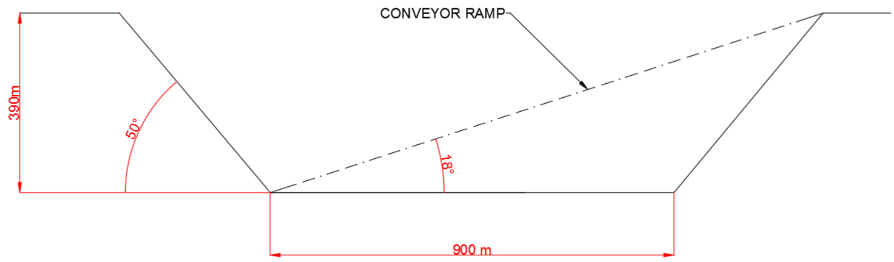

Figure 26. 390m pit cross section.

A pit was designed for each depth, labelled 36B and 26B for the deeper and shallower pits respectively. The 36B pit is expected to achieve a greater recovery, but have a significantly higher waste production than the 26B pit.

Another constraint the designated ramp method introduces is the haul road/ramp interaction. If geotechnical conditions permit it, the optimal pit haul road surrounds the pit in a spiral fashion, as exercised in the TS pit. By minimising the number of switchbacks in the pit design, the haul truck cycle time is minimised (Mine-engineer, 2007). However, if implemented into the IPCC pit, this spiral design will intersect the conveyor ramp. As discussed earlier there are numerous techniques to allow for traffic to intersect each other, such as a tunnel and conveyor bridge, however these systems are complicated and expensive to utilise (Turnbull and Cooper, 2009). Therefore, the most effective haul road method is to use numerous switchbacks to allow for the haul road to zig-zag down the side of the pit which does not expand. Figures 27 and 28 show the first phase for the IPCC 36B mine. As can be seen, the haul road zig-zags down the side of the pit that will not expand, continuing this trend until approximately 300 m deep, where it begins to move round the other side of the pit. The conveyor slot is also visible. Both pit designs used the same haul road, berm width, slope and bench height dimensions as the TS pit, summarised in Table 9.

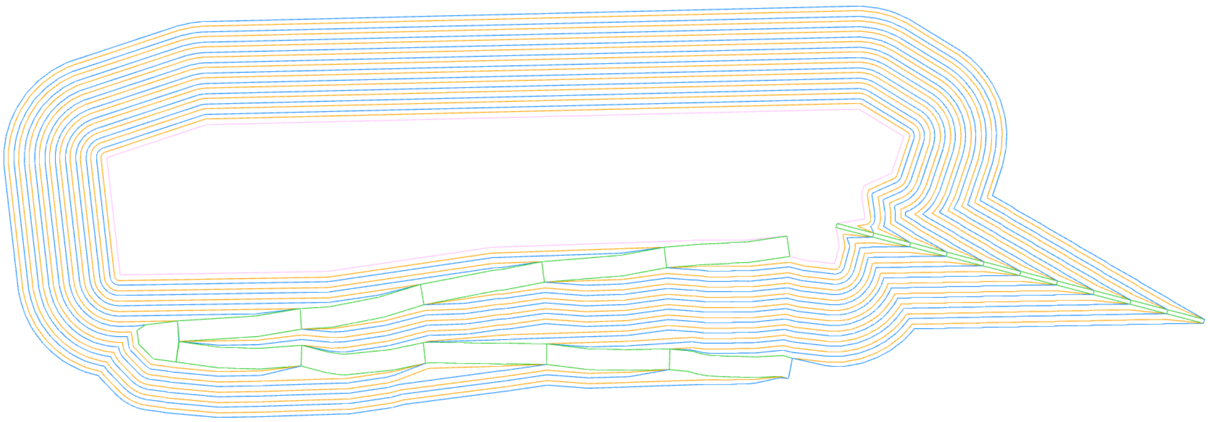

Figure 27. IPCC 36B plan view.

6.3.2.4. IPCC 36B

The IPCC 36B pit design resulted in 36 benches, resulting in a 540 m deep pit. Figures 29, 30 and 31 show the progression of the pit, showing all four pit phases. As can be seen the haul road remains on the conveyor ramp side of the pit until approximately 300 metres deep where it wraps around to the other side. Figures 32 and 33 show the final pit design strings.

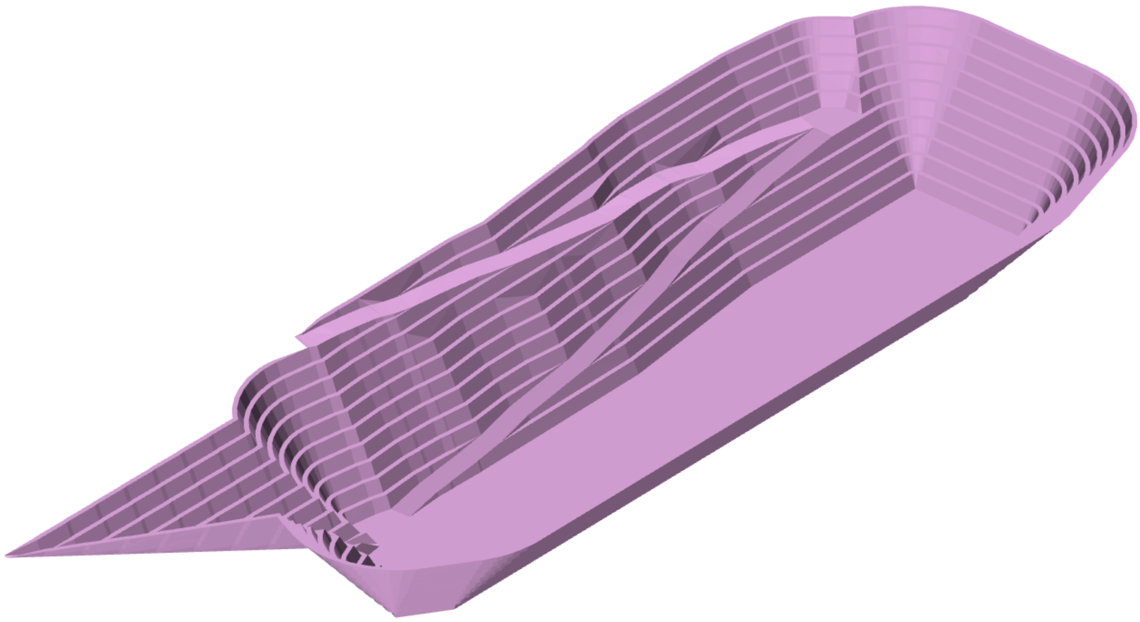

Figure 28. IPCC 36B Phase 1.

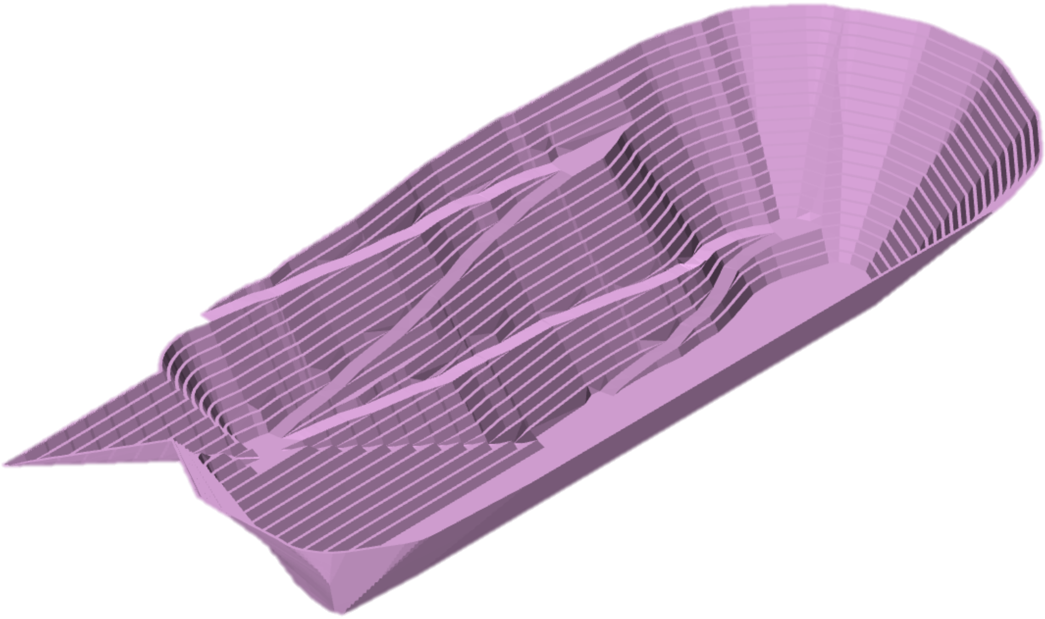

Figure 29. IPCC 36B Phase 2.

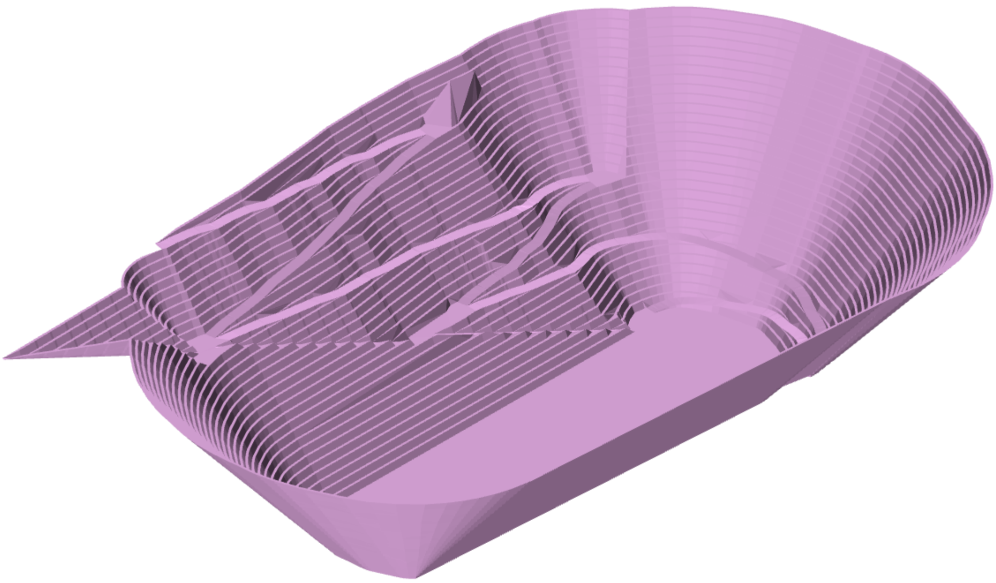

Figure 30. IPCC 36B Phase 3.

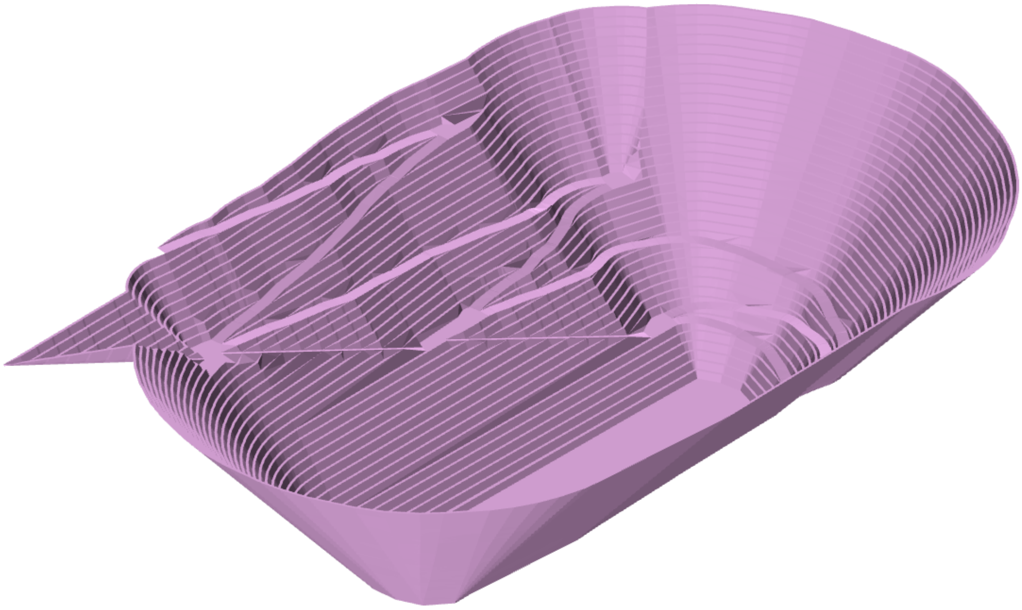

Figure 31. IPCC 36B Final pit shape.

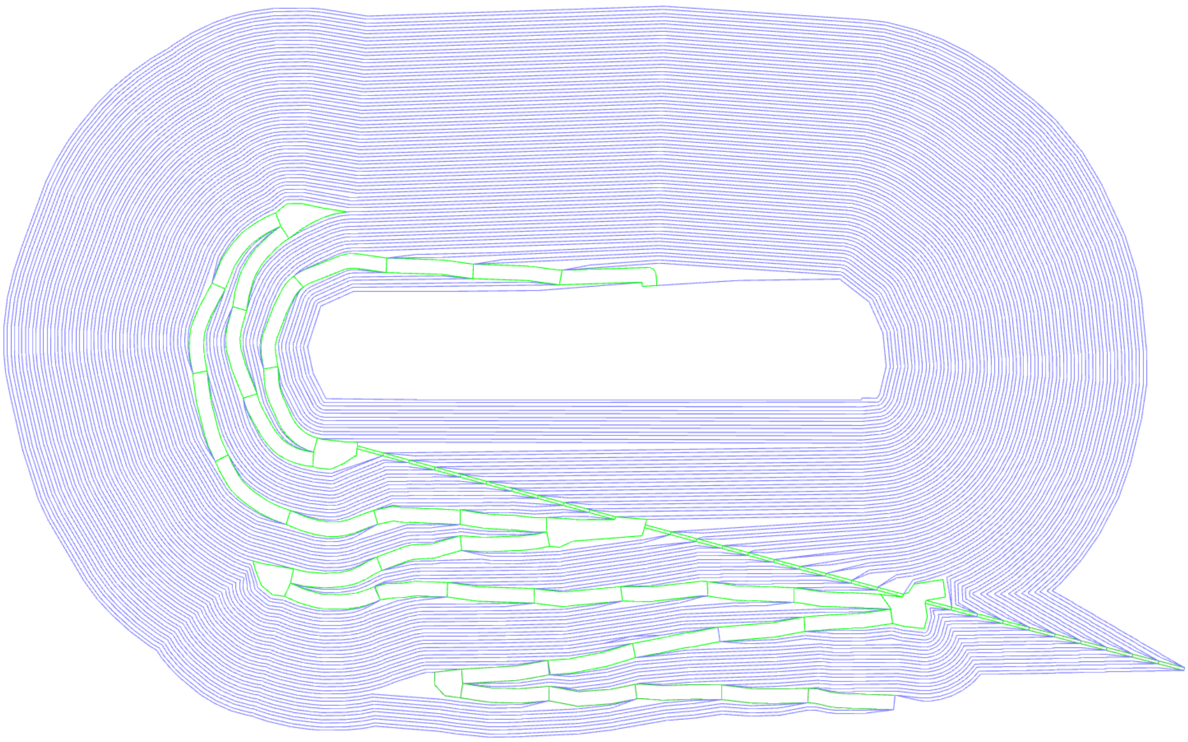

Figure 32. IPCC 36B pit strings.

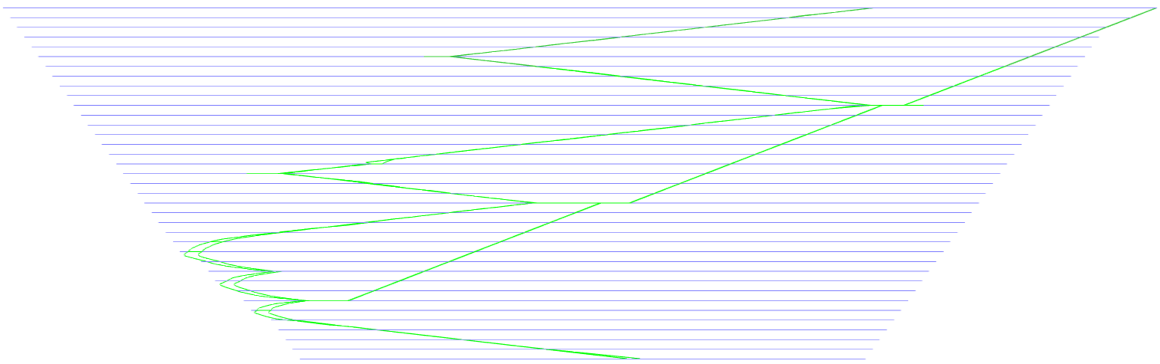

Figure 33. IPCC 36B side view.

The optimum IPCC 36B pit resulted in a probable mineable reserve of 253 Mt, achieving a 70% recovery, at a stripping ratio of 1 to 5.3. The resultant grade was determined to be 0.55% Cu. The constant production rate of 25 Mtpa allowed for a 45 year life of mine, resulting in a total waste output of 1080 Mt, as shown in Figure 34. The pit characteristics are summarised in Table 11.

Table 11.
IPCC 36B pit values.

Description	*Result*
Total Material	1335.6 Mt
Total Ore Tonnage	253 Mt
Total Waste Moved	1080 Mt
Average Ore Grade	0.55% Cu
Recovery	70%
Life of Mine	45 Years

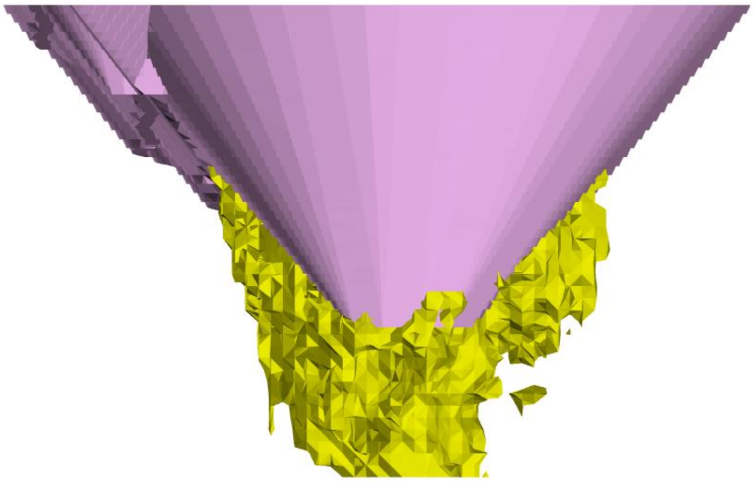

Figure 34. IPCC 36B pit resource.

6.3.2.5. IPCC 26B

The IPCC 26B pit design incorporated 26 benches, resulting in a 390 m deep pit. The pit design is very similar to the IPCC 36B, however does not have the conveyor slot as it is 150 m shallower than IPCC 36B. Figures 35, 36 and 37 show the progression of the pit, showing all three pit phases. Figures 38 and 39 show the final pit design strings.

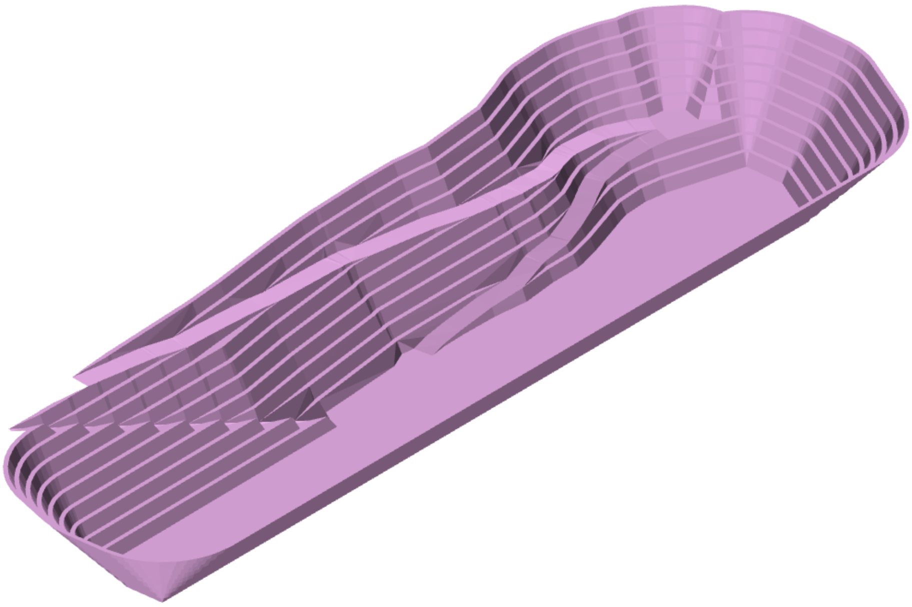

Figure 35. IPCC 26B phase 1.

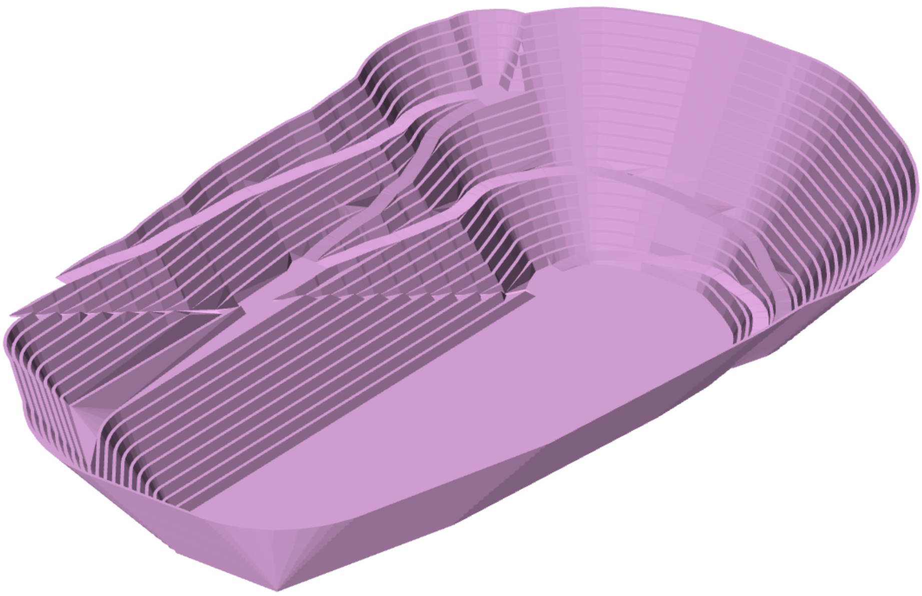

Figure 36. IPCC 26B phase 2.

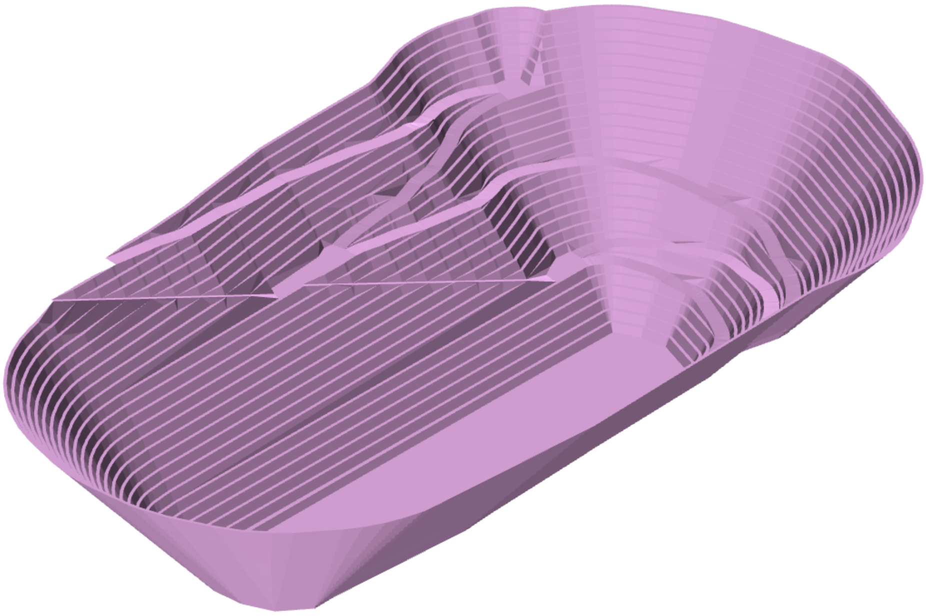

Figure 37. 26B final pit shape.

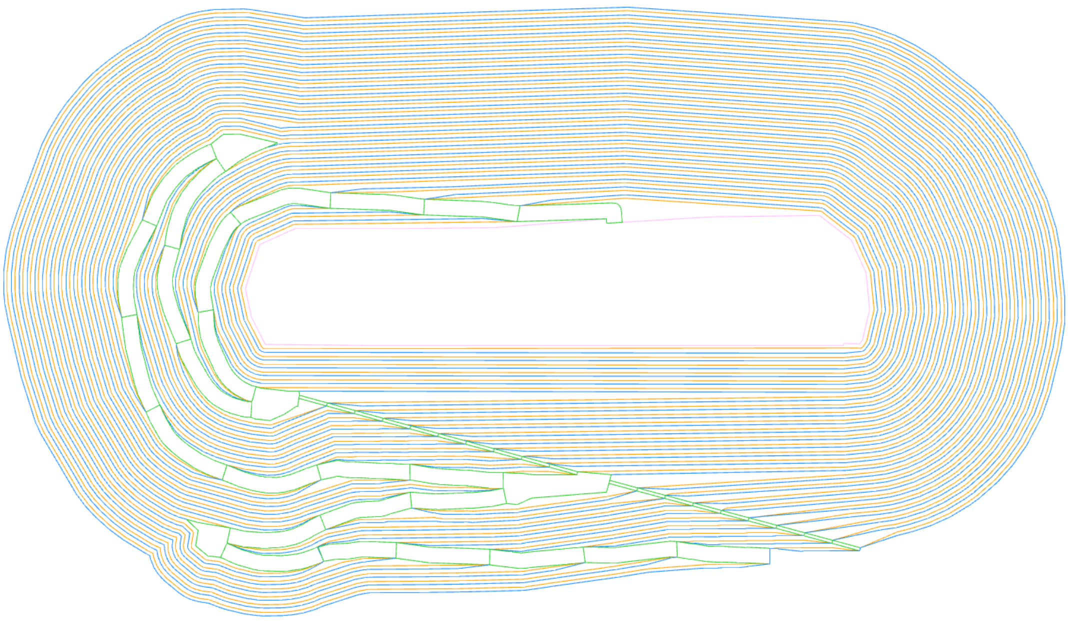

Figure 38. 26B final pit shape.

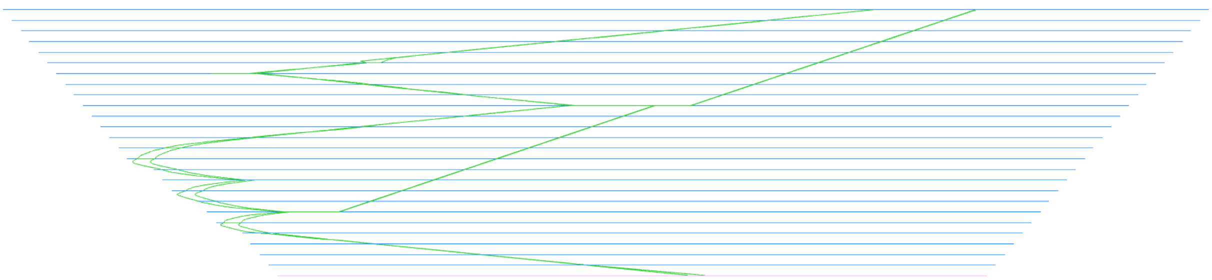

Figure 39. 26B side view.

The optimum IPCC 26B pit resulted in a probable mineable reserve of 175 Mt, achieving a 50% recovery, at a stripping ratio of 1 to 3.8. The resultant grade was determined to be 0.53% Cu. The constant production rate of 25 Mtpa allowed for a 26 year life of mine, resulting in a total waste output of 175 Mt, see Figure 40. The pit characteristics are summarised in Table 12.

Table 12.
IPCC 26B pit values.

Description	*Result*
Total Material	665 Mt
Total Ore Tonnage	175 Mt
Total Waste Moved	489 Mt
Average Ore Grade	0.53% Cu
Recovery	55%
Life of Mine	26 Years

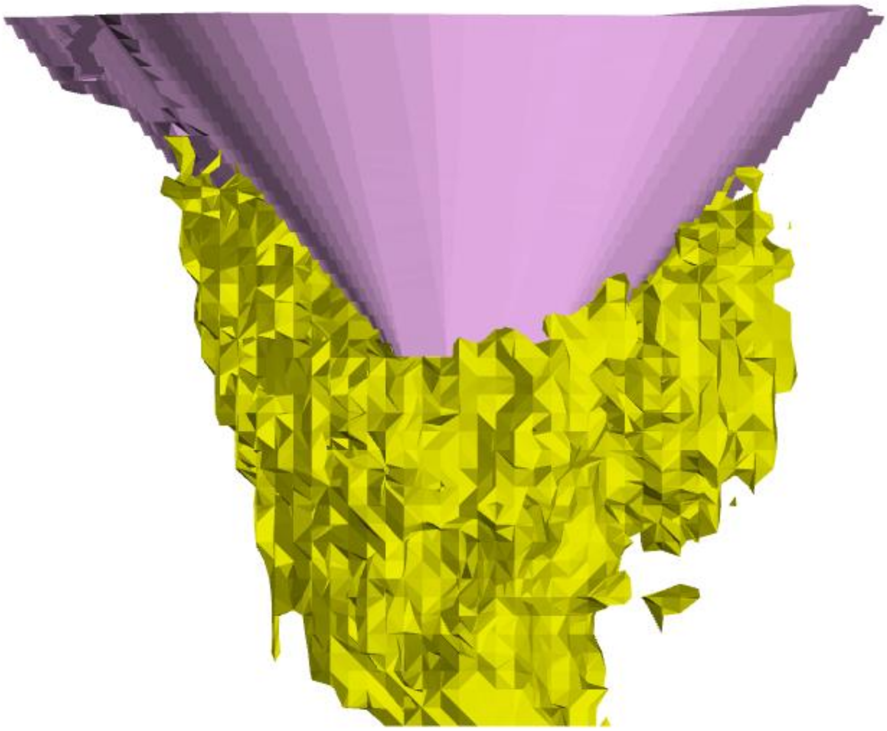

Figure 40. IPCC 36B pit resource.

6.4. PRODUCTION SCHEDULE

Production scheduling aims to *define the most profitable extraction sequence of the mineralized material from the ground that produces maximum possible discounted profit while satisfying a set of physical and operational constraints* (Niemann-delius & Khan, 2014). The production scheduling process for all three mines was done using *Deswik*'s interactive scheduler engine. As stated earlier, the production schedule sequence differs for IPCC and TS systems due to the set of constraints introduced in IPCC operations. Whilst typical scheduling methods such as Lerchs-Grossman can be used for conventional TS pits, they have to be altered for IPCC operations in order to satisfy the direction of mining. The optimum mine planning schedules were developed for the TS, IPCC 26B and IPCC 36B pits.

6.4.1. *TS Pit*

The TS pit production scheduling set by *Deswik*'s interactive scheduler followed a set of rules similar to 3D Lerchs-Grossman, allowing it to extract the deposit in the most profitable way. Figure 41 displays the scheduled tonnage profile for both waste and ore for the TS pit. It can be observed that large ore extraction occurs early in the mine life, and a majority of the ore mined in the middle years (years 10 – 17).

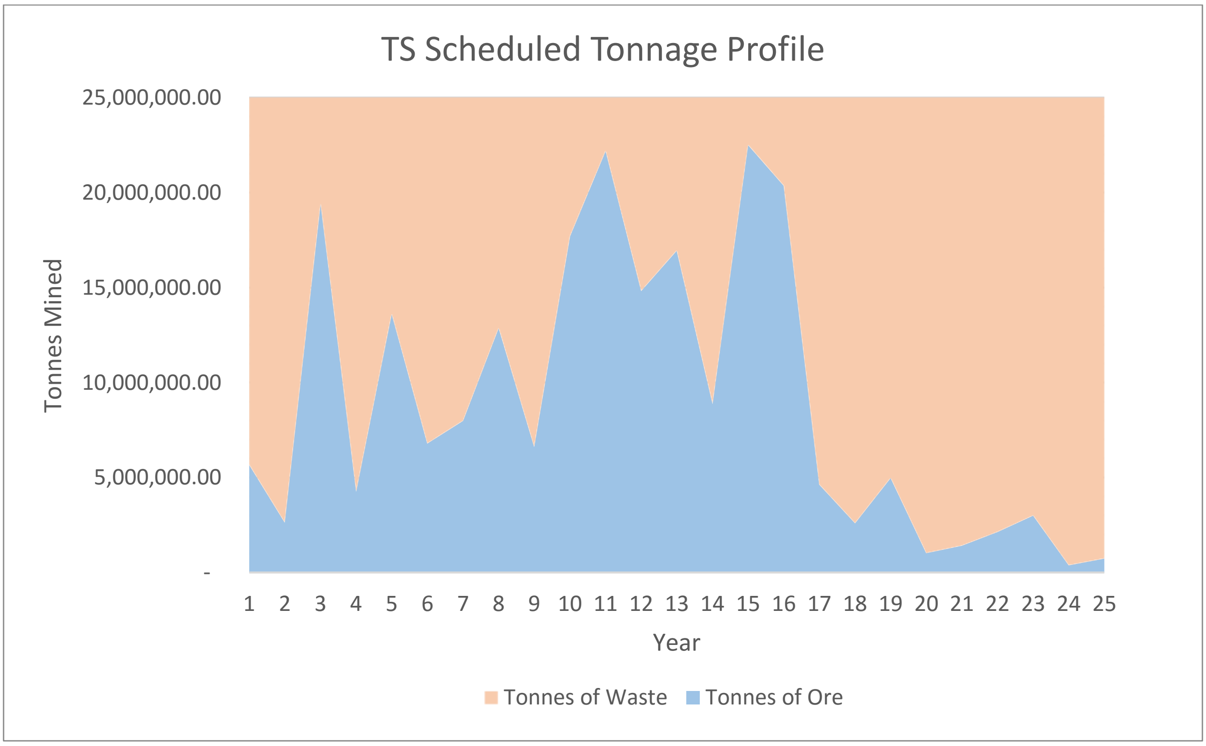

Figure 41.TS Scheduled Tonnage Profile.

6.4.2. *IPCC*

The production scheduling for IPCC systems used a similar method to the TS interactive scheduler, however a number of rules are introduced to control the direction of mining. Due to the constraints of IPCC mining, the sequence followed a more bench by bench technique than the TS. This resulted in a significantly larger waste production early in the mine life, with very little ore being retrieved until the fourth year of operations for both 26B and 36B, as shown in Figures 42 and 43.

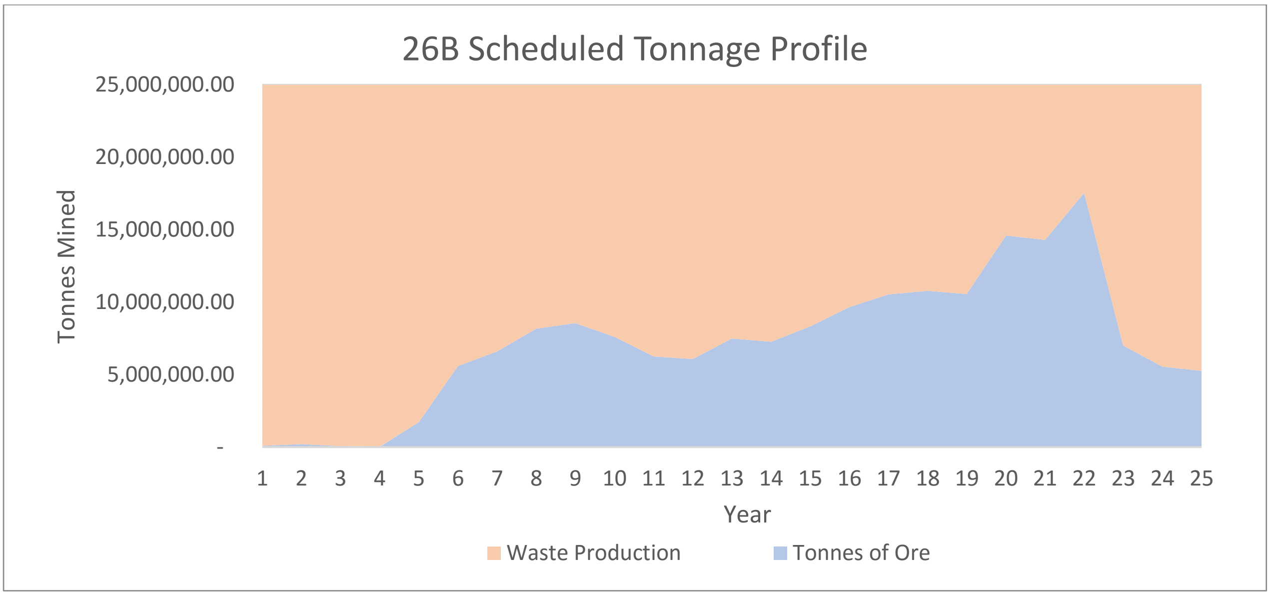

Figure 42. 26B scheduled tonnage profile.

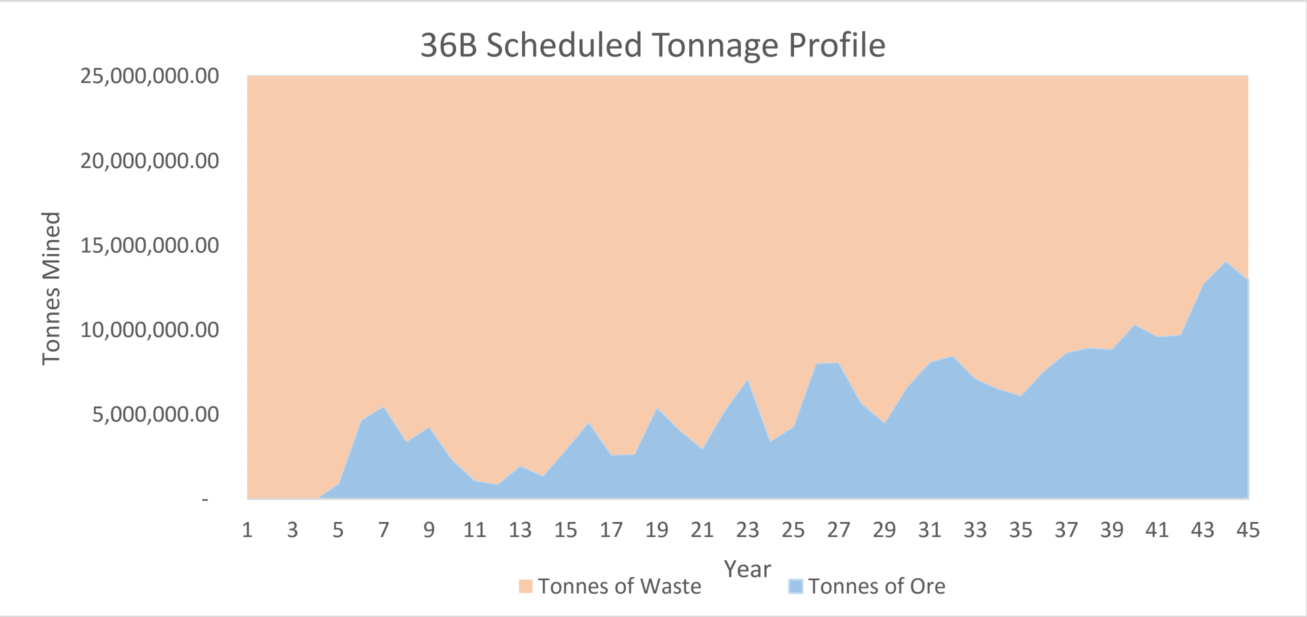

Figure 43. 36B scheduled tonnage profile.

6.5. COST ESTIMATION

A critical part in the mine planning stage of this research project was applying accurate costs, capacities and parameters to the model, assisting in making this project as similar to real world scenarios as possible. The CostMine cost guide is the industry standard for mine cost estimating, and was used to investigate the parameters and costs.

6.5.1. *Conveyors*

Firstly, the capital costs and operational costs for in-pit conveyors were investigated. Table 13 shows the costs for fixed conveyors with rigid steel idler supports, 610 m in length, suitable for material weighing 800 kg/m3. The CostMine handbook states *For shorter or longer systems, adjust the base installed price by adding or subtracting the installed price per metre. Installed price includes shifting rail, field wirings, controls, steel and mechanical installation for a "turn key" project.*

Table 13. In-pit conveyor system costs (CostMine, 2016).

Belt Width (cm)	*Capacity (tph)*	*Belt Rating (kg/cm width)*	*Capital Cost ($)*	*Capital Cost per metre ($)*	*Operating Cost per hour ($)*
76.2	454	59	1 976 000	3,239	209.30
91.4	680	59	2 206 000	3,616	238.82
107	907	78	2 380 000	3,901	263.09
122	1361	78	2 474 000	4,055	279.84
137	1814	107	3 319 000	5,440	375.03
152	2268	107	3 722 000	6,101	428.72
183	2722	107	4 176 000	6,845	487.12
183	3629	143	4 298 000	7,045	514.50

The only conveyor listed in Table 13 that satisfied a production of 25 Mtpa was the conveyor with a capacity of 3629 tph, therefore this conveyor was implemented into the model as the in-pit conveyor using a cost per metre of $7,045.90.

Table 14 represents the costs for overland conveyor systems, similar to the in-pit conveyors the price per metre was established as instructed by CostMine. As can be seen, the only conveyor to satisfy the annual production rate was the conveyor with a 4000 tph capacity. This conveyor was implemented into the model as the overland conveyor.

Table 14. Overland IPCC system costs (CostMine, 2016).

Belt Width (cm)	*Capacity (tph)*	*Belt Rating (kg/cm width)*	*Capital Cost ($)*	*Capital Cost per metre ($)*	*Operating Cost per hour ($)*
76.2	500	800	4 551 000	2817	506
76.2	680	1000	4 736 000	2932	540
91.4	1000	1000	5 202 000	3221	600
107	1500	1000	5 840 000	3616	667
122	2000	1200	6 304 000	3903	782
137	2500	1200	7 500 000	4643	926
152	3000	1200	7 848 000	4859	960
183	4000	1200	8 900 000	5510	1115

6.5.2. *Trucks*

The parameters for typical trucks were investigated, such as the bucket capacity, tonnage, and empty loading height. Associated operating and capital costs for trucks with specific parameters were investigated using the CostMine cost guide. Table 15 displays the haul truck costs and parameters. The truck that best matched the CAT 797 was the 250 tonne truck, and thus was implemented into the model.

Table 15. Haul truck costs and parameters (CostMine, 2016).

Tonnage	*Haul Capacity (m^3)*	*Empty Loading Height (m)*	*Capital Cost ($)*	*Operating Cost ($/hr)*
40	24.5	3.3	958 000	73
60	30.6	3.6	987 500	76
100	35.2	3.8	1 239 000	101
100	57.3	4.2	1 911 000	140
150	78.0	5.0	3 238 000	204
200	105	5.5	4 407 000	275
250	148	5.9	5 310 000	372

6.5.3. *Stackers/Spreaders*

The CAPEX and OPEX for stackers and spreaders were investigated, and the CostMine cost guide summary for stackers and spreaders is shown in Table 16. The stacker that best matched the production rate had a capacity of 3628 tph, and the associated CAPEX and OPEX adopted within the model.

Table 16. Spreader/stacker costs and parameters (CostMine, 2016).

Capacity (tph)	*Length (m)*	*Width (cm)*	*Cost ($)*	*Operating Cost ($/hr)*
1270	200	76.2	7 570 000	475
1995	400	91.4	13 500 000	858
2721	600	122	19 800 000	1271
3628	200	137	9 600 000	642
4535	200	152	10 890 000	733

6.5.4. *Crusher*

Semi-mobile crusher prices were investigated, as they were not included in the CostMine handbook. The price of a semi-mobile crusher varies with manufacturer, companies such as; Mammoet, Sandvik, and Joyal provide semi-mobile crushing solutions to mining companies. The prices for these systems range from $5 000 000 AUD to $15 000 000 AUD depending on the throughput. A capital cost of $13 200 000 was adopted for this investigation to be conservative. Operating costs of $310 per hour were used.

6.5.5. *Cost Summary*

Table 17 summarises the costs used in the investigation.

Table 17. Equipment costs summary (CostMine, 2016).

Equipment	*Cost per Unit ($ AUD)*	*OPEX ($/hr)*
In-pit conveyor	5510 per m	515
Overland conveyor	4859 per m	1115
Haul Truck	5 310 000	272
Stacker/Spreader	9 600 000	642
Crusher	13 200 000	310

6.6. Fleet Size Estimation

RPMGlobal Talpac software was used to determine fleet sizes at various stages in the mine life. *Talpac* is a haulage and loading simulator that provides a list of industry equipment and associated specifications to be selected for simulations. Haul road lengths were measured from the three pit designs for various stages in the mine life and applied into Talpac to obtain realistic fleet size and cycle times.

6.6.1. *TS Pit*

The fleet size for the TS pit increases incrementally, as the pit gets larger the haul road lengthens, and raises the number of trucks needed in the fleet size to maintain production. As expected, the maximum haul road distance was during the final year of operations (year 25), and the cycle travel distances are shown in Table 18. This cycle required a fleet size of 23 trucks, and the travel time was approximately 44 mins/cycle. The cumulative fleet size per year is shown in Figure 44.

Table 18. TS maximum haul road distances.

Title	*Distance (metres)*	*Grade*
Haul Road Out	6000	10%
Pit to WRD	1000	0%
WRD to Pit	1000	0%
Haul Road Down	6000	-10%

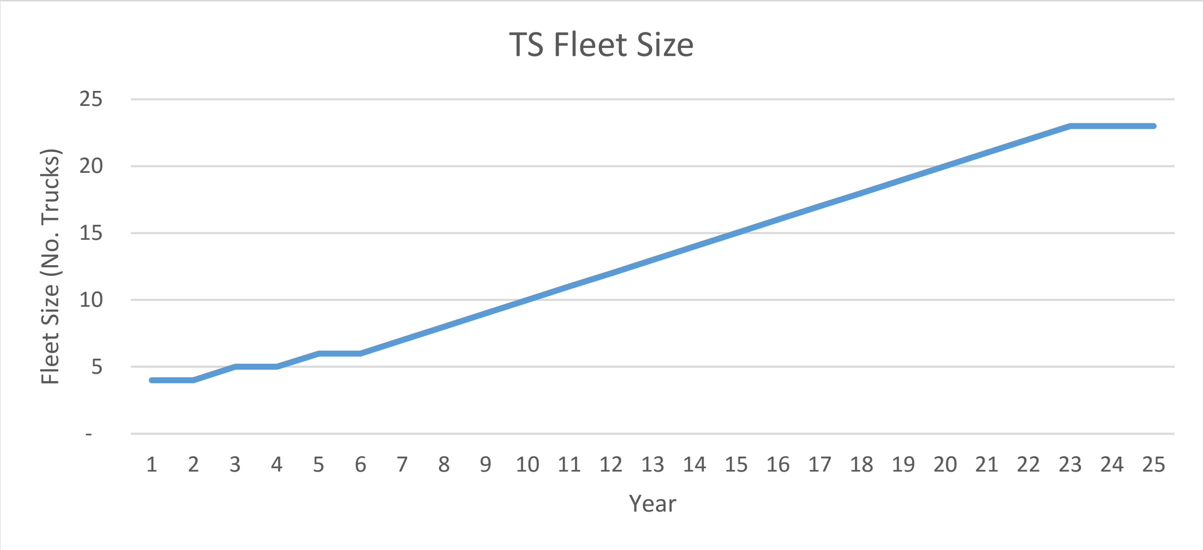

Figure 44. TS Fleet Size.

6.6.2. *IPCC 26B*

The semi-mobile IPCC system reduces the fleet size for the mine by reducing the distance the trucks need to travel to dump their load. The minimum fleet size will occur immediately after a crusher move has occurred, as the distance from the loader to the crusher will be at its smallest. As the crusher remains fixed and the loader progresses around the working area of the mine the truck fleet size will increase. For the initial box cut (first 3 years of operation) the mining system was purely TS, and the crusher was introduced into year 4. As a result, the largest haul route and subsequently the largest fleet size occurs in year 3. This resulted in a fleet size of eight trucks for year 3. Over the 25 year life of mine, the crusher was moved a total of 3 times, and therefore after the third year the fleet size was capped at five trucks before the crusher was moved again. Table 19 shows the maximum haul road specifications, which occurs at year 3, resulting in a fleet size of 8 trucks. The cumulative fleet size per year is shown in Figure 45, crusher moves occurred in years 4, 13 and 22.

Table 19. IPCC 26B maximum haul road distances.

Title	*Distance (metres)*	*Grade*
Haul Road Out of Pit	1030	10%
Pit to WRD	1000	0%
WRD to Pit	1000	0%
Haul Road Into Pit	1030	-10%

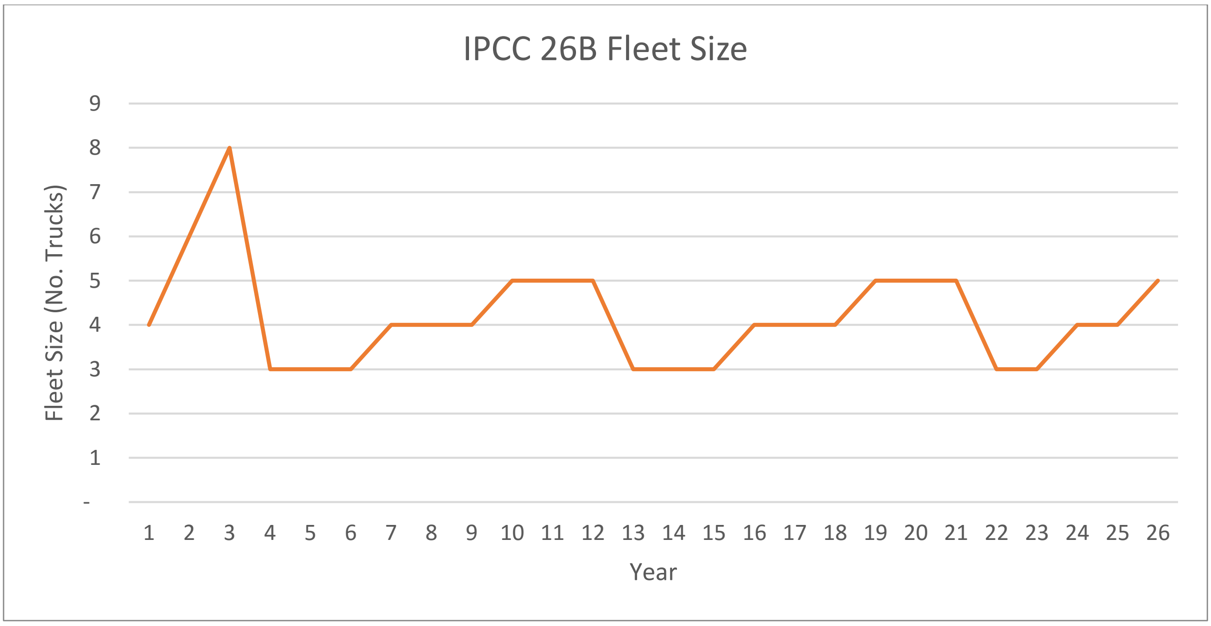

Figure 45. IPCC 26B Fleet Size.

6.6.3. *IPCC 36B*

Similar to pit 26B, the largest fleet size for pit 36B was during the initial years of operations during the boxcut, resulting in nine trucks. The largest haul distance occurred in year 4, and is summarised in Table 20. Over the 45 year life of mine the crusher was moved five times, capping the fleet size at five trucks after year 4. Crusher moves occurred in years 5, 11, 17, 27 and 38. The cumulative fleet size per year is shown in Figure 46.

Table 20. IPCC 36B maximum haul road distances.

Title	*Distance (metres)*	*Grade*
Haul Road Out of Pit	1230	10%
Pit to WRD	1000	0%
WRD to Pit	1000	0%
Haul Road Into Pit	1230	-10%

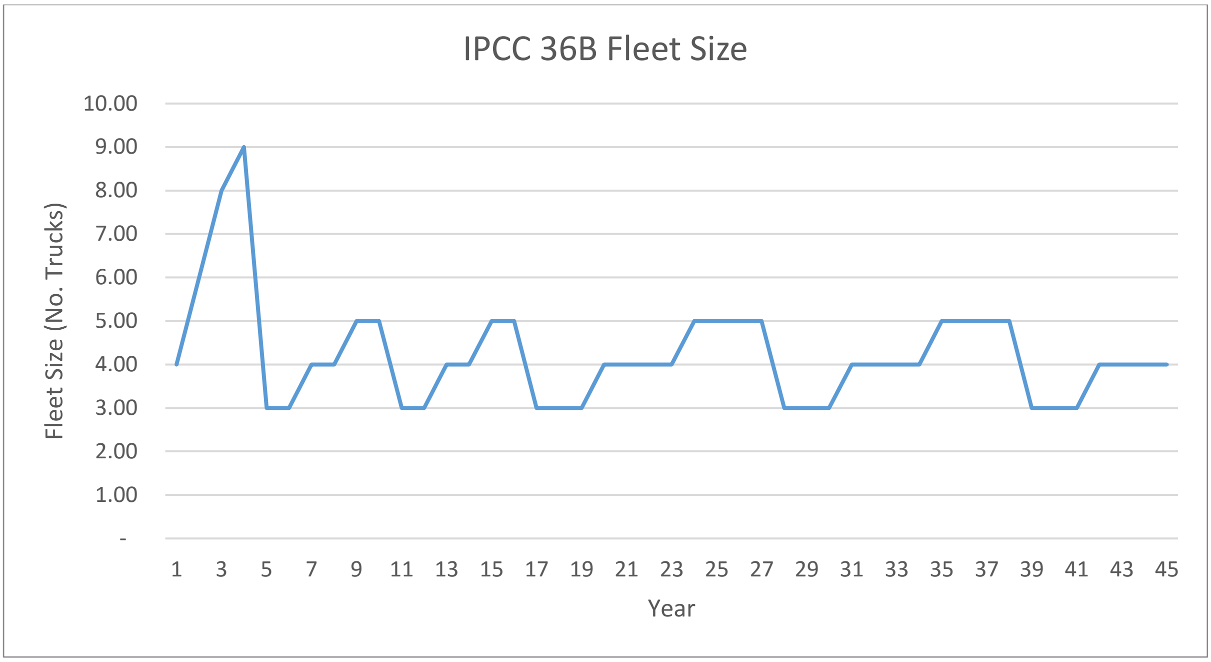

Figure 46. IPCC 36B Fleet Size.

7. RESULTS

7.1. REVENUE

The revenues for the three pits were calculated. *Deswik*'s interrogation function was used to determine the properties of each block that is being extracted, and these were input into the mathematical model, incorporating gold and copper prices as well as mining and processing costs to compute the revenue for each block. The extracted blocks for each pit were then summed, providing a total revenue for the pit. Table 21 summarises the total revenues for all three simulations. IPCC 36B yielded the greatest undiscounted revenue with $16.15 B, this is to be expected as the low mining cost presented by the IPCC mining system allows pit 36B to attain a greater recovery than the TS pit. IPCC 26B yielded the lowest undiscounted revenue, in accordance with its low recovery.

The revenues could be broken down year by year in accordance with the production schedule. A discount rate of 10% was applied to the yearly revenues, and the total discounted revenues are shown in Table 21. The TS system had the greatest discounted revenue and IPCC 36B had the lowest discounted revenue, this was largely due to the block extraction schedule. As stated earlier, the TS pit allowed for early access due to the pit shape and progression, allowing for early ore sale. The IPCC pits were restricted to waste production early in the mine life, and large ore production later in the mine life. When the discount rate was applied, the present value of the later years was significantly minimised, reducing the profitability of the mines.

Table 21. Revenue.

Pit Simulation	*Undiscounted Revenue ($)*	*Discounted Revenue ($)*
TS	13.50 B	5.05 B
IPCC 26B	10.22 B	2.90 B
IPCC 36B	16.15 B	1.90 B

Figure 47 illustrates the cumulative yearly undiscounted revenue of the pit. The TS system exhibits a steady revenue increase over the 25 years, whilst the IPCC systems have a large delay in the beginning of the mine life and then increase.

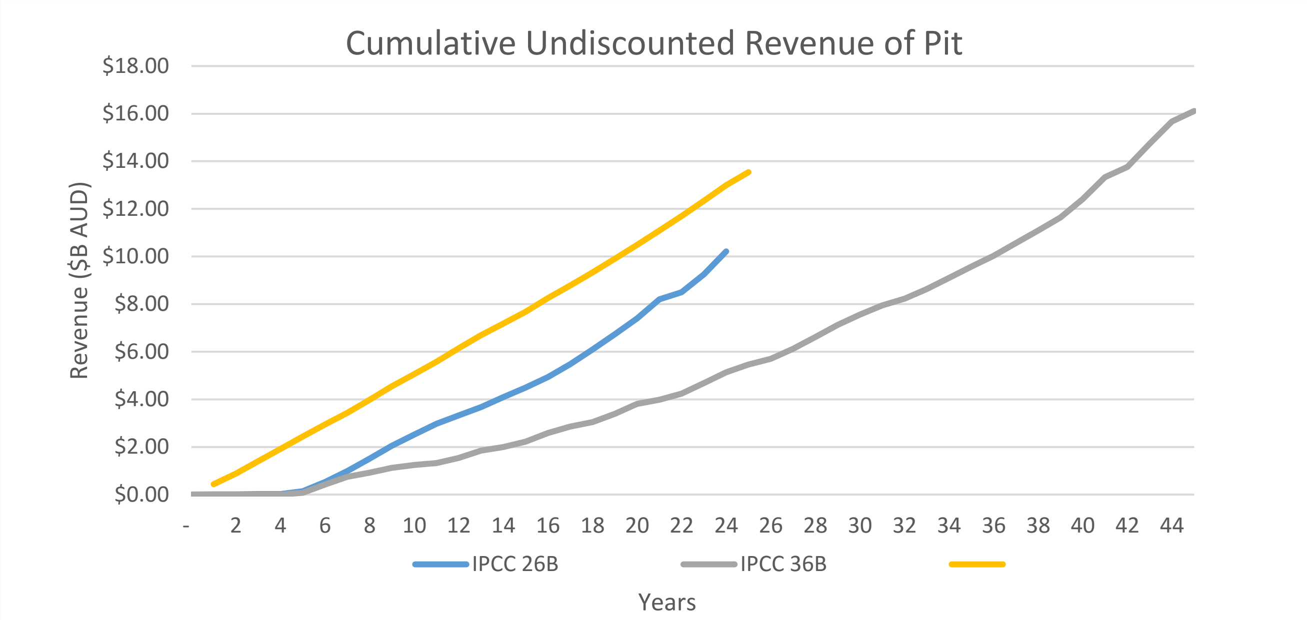

Figure 47.Cumulative yearly revenue.

When the discount rate is applied to the yearly revenue, as shown in Figure 48, the IPCC pits have a slow increase, due to the early waste production. The 'dampening' effect of the discount rate on all three pits is quite evident compared to the undiscounted revenue. Table 29 in Appendix C summarises the yearly discounted revenues.

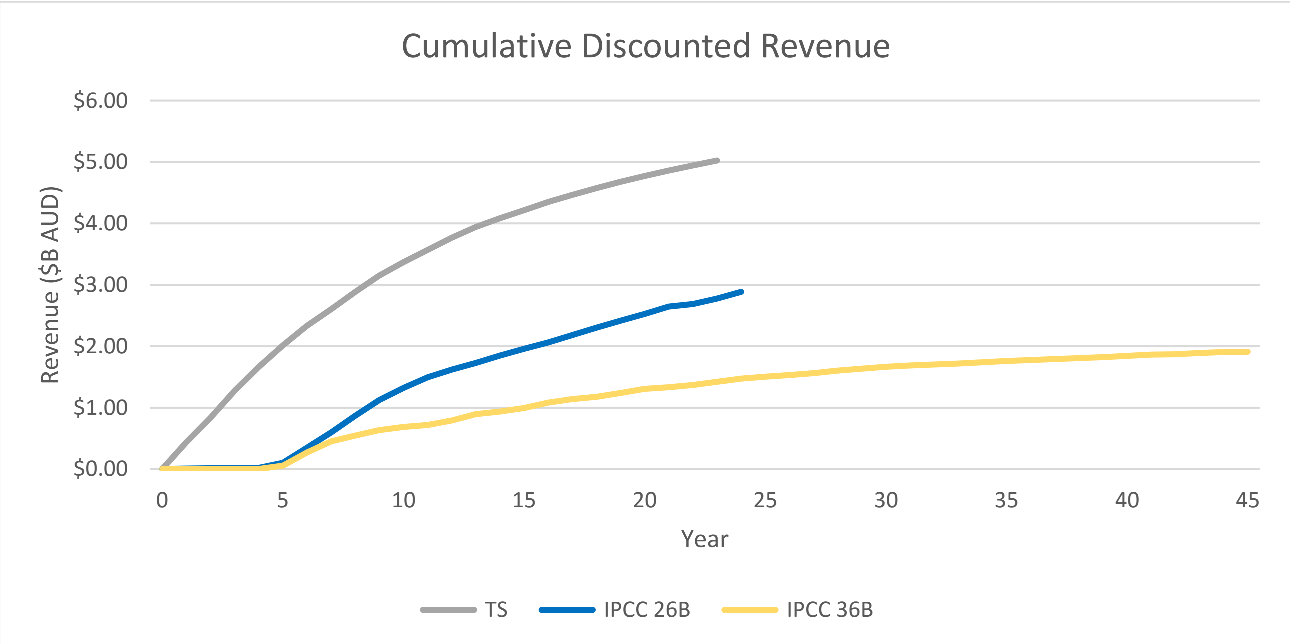

Figure 48. Discounted Cumulative Revenue.

7.2. OPEX

The yearly operational costs were calculated for all three pit simulations. These OPEX values were the result of combining equipment costs per hour (attained from CostMine) with yearly operations labour costs and salaried labour. It was assumed that the operations team would consist of four crews working a 12 hour shift.

It was found that the largest influence on the OPEX was the number of trucks in operation. The total operational employees changed from year to year for all three mines as the number of trucks, and hence the associated labour (truck drivers and mechanics), increased for TS each year and fluctuated for IPCC systems. It was assumed that within each crew, there was one truck driver for each truck and one mechanic for every two trucks. These operational workers that are dependent on the fleet size were labelled truck workers. For example, in the initial years of the TS, when the fleet size was four, the number of truck workers was 24. However, in the final years when the fleet size is 23, the number of truck drivers is 138. Table 30 in Appendix D summarises the yearly truck workers for all three mines.

The yearly operational costs for all three mine simulations were plotted, and shown in Figure 49. The TS OPEX increases steadily throughout the mine life, as a result of the pit haul road length increasing and hence raising the labour and equipment costs associated with the trucks. Both IPCC OPEX follow a similar trend to one another, whereby the costs start to increase until a crusher move occurs and the OPEX drops. As stated earlier, this is because with each crusher move the fleet size decreases, and hence the labour and equipment costs associated with the trucks decrease. Both 36B and 26B experience the greatest OPEX in the initial years of the mine life when the IPCC system was not yet activated and the fleet size was at its greatest.

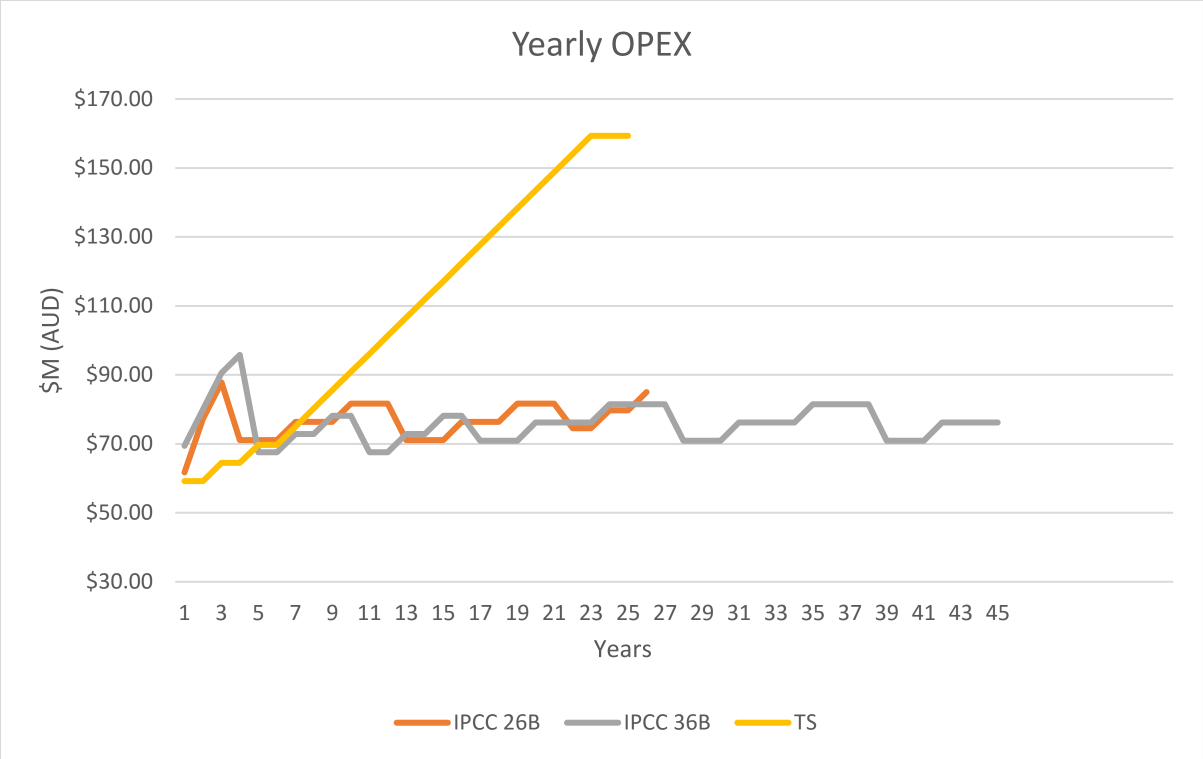

Figure 49. Yearly OPEX.

The average mining cost for all three pit simulations were calculated, and the results are summarised in Table 22. It is evident that based on equipment costs alone, the IPCC systems are $0.30 to $0.37 cheaper per tonne than the TS system. Taking into account labour and salary, the total mining cost for IPCC has savings of up to $1.43 per tonne compared to TS operations. It can be seen that 26B offers the lowest overall mining cost, due its shallow pit necessitating less conveyor power requirements than 36B and a smaller fleet size than the TS system.

Table 22. Average mining cost.

	TS	*IPCC 26B*	*IPCC 36B*
Equipment ($/t)	1.26	0.88	0.96
Labour & Salary ($/t)	2.97	1.92	1.91
Total Mining Cost ($/t)	4.23	2.80	2.87

7.3. CAPEX

The equipment capital costs were calculated for all three simulations, using the prices obtained from the CostMine handbook. On top of these equipment costs, a constant CAPEX cost of $200 M AUD was adopted for all three simulations. This was to account for infrastructure, i.e workshop and processing plant, as well as equipment that will be required for all three mine simulations, including;

- Dozers;
- Graders;
- Water trucks; and
- Drilling and blasting equipment.

7.3.1. *TS*

The total undiscounted CAPEX for the TS pit was determined, and shown in Table 23. This resulted in a total undiscounted CAPEX of $122.1 M AUD.

Table 23. TS CAPEX.

Equipment	*Unit Cost ($)*	*Quantity*	*Cost ($)*
In-pit conveyor	7046 /m	0	0
Overland conveyor	4859 /m	0	0
Crusher	13 200 000	0	0
Spreader	9 600 000	0	0
Stacker	9 600 000	0	0
Truck	5 310 000	23	122 130 000
Total			122 130 000

7.3.2. *IPCC 26B*

The total undiscounted CAPEX for the IPCC 26B pit was determined, and shown in Table 24. This resulted in a total undiscounted CAPEX of $86.6 M AUD.

Table 24. IPCC 26B CAPEX.

Equipment	*Unit Cost ($)*	*Quantity*	*Cost ($)*
In-pit conveyor	7046 /m	980 m	6 904 983
Overland conveyor	4859 /m	1000 m	4 859 000
Crusher	13 200 000	1	13 200 000
Spreader	9 600 000	1	9 600 000
Stacker	9 600 000	1	9 600 000
Truck	5 310 000	8	42 480 000
		Total	86 614 000

7.3.3. *IPCC 36B*

The total undiscounted CAPEX for the IPCC 36B pit was determined, and shown in Table 25. This resulted in a total undiscounted CAPEX of $95.4 M AUD.

Table 25. IPCC 36B CAPEX.

Equipment	*Unit Cost ($)*	*Quantity*	*Cost ($)*
In-pit conveyor	7046 /m	1470 m	10 357 500
Overland conveyor	4859 /m	1000 m	4 859 000
Crusher	13 200 000	1	13 200 000
Spreader	9 600 000	1	9 600 000
Stacker	9 600 000	1	9 600 000
Truck	5 310 000	9	47 790 000
Total			95 406 000

Table 26 summarises the total undiscounted CAPEX for all three pit simulations. The IPCC 26B pit has the smallest CAPEX, this is to be expected as 26B has the smallest fleet size, and smallest conveyor length. The TS system has the greatest total undiscounted CAPEX, due to the large fleet size. However, the advantage of the TS system is the implementation of trucks throughout the mine life, rather than purchasing a majority of the equipment in the initial years of the mine life. IPCC systems require the conveyor, spreader, stacker and crusher as soon as the IPCC operations are expected to begin, making the system more CAPEX intensive earlier in the mine life. Therefore the CAPEX was looked at on a yearly basis. It was assumed that all equipment needed replacing after 7 years. Figure 50 shows the cumulative undiscounted CAPEX for all three scenarios.

Table 26. CAPEX summary.

Pit Simulation	*Cost ($)*
TS	122 130 000
IPCC 26B	86 614 000
IPCC 36B	95 406 000

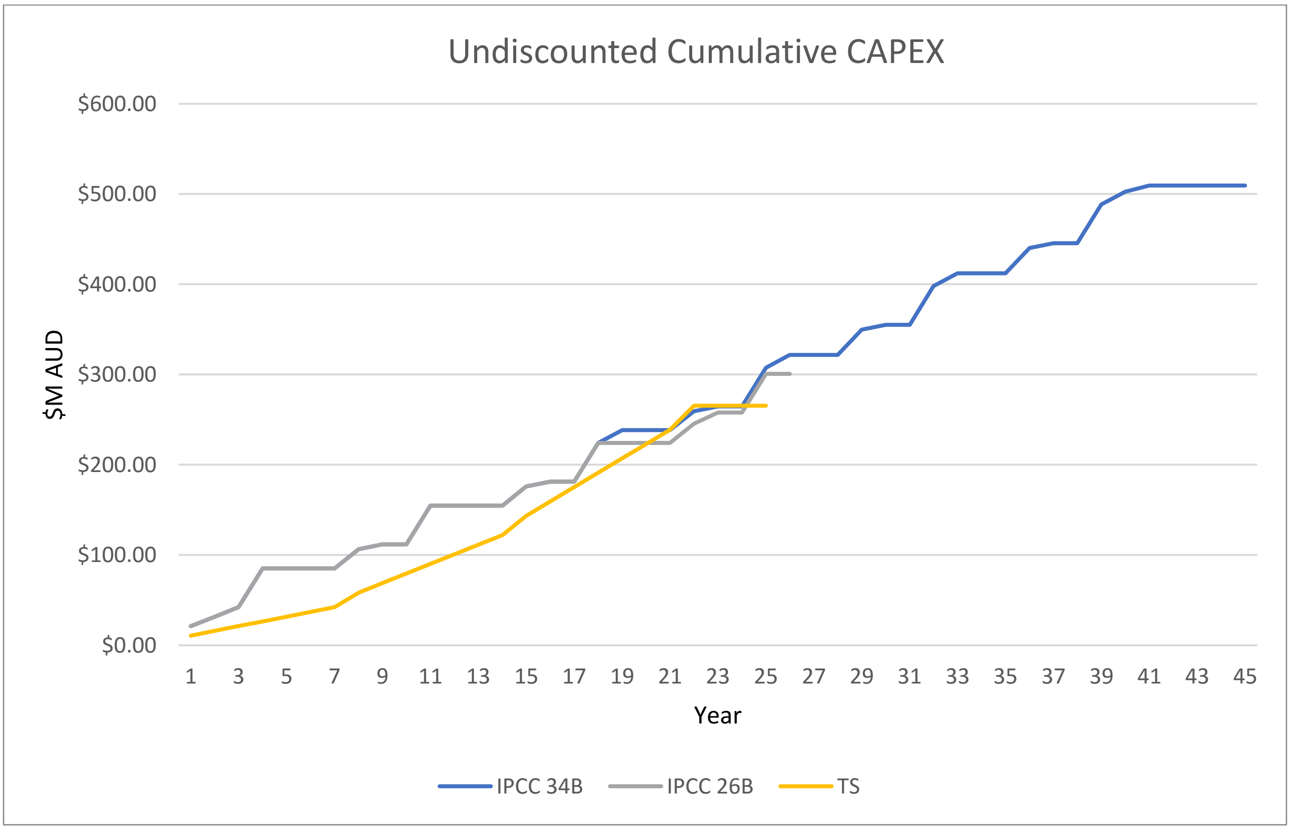

Figure 50. Undiscounted Cumulative CAPEX.

It’s evident from the plot of yearly undiscounted cumulative CAPEX that the TS system offers the lowest CAPEX in the initial years of the mine life, and then steadily increases each year with the addition of trucks to the fleet. The IPCC systems have a larger CAPEX during the initial years of the mine life, and both systems are relatively identical until approximately year 19 when the crusher moves and conveyor additions begin to differ due to pit shapes. At year 22, all three systems have very similar cumulative CAPEX.

Figure 51 shows the discounted cumulated CAPEX, with a 10% discount rate applied. The margin between the IPCC systems and TS is greater when the discount rate is applied, it can be seen that TS is consistently lower than the IPCC systems, and at the end of the mine life is approximately $30 M AUD less than the others.

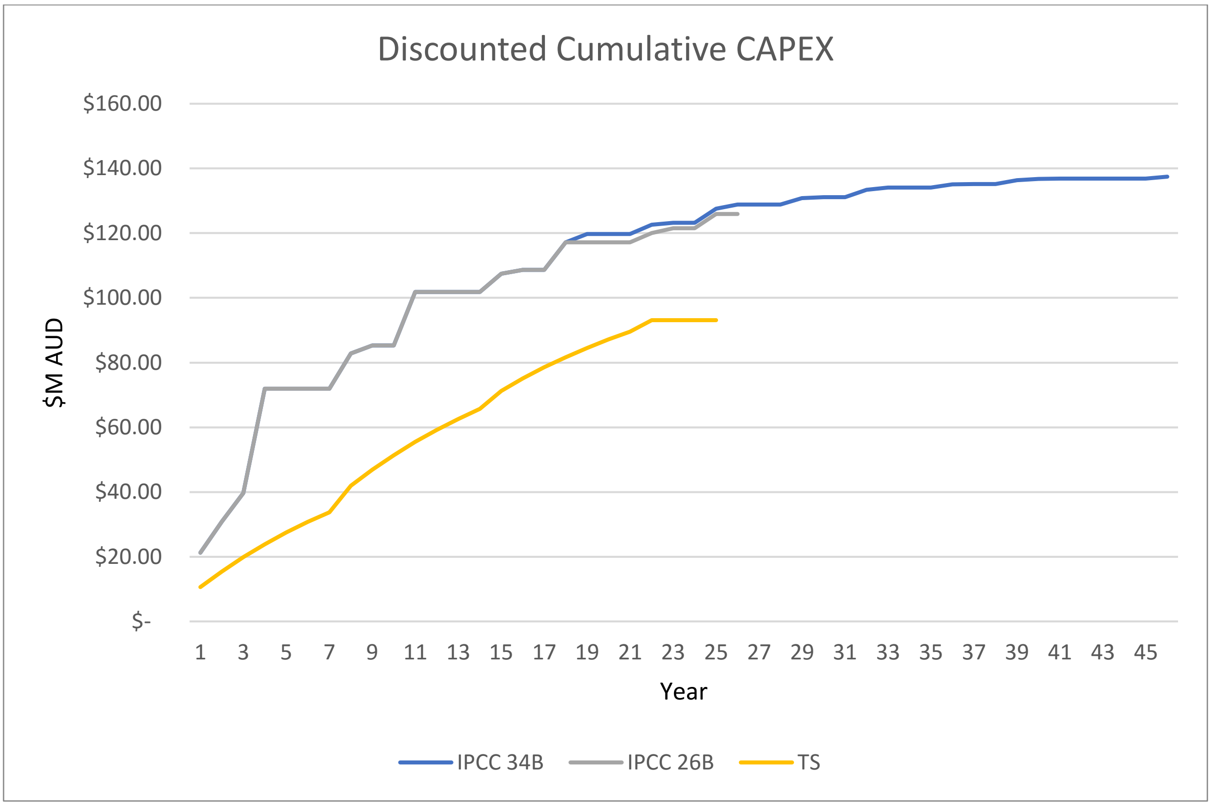

Figure 51. Discounted Cumulative CAPEX.

7.4. NPV

The revenue, OPEX and CAPEX for each simulation was input into an FTM to provide a representation of a real world financial system. The FTM uses a mathematical model to represent the performance of a mine and can provide a project NPV and internal rate of return (IRR). The NPV is the difference between the present value inflows and outflows, and is used to analyse the profitability of a project. A positive NPV indicates the project is profitable, whereas a negative NPV represents net loss. The IRR is defined as *the interest rate at which the net present value of all the cash flows (both positive and negative) from a project or investment equal zero* (Investing Answers, 2013). The NPV and IRR are summarised for all three pits in Table 27.

Table 27. FTM results.

Pit Simulation	*NPV ($M AUD)*	*IRR*
TS	1325	63%
IPCC 26B	553	10%
IPCC 36B	108	-11%

The TS pit yielded the greatest NPV, with $1.325 B, followed by IPCC 26B with $553 M and IPCC 36B with $108 M. Figure 52 shows the yearly NPV. Both IPCC mines exhibit negatively increasing present values for the initial years of the mines. This is because the IPCC pits extract very little ore in the early years, and the OPEX exceeds any profits, thus generating negative cash flows. After year 6 the cash flows become positive, and the present value breaks even in year 12 for 26B and year 30 for 36B. With a mine life of 45 years, any profit made by 36B later in the mine life was rendered insignificant by the discount rate, resulting in a substantially low NPV.

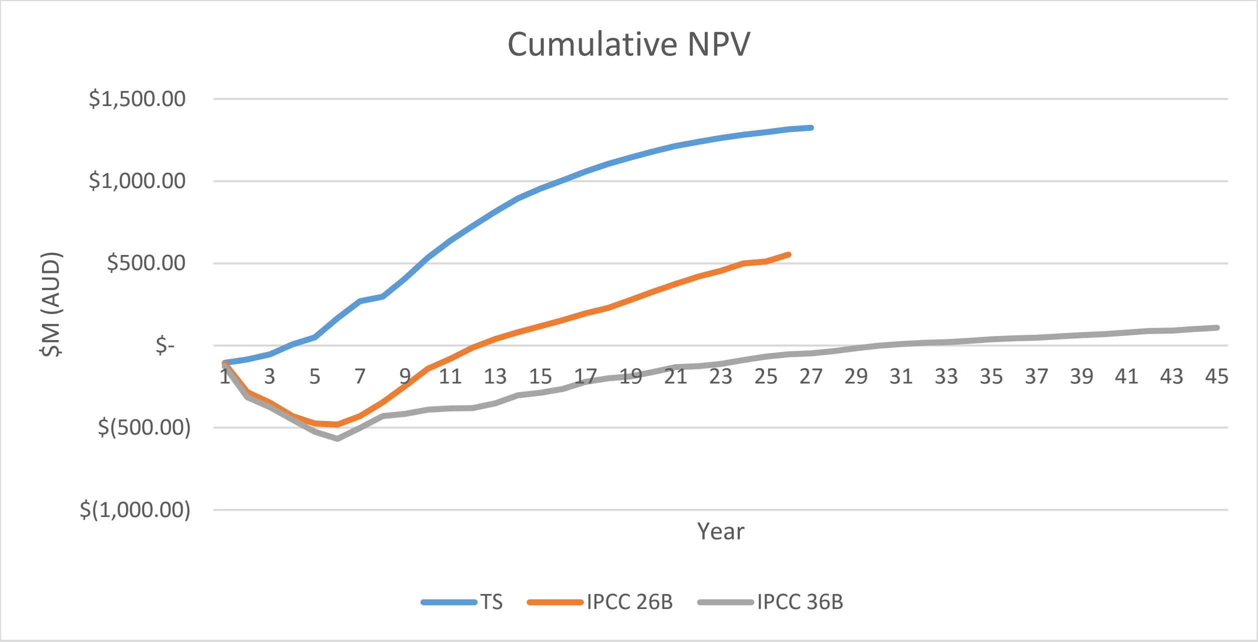

Figure 52.Cumulative NPV.

8. COMPARISON

A comparison between IPCC and TS as a method of haulage in open pit metalliferous mines has been made on a costs basis.

It was found that IPCC is a cheaper mode of haulage than TS from an OPEX standpoint, largely due to driverless operations and electrical power. The maximum fleet size for the TS pit was calculated to be 23 trucks, whilst the maximum fleet size for both IPCC simulations was 8 and 9. This resulted in savings of up to $1.43 per tonne, and up to $36 M annually. The low mining costs associated with IPCC operations allowed for a greater optimum pit depth, and hence a greater pit recovery than the TS pit. Accordingly, IPCC yielded the greatest undiscounted pit revenue. However, when revenue streams were analysed on a yearly basis and a discount rate was applied, the TS system had the greatest undiscounted revenue due to the production scheduling of both systems.

Whilst the TS system presented the highest total undiscounted CAPEX due to the immense fleet size, it was found that by implementing the relative equipment when necessary over time the TS had the lowest CAPEX.

The largest influence on the NPV was the pit progression due to the respective mining system. It was found that the mining constraints associated with IPCC operations hindered the profitability of the mine. Through early waste production and delayed ore extraction. The early ore retrieval made possible by the TS pit presented a larger NPV than the IPCC pits. Therefore, the TS system is the most ideal mining system when NPV is regarded as the highest priority in mine valuation. Conversely, the large recovery presented by IPCC 36B indicates that IPCC systems can be the most ideal mining system in countries that value recovery over NPV in mine valuation, such as Post-Soviet countries.

The current global environment has seen mining companies facing increasing pressure in seeking investment (Patterson, Kozan & Hyland, 2017). The low capital investment and short payback period offered by TS systems have the potential to make them more appealing to investors, compared to IPCC methods.

The deep, steep and tabular nature of the deposit considered was a controlling factor on the success of the IPCC system. As discussed earlier, the inherent constraints associated with IPCC systems hinder the profitability of the mine, and deep tabular deposits require waste intensive pit progression in order to achieve optimal recovery. Metalliferous open pit mines of the future will be increasing in depth and decreasing with grade as a majority of the earth's shallow and high grade deposits have been exploited (Turnbull and Cooper, 2009). As a result, upfront TS OPEX costs are expected to increase, however, as optimum pit designs are more readily accommodated by TS operations, the system will remain the most economical choice of haulage in most cases.

The necessary conveyor ramp set up required for a shallow pit is significantly less than a deeper deposit, and allows for earlier ore extraction. For deposits such as iron ore or shallow coal seams, improved production scheduling and lower operational costs ensure that IPCC systems will be a competitive alternative to TS. For IPCC to compete with TS in deep metalliferous open pit mines, associated waste mining and upfront capital expenditure will need to reduce.

Currently sandwich conveyors are rarely used in in-pit operations due to capacity limitations and high unit CAPEX. If the maximum capacity was increased, the conveyor ramp angle could increase significantly and allow for earlier ore access. Additionally, as geotechnical technology improves and underground development becomes cheaper, the tunnel method could allow for IPCC operations to compete economically. Whilst this investigation only considered using either an IPCC system or TS, the combination of both systems could combine the benefits of IPCC (low OPEX, superior safety and low carbon emissions) with the optimum pit benefits of TS systems (early ore access and greater flexibility). Figure 53 shows a pit design that implements a conveyor slot into an existing TS pit later in the mine life.

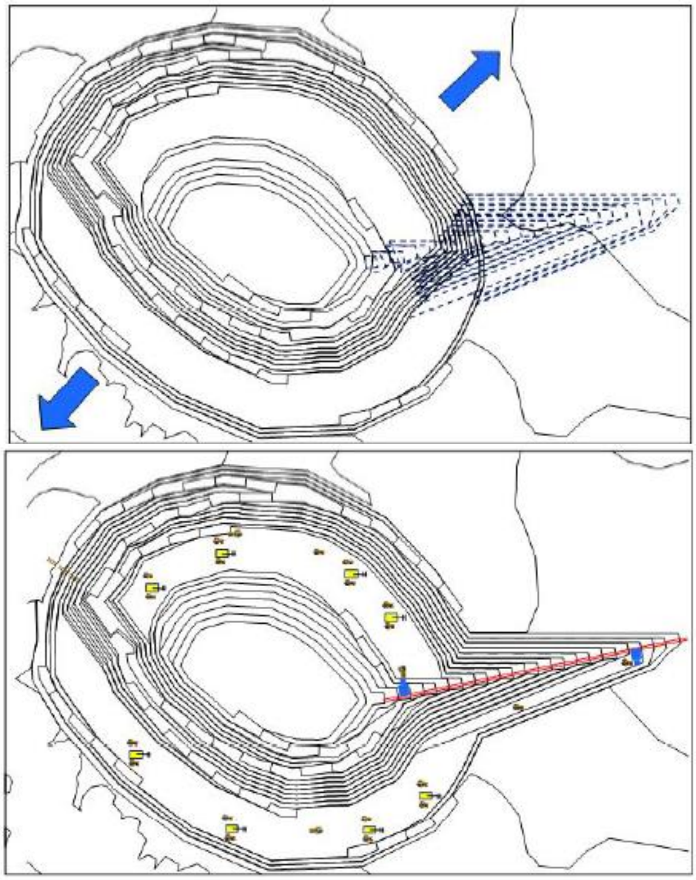

Figure 53. Deep cylindrical deposit pit extents (Tutton and Streck, 2010).

This investigation shows that for deep open pit metalliferous deposits the conventional TS system remains the most economically viable mining method. This provides mine planners and engineers with further understanding of IPCC implementation in open pit mines, demonstrating the constraints and limitations that are inherent in IPCC systems, in particular the designated ramp method. An understanding of the advantages and disadvantages that IPCC have compared to TS systems on deep open pit deposits will assist mine planners to make the best informed decision for mine planning.

To counter the rising costs of TS systems, rather than searching for alternative material transportation methods, one solution could be refining and improving the existing truck technology. Implementing automation technology and electrical power into truck fleets could see TS eclipse IPCC in operational cost efficiency.

9. CONCLUSIONS

During mine planning, mining method selection is critical in achieving the optimal pit design. As metalliferous open pit mining advances to new depths, companies search for cost effective alternatives to the conventional TS mining method. Increased haulage routes, and associated fleet size and operational costs, result in significant TS material transportation costs. In these contexts, IPCC is increasingly a viable alternative to conventional TS operations, in part due to lower inherent operational costs. This project aimed to investigate the performance of a semi-mobile IPCC system as replacement to TS operations in an open pit copper-gold mine.

An extensive review of literature regarding open pit mining systems and mine planning concluded that there is a gap of understanding in the industry for determining the optimum system for open pit metalliferous mining. The review highlighted a shared ideology suggesting that whilst IPCC systems present the benefit of much lower operating costs, the system represents a significant up-front capital investment which the mine is only able to depreciate over a sufficient mine life. IPCC implementation also requires extensive mine planning and scheduling, deterring mine planners away from the advantages that the system is capable of providing.

The methodology for this project followed a conventional mine planning, design optimisation (including a block model), pit design, scheduling and cost estimation. A deep copper-gold block model was developed and used as the deposit for all mine scenarios. The copper COG was found to be 0.31% Cu for the TS system and 0.29% Cu for IPCC. *GEOVIA Whittle* software was used to determine the optimum mine plan, pushback sequence and optimum depths for the TS pit. Due to the limitations within *GEOVIA Whittle*, only the optimum depth for the IPCC pit could be generated, not the pushback sequence.

The TS pit was designed, resulting in a probable mineable reserve of 190 Mt, achieving a 60% recovery, at a stripping ratio of 1 to 2.2. Two IPCC pits were designed, IPCC 26B and 36B at depths of 390 m and 540 m respectively. IPCC 26B resulted in a probable mineable reserve of 175 Mt, achieving a 50% recovery, at a stripping ratio of 1 to 3.8. IPCC 36B pit resulted in a probable mineable reserve of 253 Mt, achieving a 70% recovery, at a stripping ratio of 1 to 5.3. The TS pit had a maximum fleet size of 23 trucks, whilst the IPCC pits only required 8 to 9 trucks, predominately being used in the initial years of the mine life.

IPCC yielded the greatest total undiscounted pit revenue; however, applying production scheduling and a discount rate to the revenues found that TS had the greatest total discounted revenue. IPCC presented lower operational costs than TS, with savings of up to $1.43 per tonne of material moved, and up to $36 M annually in OPEX.

Examination of the required capital costs on a yearly basis revealed that TS had the lowest CAPEX requirement, due to truck addition over time compared to the large upfront CAPEX that was necessary for IPCC operations. Using the yearly cash flows, OPEX and CAPEX, a FTM was generated. The TS pit yielded a greater NPV of $1.325 B, calculated on an IRR of 63%, as compared to an NPV of $555 M to implement IPCC material transport. In this instance, the TS mining method is more economical. In general terms, TS mining may be a more ideal mining system in countries where NPV is regarded as the highest priority in mine valuation.

10. RECOMMENDATIONS AND FUTURE WORK

Future work of the project includes investigating situations in which IPCC could yield a greater NPV than TS, thus assisting mine planners in recognising when one method of haulage is more economically suitable than the other. Work could also be conducted on which pit exit strategy is the most economical, or which mode of crusher portability is most effective in metalliferous deposits.

Future research work on the project could be in the following areas.

- Conduct a comparison between TS and IPCC in the context of:
  - Systems on shallower deposits, such as an iron ore deposit or shallow coal seams;
  - Use of alternative modes of crushing portability, including fully mobile and fixed in-place; and
  - Use of alternative pit exit strategies, such as tunnel exit or haul road method.
- Investigate a comparison between TS and a hybrid TS/IPCC mine that utilises the benefits of both systems.
- Compare real world data from an existing IPCC mine, including block model data, production schedule, OPEX, CAPEX and cash flows, and conduct a similar TS mine simulation on the same deposit, ultimately comparing the two.

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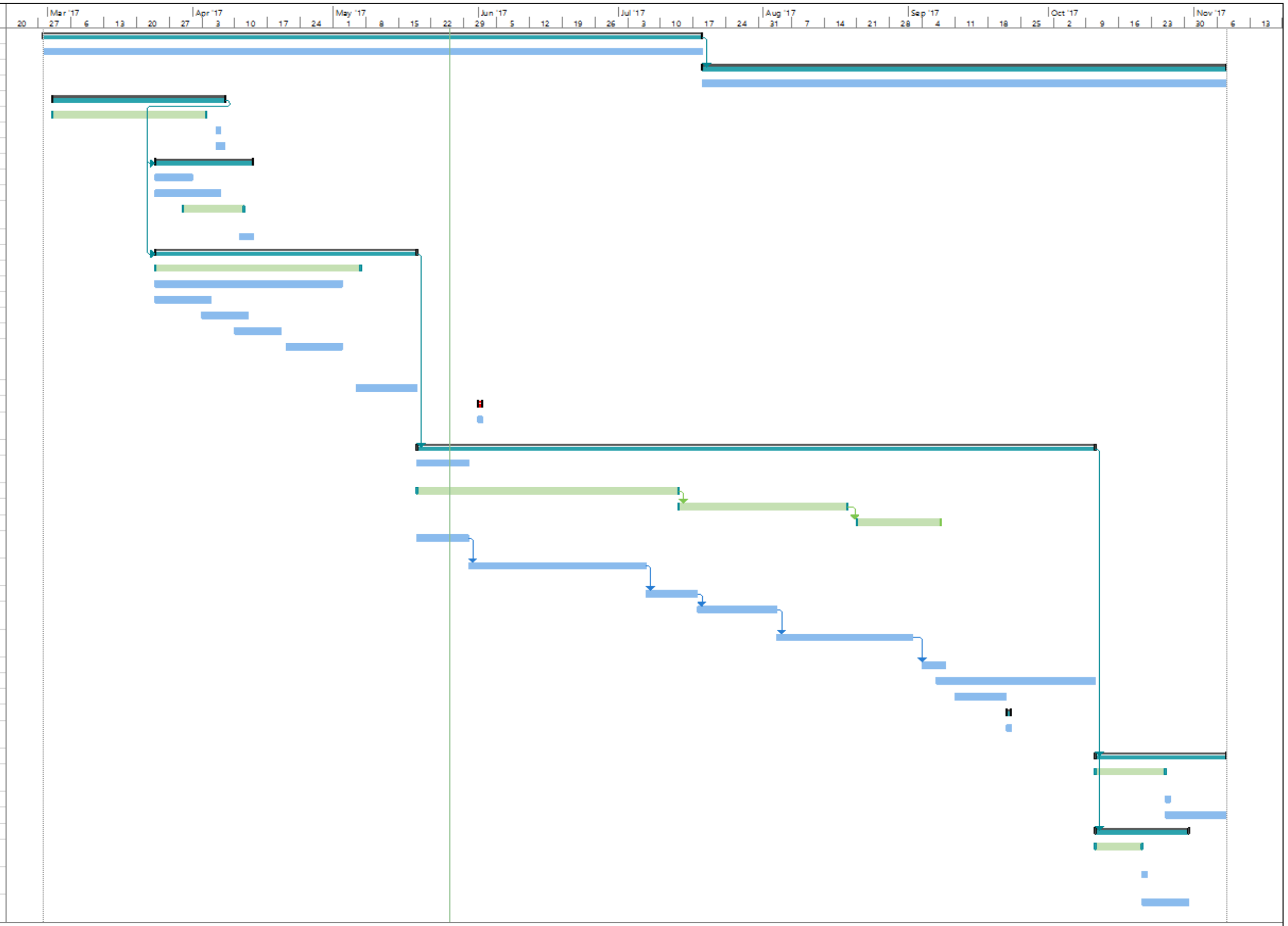
APPENDIX A: FAILURE MODES AND EFFECTS ANALYSIS

Functional Failure	*Failure Modes*	*Consequence*	*Likelihood*	*Severity*	*Level of Risk*	*Controls*	*Adjusted Likelihood (Controls)*	*Adjusted Severity (Controls)*	*Adjusted Level of Risk*	*Final Risk Rating*
Incomplete Report	Failure to submit research project	Research project cannot be credited to university studies, hence failure.	2	5	High (10)	Set milestones, complete project ahead of time and work during holidays and breaks	1	5	Medium (5)	1
	Cannot acquire data	Minimal data analysis can be undertaken	2	4	Medium (8)	Follow deadlines, vary across multiple software types	1	4	Medium (4)	=2
	Personal Injury or illness	Research project cannot be submitted or is postponed	2	3	Medium (6)	Work on project consistently and familiarise self with extension application process	2	2	Medium (4)	=2
	Lost or corrupt data/report	Research project cannot be submitted, lost time in recovering files and or standard is reduced	2	5	High (10)	Backing up data in multiple locations, for example online and external hard-drives	1	4	Medium (4)	=2
Partially completed	Insufficient data	Collected data is incomplete or unreliable	2	4	Medium (8)	Follow deadlines, vary across multiple software types	1	4	Medium (4)	=2
	Criteria is not adhered to	Important sections or details missing	3	3	Medium (9)	Criteria is thoroughly read through while completing project	1	3	Low (3)	=3
	Illness or accident	Quality of research project is reduced in quality	2	3	Medium (6)	Work on project consistently and familiarise self with extension application process	2	2	Medium (4)	=2
	Failure to make deadlines	Sections of the project are incomplete	4	4	Very High (16)	Set milestones, complete project ahead of time and work during holidays and breaks	2	2	Medium (4)	=2
Completed to a lower standard than desired	Inadequate literature available	Report lacks background information and is ill-informed	1	3	Low (3)	Source literature from elsewhere, expand horizons and call industry experts	1	2	Low (2)	=3
	Poor time management	Inadequate time to edit and or consult with supervisor	2	4	Medium (8)	Set milestones, complete project ahead of time and work during holidays and breaks	1	3	Low (3)	=3
	Loss of contact with academic supervisor	No advice or supervision for research project	1	5	Medium (5)	Set regular meetings with supervisor and seek assistance from other staff if need be	1	2	Low (2)	=4
	Insufficient data	Collected data is incomplete or unreliable	2	3	Medium (6)	Follow deadlines, vary across multiple software types if need be	1	2	Low (2)	=4
	Inadequate proofreading, editing and time to fix mistakes	Information could be presented poorly, grammatical mistakes and formatting errors	2	4	Medium (8)	Set milestones, complete project ahead of time, work during holidays and breaks and request a peer review	2	2	Medium (4)	=2
	Project strays too far from intended objective	Standard of report is lower than desired due to conflicting scopes and visions	2	4	Medium (8)	Modify or slightly adjust the scope and objectives	2	2	Medium (4)	=2
Completed with incorrect information	Inadequate literature available	Report lacks background information and is ill-informed	1	2	Low (3)	Source literature from elsewhere, expand horizons and call industry experts	1	1	Very Low (1)	=5
	Insufficient data	Collected data is incomplete or unreliable	2	3	Medium (6)	Follow deadlines, vary across multiple software types	1	3	Low (3)	=3
	Loss of contact with academic supervisor	No advice or supervision for research project	1	3	Low (3)	Set regular meetings with supervisor and seek assistance from other staff if need be	1	2	Very Low (2)	=4
Irrelevant to industry	Recommendations and conclusions are based off inconclusive data	Suggestions cannot be used in the industry	1	4	Low (4)	Consultation with supervisor and industry professionals	1	3	Low (3)	=3
	Conclusion and recommendations provide no benefit	The report is rendered useless and irrelevant to industry	1	4	Low (4)	Consultation with supervisor and industry professionals	1	3	Low (3)	=3
	Scope is too narrow or too wide-ranging	The report is too large or small to make an impact in the industry	1	2	Low (2)	Consultation with supervisor and industry professionals	1	1	Very Low (1)	=5

APPENDIX B: GANTT CHART

ID	Task Name	Duration	Start	Finish
1	**MINE4122 Supervisor Consultations**	**101 days**	**Tue 28/02/17**	**Tue 18/07/17**
2	Thesis Meetings	101 days	Tue 28/02/17	Tue 18/07/17
3	**MINE4123 SupervisorConsultations**	**80 days**	**Wed 19/07/17**	**Tue 7/11/17**
4	Thesis Meetings	80 days	Wed 19/07/17	Tue 7/11/17
5	**Research Project Proposal**	**27 days**	**Thu 2/03/17**	**Fri 7/04/17**
6	Finish Draft	23 days	Thu 2/03/17	Mon 3/04/17
7	Raise Questions with Supervisor	1 day	Thu 6/04/17	Thu 6/04/17
8	Finalise Proposal to Submit	2 days	Thu 6/04/17	Fri 7/04/17
9	**Annotated Bibliography**	**15 days**	**Fri 24/03/17**	**Thu 13/04/17**
10	Background Research	6 days	Fri 24/03/17	Fri 31/03/17
11	Reference Selection	10 days	Fri 24/03/17	Thu 6/04/17
12	Prepare Annotated Bibliography Draft	9 days	Thu 30/03/17	Tue 11/04/17
13	Edit Draft for Final Submission	3 days	Tue 11/04/17	Thu 13/04/17
14	**Project Report**	**40 days**	**Fri 24/03/17**	**Thu 18/05/17**
15	Prepare Draft 1	32 days	**Fri 24/03/17**	Sat 6/05/17
16	Literature Review	28 days	Fri 24/03/17	Tue 2/05/17
17	Research of IPCC Methods	8 days	Fri 24/03/17	Tue 4/04/17
18	Research of IPCC Limitations	8 days	Mon 3/04/17	Wed 12/04/17
19	Research of IPCC Parameters	8 days	Mon 10/04/17	Wed 19/04/17
20	Research of IPCC Pit Designs, Scheduling and Geotechnical Limitations	8 days	**Fri 21/04/17**	Tue 2/05/17
21	Finalise Project Progress Report	10 days	Sat 6/05/17	Thu 18/05/17
22	**Project Plan Agreement**	**1 day**	**Thu 1/06/17**	**Thu 1/06/17**
23	Milestone: Complete Project Plan Agreement wih Supervisor	1 day	Thu 1/06/17	Thu 1/06/17
24	**Thesis Examiners' Copy**	**103 days**	**Fri 19/05/17**	**Tue 10/10/17**
25	Make Corrections based off Project Plan Agreement	7 days	**Fri 19/05/17**	Mon 29/05/17
26	Draft 1: Prepare Thesis Draft 1	40 days	**Fri 19/05/17**	Thu 13/07/17
27	Draft 2: Prepare Thesis Draft 2	26 days	Fri 14/07/17	Fri 18/08/17
28	Draft 3: Prepare Thesis Draft 3	14 days	Mon 21/08/17	Thu 7/09/17
29	Create Conceptual Deposit Block Model	7 days	**Fri 19/05/17**	Mon 29/05/17
30	Pit Design and Scheduling (IPCC & T&S)	28 days	Tue 30/05/17	Thu 6/07/17
31	Pit Slope Geotechnical Analysis	7 days	Fri 7/07/17	Mon 17/07/17
32	Economic Evaluation of Mining Methods	13 days	Tue 18/07/17	Thu 3/08/17
33	Comparison Between Mining Methods	21 days	Fri 4/08/17	Fri 1/09/17
34	Data and Results Validation	5 days?	Mon 4/09/17	Fri 8/09/17
35	Edit and Finalize Thesis	24 days	Thu 7/09/17	Tue 10/10/17
36	Presentation Preperation	9 days	Mon 11/09/17	**Thu 21/09/17**
37	**Project Presentation**	**1 day**	**Fri 22/09/17**	**Fri 22/09/17**
38	Milestone: Thesis Topic Presentation	1 day	Fri 22/09/17	Fri 22/09/17
39	**Thesis Revised Copy**	**20 days**	**Wed 11/10/17**	**Tue 7/11/17**
40	Draft 1: Thesis Revised Copy Draft 1	11 days	Wed 11/10/17	Wed 25/10/17
41	Present Draft to Supervisor	1 day?	Thu 26/10/17	Thu 26/10/17
42	Final Draft: Finalize Thesis	9 days	Thu 26/10/17	Tue 7/11/17
43	**AusIMM Conference Paper**	**14 days**	**Wed 11/10/17**	**Mon 30/10/17**
44	Draft 1: Prepare Conference Paper Draft 1	8 days	Wed 11/10/17	Fri 20/10/17
45	Fix Corrections Made By Supervisor	1 day	Sat 21/10/17	Sat 21/10/17
46	Final Draft: Prepare Final Conference Paper	7 days	Sat 21/10/17	**Mon 30/10/17**



Project: Thesis Project Schedule
Date: Fri 26/05/17

Task | Split | Milestone | Summary | Project Summary | Inactive Task | Inactive Milestone | Inactive Summary | Manual Task | Duration-only | Manual Summary Rollup | Manual Summary | Start-only | Finish-only | External Tasks | External Milestone | Deadline | Progress | Manual Progress

APPENDIX C: DISCOUNTED REVENUE

Table 28.

Yearly Undiscounted Revenues

Year	IPCC 26B	IPCC 36B	TS
1	4,476,189.55	-	433,388,007.83
2	6,614,142.34	-	444,222,708.02
3	2,005,097.59	-	530,900,309.59
4	4,819,542.75	-	509,230,909.20
5	125,815,546.68	76,596,109.92	520,065,609.39
6	389,652,622.26	350,974,628.35	514,648,259.29
7	449,017,182.26	318,754,980.61	476,726,808.61
8	530,315,402.56	179,581,910.48	547,152,359.88
9	540,122,536.81	195,795,924.79	574,239,110.37
10	469,522,814.66	116,271,302.13	509,230,909.20
11	449,489,163.40	87,903,733.77	520,065,609.39
12	355,592,413.67	207,189,990.06	568,821,760.27
13	349,681,379.17	319,184,440.97	552,569,709.98
14	410,886,592.36	147,966,175.20	482,144,158.71
15	399,520,486.21	218,096,880.66	498,396,209.00
16	455,949,188.79	371,838,826.19	568,821,760.27
17	528,889,524.45	267,986,889.11	530,900,309.59
18	621,481,161.00	190,165,953.76	552,569,709.98
19	643,217,515.28	337,600,768.69	568,821,760.27
20	654,953,846.83	418,216,597.47	585,073,810.57
21	814,760,433.71	174,274,741.58	595,908,510.76
22	294,504,379.17	254,800,705.22	617,577,911.15
23	752,353,170.66	445,178,300.24	639,247,311.55
24	961,310,851.37	454,714,705.49	660,916,711.94
25	-	329,530,657.58	541,735,009.78
26	-	242,616,093.94	-
27	-	418,842,985.90	-
28	-	494,829,435.73	-
29	-	510,089,495.15	-
30	-	423,682,619.19	-
31	-	387,757,650.59	-
32	-	291,908,749.92	-
33	-	404,136,883.43	-
34	-	470,083,521.81	-
35	-	470,966,575.47	-
36	-	455,105,537.58	-
37	-	536,847,046.35	-
38	-	526,002,044.23	-
39	-	546,721,751.97	-
40	-	759,889,422.25	-
41	-	933,496,627.03	-
42	-	434,219,146.91	-
43	-	955,251,711.50	-
44	-	945,579,302.66	-
45	-	443,398,650.10	-

APPENDIX D: TRUCK WORKERS

Table 29

Yearly Truck workers.

Year	*IPCC 24*	*IPCC 34*	*TS*
1	24	24	24
2	36	36	24
3	48	48	30
4	18	54	30
5	18	18	36
6	18	18	36
7	24	24	42
8	24	24	48
9	24	30	54
10	30	30	60
11	30	18	66
12	30	18	72
13	18	24	78
14	18	24	84
15	18	30	90
16	24	30	96
17	24	18	102
18	24	18	108
19	30	18	114
20	30	24	120
21	30	24	126
22	18	24	132
23	18	24	138
24	24	30	138
25	24	30	138
26	30	30	-
27	-	30	-
28	-	18	-
29	-	18	-
30	-	18	-
31	-	24	-
32	-	24	-
33	-	24	-
34	-	24	-
35	-	30	-
36	-	30	-
37	-	30	-
38	-	30	-
39	-	18	-
40	-	18	-
41	-	18	-
42	-	24	-
43	-	24	-
44	-	24	-
45	-	24	-