



FACULTY OF ENGINEERING AND TECHNOLOGY

DEPARTMENT OF MINING AND GEOLOGICAL ENGINEERING

A THESIS REPORT ENTITLED:

**ECONOMIC EVALUATION OF MATERIALS HANDLING SYSTEMS IN
DEEP OPEN PITS: A CASE STUDY OF JWANENG MINE**

BY

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**SUBMITTED IN FULFILLMENT OF THE REQUIREMENTS FOR THE AWARD
OF THE DEGREE OF MASTER OF ENGINEERING IN MINING
ENGINEERING**

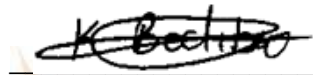
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June 2021

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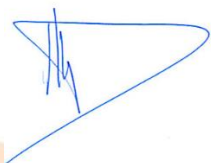
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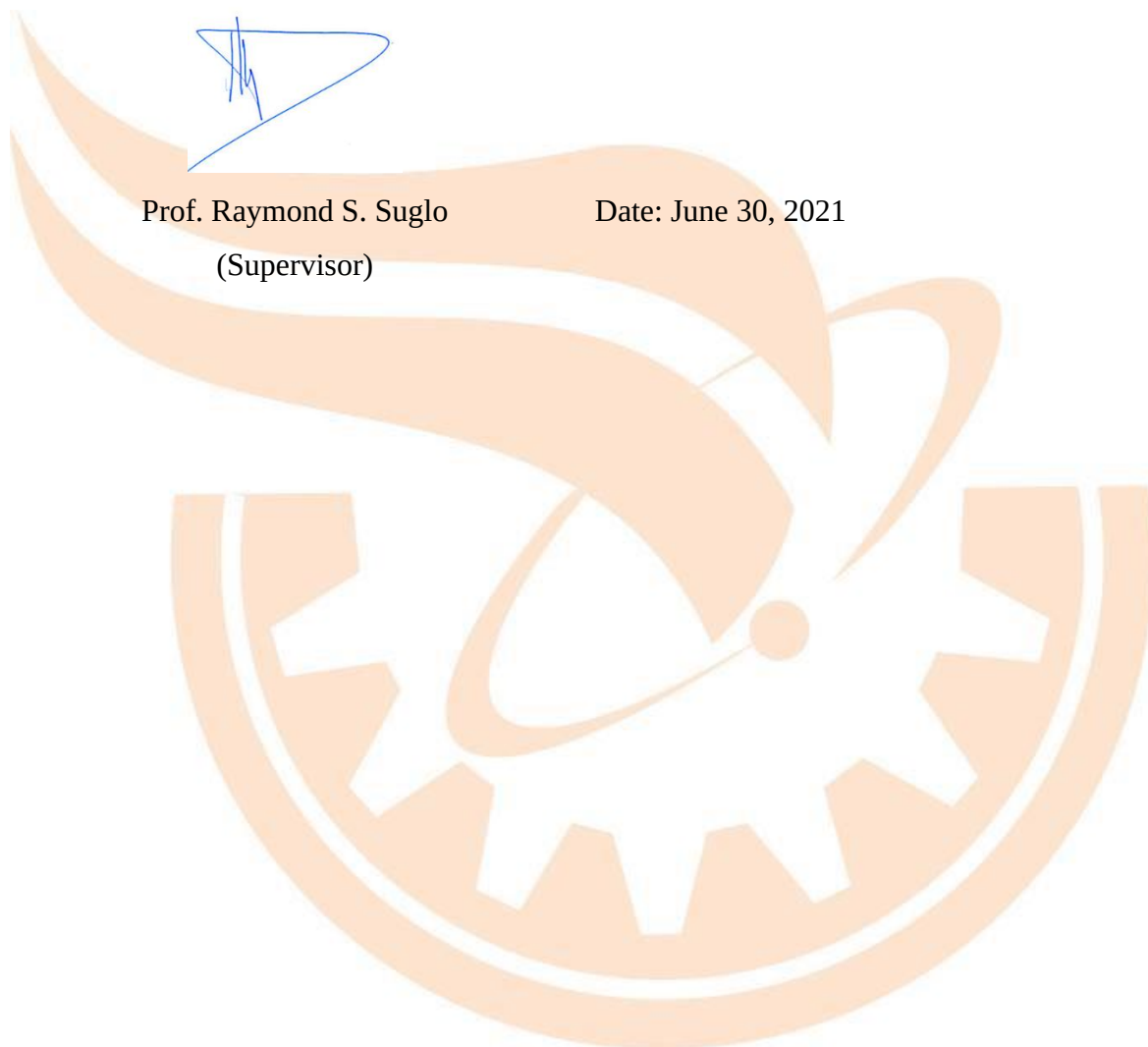
CERTIFICATION

The undersigned certifies that he has read and hereby recommends for acceptance by the Faculty of Engineering and Technology a dissertation/thesis titled **Economic Evaluation of Materials Handling Systems in Deep Open Pits: A Case Study at Jwaneng Mine** in fulfilment of the requirements for the degree of Master of Engineering in Mining Engineering of the BIUST.



Prof. Raymond S. Suglo
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Date: June 30, 2021



ABSTRACT

Truck-shovel haulage system (TSHS) is the commonly used haulage system in open pit mines due to its flexibility and economy. In-pit Crushing and Conveying (IPCC) haulage system has advanced and is an alternative to TSHS in deep open pit mines. The objectives of this research are to evaluate the use of the TSHS and IPCC in materials handling, conduct economic analysis of the TSHS and IPCC systems for deep open pit mines and suggest optimal ways of handling materials from them.

The unit haulage costs, net present value (NPV), internal rate of return (IRR), discounted payback period (DPP) were calculated while sensitivity and risk analyses was conducted on the two materials handling options. The results show that the IPCC system has lower unit costs than TSHS; the NPV of IPCC system is BWP 282.25 billion while that of the TSHS is BWP 278.83 billion; the DPP of the TSHS and IPCC systems in ore and waste are 2.94 months and 2.13 months respectively while the IRR of the TSHS and IPCC systems in ore and waste are 147% and 185% respectively. As the NPVs of both systems are positive, with very short DPPs and high IRRs, the systems are considered to be economically viable. However, the IPCC is considered more economically viable than TSHS. The NPV of IPCC is less sensitive to variations in the input variables compared to TSHS. It is recommended that mine planners should consider using IPCC haulage system in deep open pits.

DEDICATION

I dedicate this thesis to my amazing father, Ketlogetswe Bodiba. Your love has always been an inspiration to me to do better. Thank you, dad.



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CHAPTER 1

INTRODUCTION

1.1 Background of the Problem

Materials handling significantly contributes to the capital and operating costs of an open pit mine. According to Alberts (1984) transportation of ore and waste constitutes up to 40% of the total mining costs. This has prompted mine planners to opt for the best ways of handling materials when designing mines. The method to be used has to be cost-effective, environmentally friendly and yet meet the production demands of the mine. Dzakpata *et al.* (2016) argued that the traditional Truck Shovel Haulage System (TSHS) is used in 95% of open pit mines worldwide due to its flexibility and economy of scale. TSHS is the only method used in surface mines in Botswana. It is a very reliable and flexible haulage system with a reasonable capital expenditure (CAPEX) but has a very high operating expenditure (OPEX). TSHS can cost up to 50% of the total mining costs (Mohammadi *et al.*, 2011). The system involves an excavator (mainly a shovel) loading the blasted materials (ore or waste) into dump trucks in the pit, the trucks hauling the materials from the pit through ramps and haul roads, dumping it at the crusher, stockpile or waste dump sites and returning back to the shovel within the pit for another load. The cycle time of the truck is the basic parameter that affects the system's productivity. Therefore, TSHS requires frequent inspection and maintenance of haul roads, their widths, rolling resistance, gradients, curvatures and cross-over locations to improve the system's productivity. Dust caused by the movement of trucks on the haul roads is usually controlled by spraying the haul roads frequently with water or other dust suppressants during the shifts.

Unfortunately, most open pits get deeper with time as mining progresses, causing the companies to mine at greater depths and with larger scales of operations (Paricheh *et al.*, 2018). Jwaneng Mine is the deepest open pit mine in Botswana. It is currently mining at a depth of 470 m. The increase in depth and high stripping ratios require more trucks to be added to the fleet periodically to meet the desired production targets. A large fleet of trucks requires more operators and maintenance staff, increase in diesel consumption and noise. Large truck fleet sizes also produce large amounts of noxious gases such as carbon dioxide (CO₂), sulphur oxide (SO₂) and other gaseous and particulate pollutants (Dean *et al.*, 2015).

Therefore, there is the need to look for new materials handling methods in deep open pit mines that will lead to reduced personnel and equipment requirements, reduce the fuel consumption and emission of noxious gaseous and particulate matter. Such methods include In-Pit Crushing and Conveying (IPCC) systems, hydraulic (slurry) and pneumatic materials handling systems. The IPCC, slurrying and pneumatic transport systems require crushing of the blasted materials within the pit before it can be transported to the required destination. Thus in-pit crushing is an additional cost to the operations as crushers have to be installed within the pit. Hydraulic transport systems use a fluid (mainly water) under pressure for transportation of the materials while pneumatic systems use compressed air as the fluid for transporting the materials within large diameter pipes (Nehring *et al.*, 2018). This research focuses on the IPCC system because it offers the best alternative to the conventional truck-shovel haulage system.

The IPCC system comprises a combination of feeding, crushing, conveying and discharging systems (Osanloo and Paricheh, 2019). An excavator alone or in combination with trucks feed the material into either a fully mobile or semi-mobile crusher in the pit depending on the crusher's feeding system, mobility or shape. The material is then crushed into suitable sizes within the pit and fed into belt conveyors for transportation out of the pit and disposed of by spreaders (for waste) or stackers (for ore) at the discharge ends. The particle distribution of the material transported by a conveyor should be such that the largest material does not exceeds one-third of the belt width (Dean *et al.*, 2015). To choose the suitable crusher for the materials and conveyors, it requires a thorough knowledge and understanding of the geological characteristics and geometry of the orebody and host rock.

Since IPCC's development in the 1950s in Germany, the system has been installed by over 200 mines worldwide (Koehler, 2003). Table 1 summarises some of the mines in the world that converted from TSHS to IPCC system due to the escalating operating and maintenance costs of TSHS as the mines became deeper. Table 1 also gives the production levels achieved using the IPCC system, the type of in-pit crusher used and the power requirements of the whole system (Mohammadi *et al.*, 2011).

Table 1 Mines that Use IPCC Materials Handling System

Name of Mine	Location	Production Target (t/day)	Crusher Type	Power (kW)
Sierrita Copper Mine	Sahuarita, Arizona	180,000	3 semi-mobile gyratory	14,000
Bingham Canyon Copper Mine	Utah, USA	70,000	1 gyratory	12,900
Island Copper Mine	Port Hardy, Canada	43,000	1 fully mobile gyratory	9,700
Highland Valley Copper	Highland Valley	120,000	2 gyratory	13,500
Sishen	Northern Cape, South Africa	60,000	1 semi-mobile gyratory	11,000

Source: (Mohammadi *et al.*, 2011)

For large open pit mines, the benefits of IPCC over TSHS are (Jeric and Hrebar, 1997):

- Continuous haulage of materials.
- Increased mine and road safety due to fewer moving vehicles on the haul roads.
- Reduced road maintenance requirements (water trucks, etc.).
- Reduced dust generation.
- Lower labour requirements.
- Electrical power used in IPCC is less costly than diesel.
- Reduced emission of greenhouse gases (GHS) and noise generation.

Though this system is considered a potential alternative to TSHS, it does not entirely eliminate the use of dump trucks except in a fully mobile crusher system which is fed directly by the excavator. The disadvantages of IPCC compared to TSHS are (Dean *et al.*, 2015):

- IPCC system requires a significant up front capital expenditure (CAPEX) because the crusher and full length of the belt conveyor have to be bought and installed to start production whereas trucks can be bought in stages as production progresses.
- The system reduces the ability of a mine to easily switch mining to a different zone to adopt to unforeseen changes in market conditions or geology.
- Since the system is a series of connected systems, a shutdown in any component of the system affects the entire production.

While the waste materials do not require crushing for truck transport, it requires size reduction for conveyor belt transport resulting in an additional cost burden for handling waste in IPCC (Tutton and Streck, 2019). Furthermore, the need for relocating the crusher periodically and extension of belt conveyor systems within the pit is expensive and require frequent system shut downs for periods of 2 to 3 days (Mohammadi *et al.*, 2011).

1.2 Statement of Problem

Botswana is a mining country with eight large scale operating mines as of 2020. Four of them are open pit diamond mines, three are underground mines that mine gold, copper and coal while one is an open cast mine mining coal.

All the open pit mines in Botswana use rear dump trucks to haul the materials (waste and ore) from the pits to the crushers for processing, to stockpiles and to waste dump sites. As the depth of the pits increase with time, the trucks take longer times to haul the materials from the loading points in the pit to either the crushers or dump sites (off-pit). The longer truck cycle times result in low production per given time. To rectify the situation and achieve the targeted production, more trucks have to be used to handle the materials. Haulage of materials by trucks over long distances result in a lot of fuel consumption, generation of dust, noise and noxious gases.

Conveyor belt systems are another option for handling materials out of pits. But this requires flatter pit slope angles and the initial installation cost is very high. Besides, it is difficult to move the conveyor belt in multiple bench operations to meet the advance in the different mining faces.

Optimising the use of TSHS and IPCC will provide a framework for minimising the unit cost of moving materials (ore and waste) in deep open pit mines. To achieve this, it requires an understanding and a comprehensive evaluation of the economic, social, safety, environmental, and technical factors when using the two systems. Therefore, the determination of the optimum production system between TSHS and IPCC will be a significant milestone to deep open pit mines worldwide. In this research, the capital and operating costs of TSHS and IPCC system will be computed and compared to come up with the better one.

1.3 Objectives of Research

The aim of this research is to analyse the operation and economics of TSHS and IPCC systems for a deep open pit mine (Jwaneng Mine) and suggest an optimal way to transport material off the pit.

The objectives of this study are to:

- Evaluate the use of the TSHS and IPCC in materials handling in deep open pit mines like Jwaneng Mine.
- Assess the production performance of the TSHS and IPCC systems at mines.
- Conduct economic analysis of the two materials handling systems with respect to the pit at Jwaneng Mine.
- Suggest an optimal way to handle materials from the pit.

1.4 Outcomes of Research

The outcome of this research will lead to an optimal materials handling system in deep open pit mines.

1.5 Methods Used

The TSHS and IPCC systems were evaluated through literature review and visits to mines. Data was collected from the mine through visits and available mine production reports. The number and type of production equipment used at Jwaneng Mine together with the mine's production targets and other parameters were used to determine the production performance of the two haulage systems. Economic analysis was conducted on the two options using economic parameters like the NPV, IRR and DPP. Sensitivity and risk analyses were also conducted on the NPV of the two systems using Monte Carlo Simulation with the @RISK software.

1.6 Structure of Thesis

Chapter 1 contains the background, statement of the problem, research objectives and outcome, methods used and the importance of this research. Chapter 2 has the general information about Jwaneng Mine. It covers the mine's geology, mining operations and equipment used. Chapter 3 has the literature review while Chapter 4 has the data collection and analysis. Chapter 5 contains the conclusions and recommendations of this work.

CHAPTER 2

GENERAL INFORMATION ABOUT JWANENG MINE

2.1 Introduction

Jwaneng Mine is the richest diamond mine in the world by value as of 2021. The mine became fully operating in August 1982. Jwaneng Mine is owned by Debswana and contributes about 60 - 70% of Debswana's total revenue. Currently, Jwaneng Mine is mining at a depth of about 456 m as of 2020. The resource consists of three separate volcanic pipes/vents namely the north, south and centre pipes (two additional small kimberlite bodies have also been intersected within the pit). The pipes are said to have erupted through Transvaal strata and the overlying Karoo sediments 245 million years ago (Anon., 2019a). The pit hosts a variety of lithologies as shown in Figure 1.

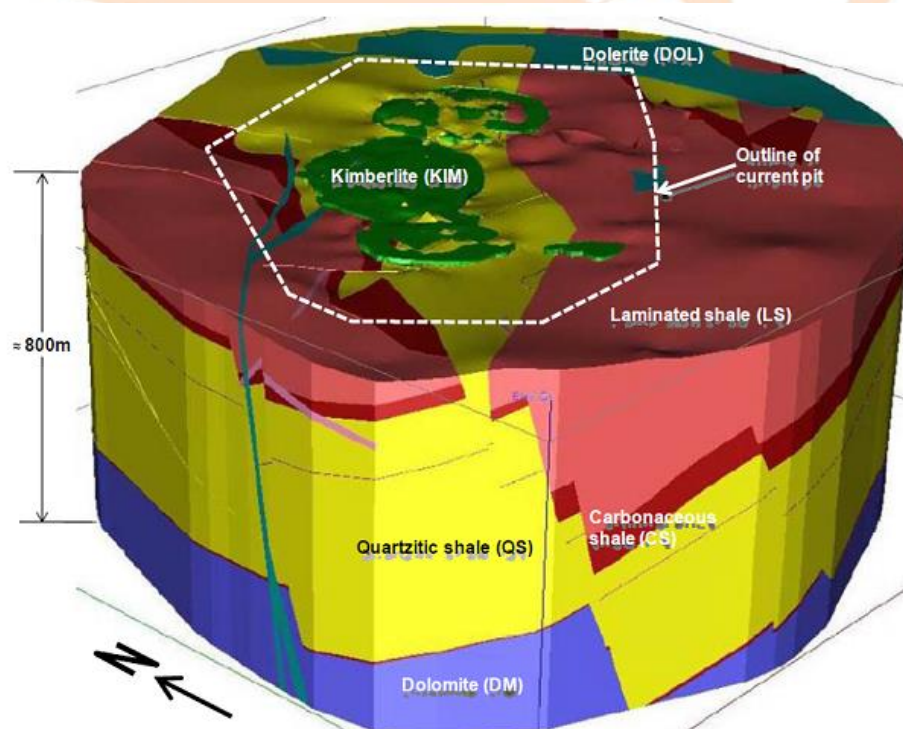


Figure 1 Lithological Domains in Jwaneng Mine DK2 Pit (Godie, 2014)

Production at Jwaneng Mine varies from 12.5 to 15 million carats per year as summarised in Table 2. This rate of production, combined with the very high quality of the diamonds, continues to make Jwaneng Mine the richest diamond mine in the world, by value.

Table 2 Production for Jwaneng Mine Operations from 2016 - 2018

Year	Total Diamonds Recovered (million carats)	Total Volume Moved (million tonnes)	Total Waste Mined (million tonnes)	Total Ore Treated (million tonnes)
2016	11.97	128.05	109.42	7.99
2017	11.86	120.10	105.13	8.42
2018	11.90	118.99	100.79	9.46

Source: (Malela, 2018)

In addition to achieving its annual production targets, Jwaneng Mine maintains an excellent safety record. Since 1984, the mine has achieved one of the lowest disabling injury incident rates in Botswana (Anon., 2019a). Over the years, there has been some potential hazards in the mine. Since the mine started mining Cut 8, a waste mining expansion project to expose ore, it has experienced a lot of loose ground falls as the area is very weak. In 2012, part of the pit wall collapsed with one fatality. As a result, there has been an upgrade on the ground monitoring system used and production is only carried out during the day at Cut 8 as a safety measure. The mine also experiences mini pit wall failures from the areas that were mined many years ago, hence the installation of ground fall safety nets to act as rock fall catchments. Wet ground and water build up has always been a challenge during rainy seasons because it reduces traction of the trucks and can cause accidents. As a safety measure, the travel speed of trucks and the number of trucks in the pit is reduced when it is raining heavily, and full production only commences after the rain has ceased.

Occasional dosing off by some truck drivers during night shifts due to fatigue is dangerous to themselves and other people in the pit as the trucks can slide off the lane to hit another truck or topple over the benches. Another fatality was reported during the afternoon shift of May 2018 in which a dump truck was reported to have driven through the safety berm and stopped on the second one at D5 dump (Malela, 2018). To avoid these accidents, tea breaks have been introduced during shifts and catchment banks are always maintained on the sides of haul roads and dumps.

2.2 Geology of Jwaneng Mine

The three steep-sided pipes of the Permian Jwaneng Kimberlite are filled to a depth of about 600 m and more. They are dominated by two contrasting textural types of kimberlite,

interpreted as resedimented volcanoclastic kimberlite and dark pyroclastic kimberlite (Webb *et al.*, 2003). The lithology and structural geology of the mine is covered in the next section.

2.2.1 Lithology

The Jwaneng open pit cuts through a package of paleoproterozoic-aged sedimentary rocks belonging to the Pretoria Group of the Transvaal Supergroup, which are underlain by Chuniespoort Group carbonates within 100 - 200 m below the current pit bottom. The rock strata dip between 10° and 40° NW but are also tilted and displaced within structure-bound blocks (Godie, 2014). The types of lithology have been grouped into the stratigraphic columns as summarised in Tables 3 and 4.

Table 3 Lithology Types in DK2 Pit

Stratigraphic Name	Rock Type (Code)	Typical Thickness (m)
Kalahari Sequence	Sand & Calcrete (CS)	55 - 60
Timeball Hill Formation	Laminated Shale (LS)	“Residual”
Lower Timeball Hill Formation	Carbonaceous Shale (CS)	30
Rooihogte Formation	Quartzitic Shale (QS)	135
Chert Pebble Conglomerate	Bevets	0 - 4
Rooihogte Formation	Quartzitic Shale (QS)	375
Lower Rooihogte Formation	Carbonaceous Shale (CS)	10
Malmani Subgroup	Dolomite (DM)	“Residual”

Source: (Godie, 2014)

2.2.2 Structural Geology

The structural geology appears to be dominated by NE-SW striking faults, with a strong normal dip-slip shear sense component. The faults are down-thrown to the SE and compartmentalise the shallow to intermediate NW dipping strata into discrete structural domains, which make useful geotechnical domains. To understand the structural geology, the pit was divided into zones (Godie, 2014). The different zones show the rock mass properties shown in Appendix 1.

Table 4 Description of Rock Types Found in the DK2 Pit

Geology Code	Name	Description	Source
CS	Carbonaceous Shale	A thinly blended shale that is rich in carbon, pyrite and organic matter. They are generally black to grey in colour and results from the anaerobic decay of organic matter.	(Senyasi <i>et al.</i> , 2016)
DM	Dolomite	A carbonate sedimentary rock composed primarily of the mineral dolomite. Dolomite is found in sedimentary basins worldwide. It is thought to form by the post depositional alteration of lime mud and limestone by magnesium-rich groundwater. It has colour ranges of white to gray and white to light brown (although other colours such as red, green, and black are possible) and is soluble in water.	(King, 2020)
DOL	Dolerite	Dark-coloured, medium-grained, basic igneous rock with the same composition as basalt and gabbro, but with a texture of inter-grown plagioclase and pyroxene.	(Palmström, 2013)
LS	Laminated Shale	A fine-grained sedimentary rock that forms from the compaction of mud. It is fissile and made up of many thin layers	(King, 2020)
QS	Quartzitic Shale	A fine-grained sedimentary rock that consists a lot of quartz mineral fragments	(King, 2020)
KM	Kimberlite	A dark-coloured, heavy, often altered and brecciated (fragmented), intrusive igneous rock that contains diamonds in its rock matrix. It has a porphyritic texture, with large, often rounded crystals (phenocrysts) surrounded by a fine-grained matrix (groundmass). It is a mica peridotite, and its most abundant mineral constituent is olivine. It is formed from the cooling of molten magma.	(Sampaolo, 2020)

2.3 Mining Operations

Open pit mining operations involve site preparation, drilling, blasting, loading and hauling of the ore to the processing plants and the waste to the waste dumps.

2.3.1 Site Preparation

The area where blasting is to take place has to be prepared to the required standards in terms of grade, floor profile and final wall exposure. The benefits of site preparation include blast hole drilling accuracy, improved safety of workers and equipment on the bench, improved drilling efficiency and tyre protection.

2.3.2 Drilling

Jwaneng Mine uses both diesel and electric powered drills. The diesel powered drills are the Atlas Copco Pit Viper 351 and the small 165 mm contractor drills used for pre-split holes. The electric powered drills include the P&H XPC320 and P&H XP100 drills. All the large drills are owned by the mine and they are used to drill 250 mm diameter holes. Blastholes are usually drilled to 16 m. Table 5 summarises the available drills owned and used by Jwaneng Mine.

Table 5 Types of Drilling Equipment in Jwaneng Mine

Type of Drills	Number
Drill - Pit Viper 251/311 mm – Cut 7	1
Drill - Pit Viper 251/311 mm – Cut 8	3
Drill - Pit Viper 251/311 mm – Cut 9	3

2.3.3 Drilling and Blasting Operations

Jwaneng Mine has contracted AEL Mining Services to handle its drilling, charging and blasting operations. Explosives used include emulsion, ammonium nitrate prills (ANPP) and blasting accessories (electronic delay detonators, boosters, splitex and pentolite cord) which are stored in magazines located in the mine. The emulsion and Ammonium Nitrate Prills Porous (ANPP) are transported in Mobile Manufacturing Units (MMUs) to the blasting sites and mixed onsite.

Blasting design parameters are handled by the Debswana crew. These are designed taking into account the presence of critical structures such as faults, the free face and the high walls. Jwaneng Mine uses coarse tailings (19 - 37 mm crushed aggregate of waste) as stemming material in the blast holes that are in ore.

2.3.4 Loading and Hauling

After blasting, the broken material is loaded into haul trucks and transported to different destinations. The waste material is transported to waste dumps, contact materials to stockpiles and the ore to the crusher and dressing materials to the loading and dumping sites. Currently the mine has three active mining phases, Cut 7 in ore while Cuts 8 and 9 are in waste. Jwaneng Mine has installed a dispatch fleet management system supplied by Modular Mining Systems to help operators, supervisors, dispatchers and managers to effectively and efficiently run and monitor the mining and materials handling equipment to achieve production goals. It uses Global Positioning System (GPS) to communicate various transactions on front line machines to a central computer located at the dispatch house. The Dispatch Modular System is used to assign haulage equipment to loading equipment and to keep track of production. This system prevents loader-truck mismatch, over trucking as well as bunching. Equipment used in the load and haul operations at Jwaneng Mine are summarised in Table 6.

Table 6 Mining and Support Equipment Used in Jwaneng Mine

Loaders	Number
LeTorneau 1850 25 m ³	1
P&H 2800XPB 38 m ³	1
Demag PC8000 35 m ³	2
Bucyrus RK430 28 m ³	1
WL268 (L1850 /Letounouer/30.53 m ³)	1
Excavator Loader (CAT/6040/22 m ³)	2
Liebherr Excavators (9350)	3
Rope Shovel - (495HR2/58 m ³)	2
Trucks	
CAT 793 (240 tons)	5
Komatsu 930E (300 tons)	60

Support Equipment	
Track Dozers	7
Komatsu WD600 Wheeldozer	5
CAT 994 Front End Loader	1
CAT 16 M Grader	3
CAT 773 80,000 litre - Water bowser	3
Komatsu HD465 50,000 litre - Water bowser	2
CAT 773 50,000 litre - Diesel bowser	2
CAT CS 533 EXL - Compactor	1
CAT 988H - Cable Handler	2
PC300-8 Rock Breaker	1
GD825 Grader	4
872G Grader	1
Liebherr 9350 Excavator	3
CAT 785D Trucks	3
Komatsu WD600 Loader	2
CAT 785D 100,000L Water bowser	1

CHAPTER 3

LITERATURE REVIEW

3.1 Introduction

This chapter contains the survey and evaluation of the available literature related to the economic evaluation of in-pit crushing and conveying, and truck-shovel haulage systems in open pit mining.

3.2 Evolution of Open Pit Mining

Open pit mining is the most common mining method used throughout the world for mineral extraction. Open pit mining is economically viable when the deposits of the commercially useful minerals are found near the surface (i.e. ≤ 500 m deep). Although it is often associated with metallic ores (e.g. Palabora Copper Mine, Mamatwan Mine and Sishen Iron Ore Mine all in South Africa), it can be used to mine any near-surface deposit such as the diamond bearing kimberlite pipes at Venetia (South Africa), Jwaneng and Orapa Mines in Botswana.

The largest open pit copper mine by excavated volume is the Chuquibambilla Copper Mine in Northern Chile. Its history goes back to the 19th century. Daily production rates have increased over the years as better mining systems were developed and improved with time. Mining equipment have evolved from being highly labour intensive to being powered by diesel and electricity lately. Also dump trucks have grown in capacity from 5 tonnes to 500 tonnes over the years (Anon., 2016).

To expose and mine the ore, it is generally necessary to excavate large quantities of waste rock and send it to the waste dump sites. This operation has to be done at the lowest cost possible while attaining maximum profits and safety. This also makes the selection of physical design parameters and scheduling of ore and waste extraction to be complex engineering decisions of great economic significance. The design and planning of an open pit mine is based on economics and constrained by certain geological, mining and environmental aspects (Anon., 2020). The introduction of improved borehole drilling techniques, data logging techniques, computers and modelling software in open pit planning and design have made this exercise a lot easier. Combining these innovations with detailed

information on capital and operating costs, mine planners are able to produce short, medium and long term mine plans.

3.2.1 Open Pit Mine Design

Traditionally, the optimisation of an open pit mine design consists primarily of defining the ultimate pit limits which, in turn, define what will eventually be removed from the ground, and dividing up the pit into manageable volumes of material often referred to as pushbacks, cutbacks or phases (Meagher *et al.*, 2014). Different techniques are used to produce the orebody model of what is predicted to be on the ground. This model is then used to produce the ultimate pit limit. A pushback is an aggregation of mining blocks and its design is to guide the sequence of extracting an orebody from the point where mining operations begin until where it stops (Consuegra *et al.*, 2013). Designing pushbacks is such that each has its own access system or link to the access system of the previous phase. Each has to have enough width and length to support the production capacity of the phase (Tutton and Streck, 2019). The pushbacks are designed from the orebody model that remains within the ultimate pit limit.

Designing an open pit mine, as guided by the ultimate pit limit, requires knowledge of design parameters by material type and/or geotechnical factors. Some parameters like the bench height, bench slope angle, bench width, grade of ore, ramp width and gradient, minimum radius for curves and inter-ramp angle limits are part of the outcomes of the overall open pit design (Poniewierski, 2018). Pit design has two stages which are short-term and long-term pit designs. Long-term pit design is to ensure that the problems or demands of operation can be dealt with within the operating level. It is important to take into consideration the laws and regulations of the local jurisdictions of the mine guiding certain pit designs (Poniewierski, 2018).

3.2.2 Open Pit Mine Production Planning

Mine planning is the planning of the extraction sequence over a particular time horizon. It may be for one year, 5 years but typically over the life of the mineral deposit. Mine planning involves long-term, short-term and operating plans. Long-term plans look into the economic potential of the mineral deposit (usually ≥ 10 years). It involves the planning of waste dumps, treatment plants and tailings dams. Short-range plans, which are usually of a

duration of one to five years, provide enough information to predict future production and capital expenditure. These predictions are constrained by physical, geological, operating and legal policies. Operating planning gives a guide to orderly mining to achieve the present and short-range objectives under the present economic and technological conditions and policy constraints. Operating planning is also called actual production planning (Johnson, 1968).

Open pit production planning can be defined as specifying the sequence of extracting ore and waste blocks from the mine to give the highest Net Present Value (NPV), subject to a variety of production, grade blending and pit slope constraints (Osanloo *et al.*, 2008). It is aimed at finding an optional solution to mine block sequencing, ore-waste discrimination by cut-off grades and determining the production rates (Kumral, 2012). Production planning uses the exploration drilling pattern, geological conditions and the size of mining equipment to determine the size of blocks in the block model (Rahmanpour and Osanloo, 2016). The geological characteristics of each block are then given using any available estimation technique. To determine the economic value of each block, the mine planner needs to look into the commodity selling price, operating costs and overall recovery (Rahmanpour and Osanloo, 2016). At the end, mine planners need to decide on which blocks are viable to mine, when they are to be mined (if viable) and how best to process a block once mined so as to maximise the mine's NPV.

3.3 Equipment Selection and Scheduling in Open Pit Mining

Mining equipment selection refers to choosing the right mining fleet to carry out the materials handling process at the optimal unit costs. Given the mine plan, the ultimate goal is to select mining equipment such that the overall cost of materials handling is minimised (Burt *et al.*, 2005). Such fleet must be able to cope with the changing nature of the mining operations. By selecting the right equipment, the mine saves time and energy needed to mine and haul the materials and also ensures that the production demands are fulfilled as per the capacity of the processing plant(s). During equipment selection, it is very important to take into account the equipment availability, its maintenance needs, compatibility, volume capacity and its cost per unit production (Burt and Caccetta, 2013). In open pit mining, there are a number of materials handling systems to choose from. These include the traditional TSHS, IPCC, hydraulic (slurry) and pneumatic transportation systems.

When selecting mining equipment, it is very vital to understand and analyse the operating performance of the equipment and ensure that the utilisation of the selected equipment is maximised through optimal scheduling. Equipment scheduling involves planning the tasks of each equipment and their maintenance. Looking at the fleet specifications and production demands (i.e. materials that need to be moved periodically) the schedules of when and where the fleet should be working is drawn. They can also help to determine how long the equipment can complete the task.

Maintenance is one of the major operating cost items for equipment. It can constitute about 30 - 50% of the overall haulage costs in an open pit mine when using the traditional truck-shovel system (Topal and Ramazan, 2010). A new system using Mixed Integer Programming (MIP) has been used to compose an annual schedule for a fixed fleet of mining trucks over years to minimise the maintenance costs. This model uses the truck's age, maintenance costs and required operating hours to achieve production target to produce an optimum truck schedule. Maintenance costs can be obtained from historical data, from other operations using the same equipment or from the manufacturer's estimated values (Topal and Ramazan, 2010).

3.4 Materials Handling in Open Pit Mines

Materials handling is the process of moving and storing materials at the lowest cost through the use of proper methods and equipment (Ray, 2008). Its cost can be up to 40% of the total production cost. Usually, the cost of owning and maintaining the equipment as well as operating the system are the main contributors to the high cost of materials handling. Materials handling can be classified into three functional scopes. These are bulk handling, industrial packaging and storage or warehouse handling. Bulk handling is particularly relevant in the mining industry. According to Ray (2008) the aim of a well-designed materials handling system is to improve production efficiency by delivering the right quantity of materials at the right place and time economically, cut down indirect labour costs, reduce damage to the materials during transportation and storage, minimise accidents during the materials handling process, reduce the overall cost of handling and meet the demands of the customer by supplying the materials in a manner convenient for end use. This requires the mine planners to choose a system that is more suited to the mine's needs. The system has to deliver and meet the production needs (production rate and material size), be less costly, safe and ensure a conducive environment. It is possible to use more than one

materials handling systems or change the system in the future. The traditional truck-shovel haulage system (TSHS) is the most common haulage system in mines worldwide. However, the use of the IPCC as the alternative to TSHS system has been increasing over the years. The first IPCC system was introduced in 1950s.

3.4.1 Truck-Shovel Haulage System

The truck-shovel haulage system (TSHS) is the most common haulage system used in open pit mining. It involves the use of mainly rear dump trucks to transport broken materials from the loading points in the pit to the dump sites for waste or crusher for ore. Crawler-mounted excavators like rope shovels, hydraulic shovels and backhoes are used to load the blasted material into dump trucks. Trackless front end loaders can also be used when a regularly scheduled loading shovel is shut down for an extended period (Kennedy, 1990). These work in combination with the trucks. Owolabi (2019) states that the trucks vary in carrying capacity from 36 tons to over 300 tons. He further notes that the size and cost of operating dump trucks is directly proportional to their truck capacities while the speed at which the truck travels is inversely proportional to its capacity. The productivity of a rear dump truck is a function of all the factors that influence dump truck cycle time. These factors include truck rolling and grade resistance, type or make of truck, haul road widths and conditions (e.g. curves and super-elevations, intersections) and the dump truck operator's efficiency (Dey *et al.*, 2017). The time taken by a single dump truck to travel from the loading point to the dump site and back is referred to as its cycle time. The cycle time involves loading the truck by a shovel, hauling of the loaded materials, queuing at the dumpsite and tipping, travelling back to the loading point empty and queuing at the loading point (see Figure 2). A dump truck's productivity is also affected by its availability and dumper-loader match.

Transportation costs differ depending on the pit's configuration and geographical location. Tutton and Streck (2019) state that the cost of the haulage component in open pit mining is about 45% of the operating costs on the life of a mine basis and about 40 - 50% of capital costs. However, Ercelebi and Bascetin (2009) note that TSHS is the largest item in the operating costs, constituting 50 - 60% of the total cost in surface mining operations.

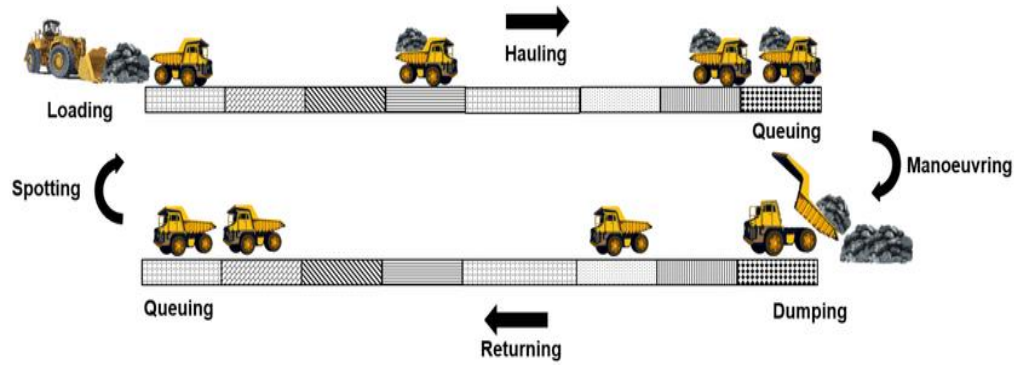


Figure 2 Cycle of Operation of Truck-Shovel Haulage System (Soofastaei *et al.*, 2016)

Figures 3 and 4 show the distribution of the typical equipment's capital and operating costs in large open pit mines using conventional shovel-truck mining methods respectively.

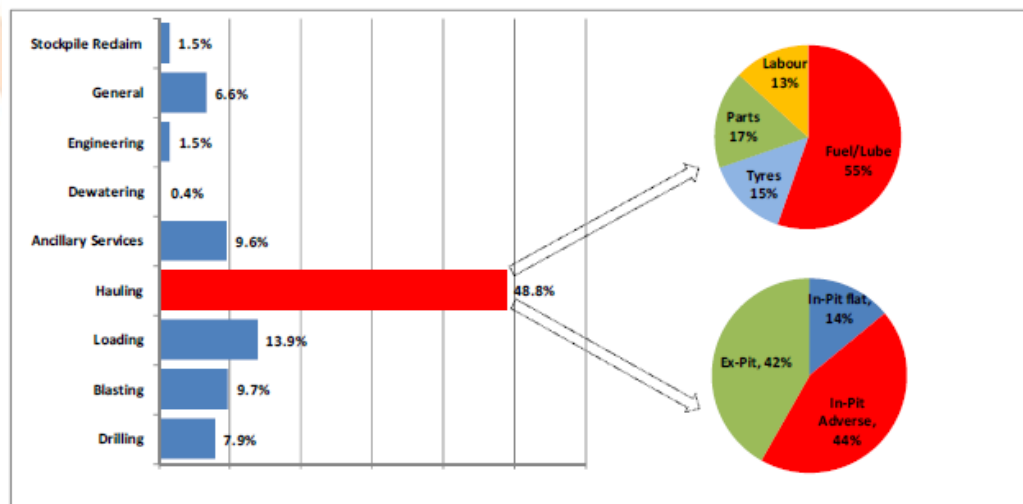


Figure 3 Typical Equipment Capital Cost Distribution in Large Open Pit Mine (Tutton and Streck, 2019)

From Figure 3 the haulage cost alone constitutes more than 15% of CAPEX. This is due to high cost of fuel used by the dump trucks with the remaining 45% being contributed by the cost of parts, labour and tyres. From Figure 4, the costs of fuel, labour, tyres and parts increase with pit depth. This is due to the fact that travel over longer haul distances result in a lot of diesel consumption by dump trucks. Also, to meet production demands, more trucks have to be added to the fleet as the mine gets deeper which results in even increased fuel consumption, more maintenance costs and personnel requirements.

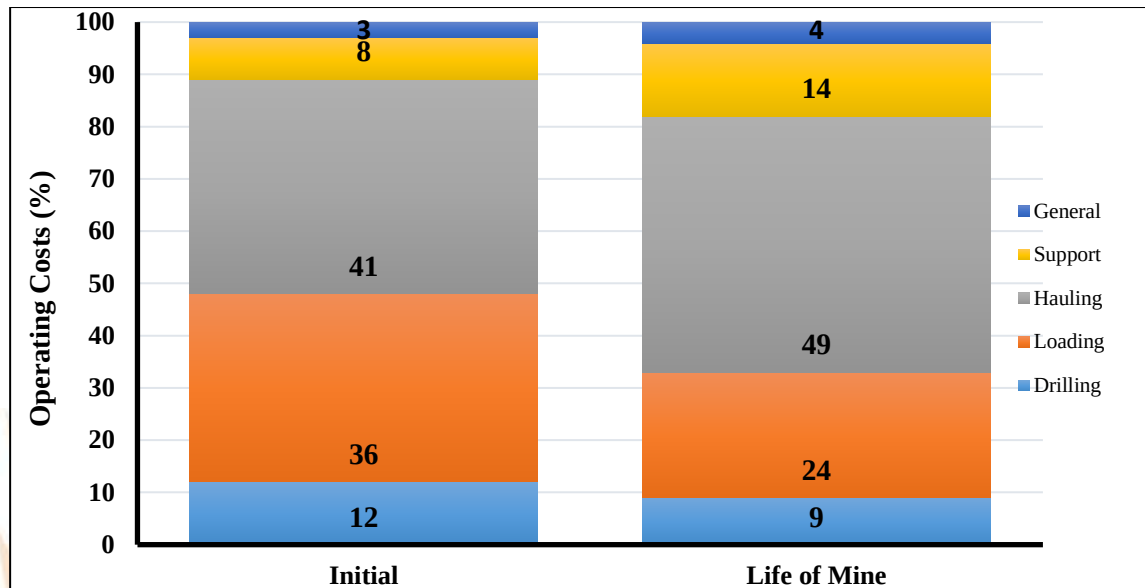


Figure 4 Distribution of Operating Cost in Large Open Pit Mines (Tutton and Streck, 2019)

Dust and noise production, and safety risks to operators and workers in the pits also increase as the pits get deeper with the increase in the number of equipment and workers. As the depth of the pits increase leading to longer haul distances of the trucks, one option available to mine operators is to use larger capacity excavators and trucks to meet their production targets and also optimise the productivities of their unit systems. However, the ever-increasing prices of diesel on the market added to the higher consumption rates of fuel by the large haulage trucks and excavators lead to higher operating costs of the TSHS.

Auxiliary equipment for TSHS: Auxiliary equipment also constitute part of the system's capital expenditure (CAPEX) and operating expenditure (OPEX). They are not responsible for the actual production but support the primary production units, and provide a safe and clean working environment (Stewart, 1995). They include dozers, motor graders, water trucks, fuel trucks, personnel buses and supervisors' vehicles. Dozers can be rubber-tyred or crawler-mounted. Their functions are swiping around the excavator's location to create clear haulage paths for dump trucks and levelling windrows on waste dumps. A motor grader is used to properly maintain haulage roads while water trucks are used for sprinkling water on haul roads, at dump sites and loading points for dust suppression.

3.4.2 In-Pit Crushing and Conveying System

The In-Pit Crushing and Conveying (IPCC) system is a continuous materials handling system that allows ore to be crushed in the pit and hauled out of the pit using conveyor belt systems. Figure 5 illustrates the IPCC system. It shows that IPCC is a series of connected systems mainly the discontinuous feed system (e.g. discrete excavator), crushing system, conveyor system and a discharge system.

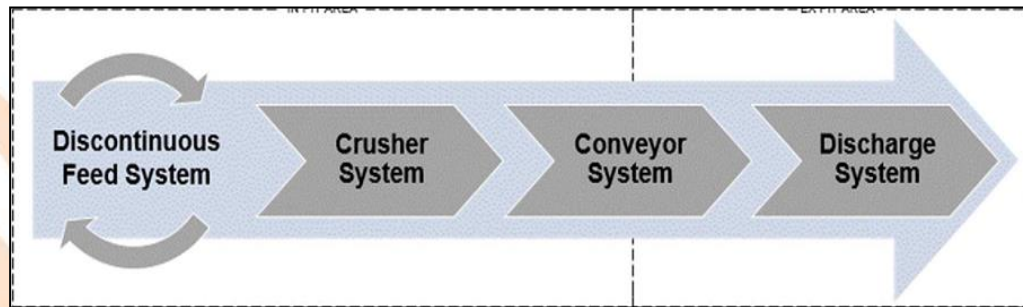


Figure 5 IPCC Haulage System Process Flow (Ritter, 2016)

The feeding system can comprise a single equipment or a number of equipment operating in combination depending on the crusher system used. Just like in the TSHS, rope shovels and hydraulic shovels, which are either diesel or electricity operated, are used to feed the crusher with materials. A rubber-tyred front end loader is usually used as an emergency loader when a regularly scheduled loading shovel is off for some time (Kennedy, 1990). According to Mohammadi *et al.* (2011), the crushing systems can either be fully mobile, semi-mobile or fixed. Fixed and semi-mobile crusher systems use shovels coupled with dump trucks as their feeding systems while fully mobile crushing systems are fed by shovels directly. The particle distribution of materials transported by a conveyor belt should be such that the largest boulder do not exceed one-third of the belt width (Dean *et al.*, 2015). The conveyor belt system transports materials off the pit and it is coupled with spreaders (for waste) or stackers (for ore) at the end as part of the discharge system.

The first IPCC system was installed in 1956 in a limestone quarry in Hover, West Germany (Mohammadi *et al.*, 2011). Since then the number of mobile IPCC operations world-wide has increased to over 1,000 including those at Twin Buttes and Sierrita Copper Mine in Arizona, Bingham Canyon Copper mine in Utah, Island Copper mines in Canada and Chuquicamata mine in Chile. A survey by Ritter (2016) notes that 447 IPCC systems have

been installed since 1956 and the installation rate has generally been increasing over the decades as shown in Figure 6.

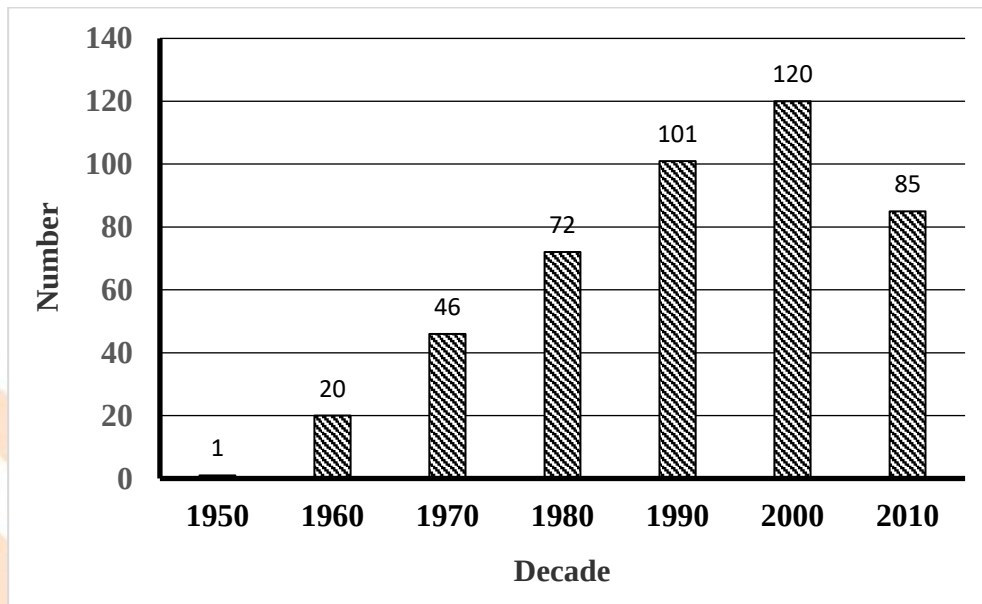


Figure 6 IPCC Installations by Decade (Ritter, 2016)

Figure 7 shows that since 1950s the number of mines using the IPCC system has increased dramatically over the years up to 120 with a slight drop in the numbers between 2000 and 2010.

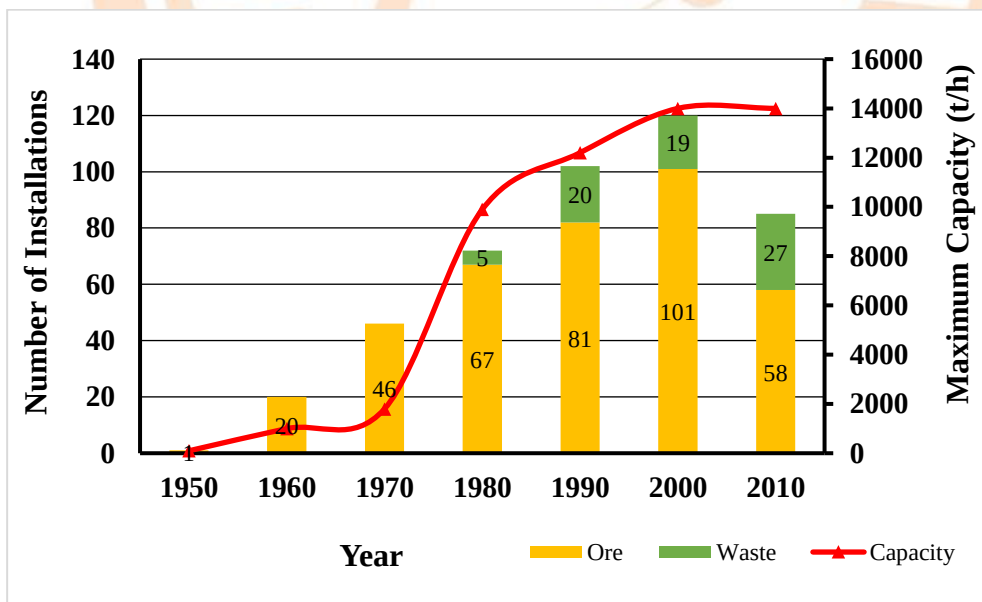


Figure 7 IPCC Installation by Decade and Type of Material Handled (Ritter, 2016)

Crusher systems: The in-pit crusher is fed with blasted materials by excavators and it reduces them to suitable sizes for transport by belt conveyors. The crusher chosen should be cost-effective yet meet the production targets. Therefore, when choosing a crushing system, the relationship between the crusher, rock strength and throughput should be considered. Three crushing systems widely used in IPCC are fixed, semi-mobile and mobile crushers. These are shown in Figure 8. This research will focused on the semi-mobile crusher system.

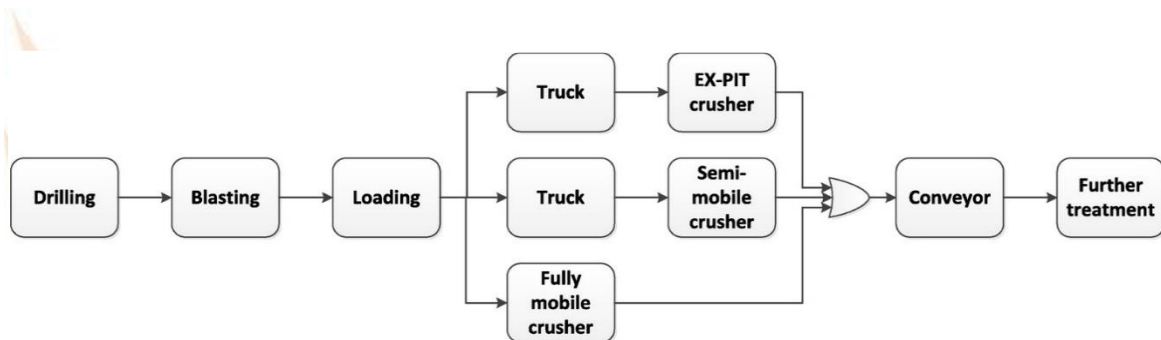


Figure 8 IPCC Conventional Transportation Systems (Paricheh *et al.*, 2017)

Fixed in-pit crushing and conveying system: This system uses both the excavator and dump trucks to feed the in-pit crusher. The dump trucks are fed by an excavator at the loading point and they haul the materials to a crusher located near the pit crest. The system is suitable for deep pits with low advancing rate, where a single crusher can meet the production rates while at the same location for not less than 10 years (Nehring *et al.*, 2018).

Semi-mobile in-pit crushing and conveying system: This system also uses both the excavator and dump trucks as its feeders. The difference is that the semi-mobile crushers are moved frequently as mining advances, usually every two to five years, depending on the rate of advance of the benches. The crushers are also located in-pit. The system is suitable for hard rocks and for crushers that are easy to install at the different locations in-pit.

Fully-mobile in-pit crushing and conveying system: This system does not rely on dump trucks but is fed directly by the excavator at the loading point. It is easily relocatable and suited for softer rocks. However, each excavator requires its own in-pit crusher.

Figure 9 shows the growth trend of IPCC in terms of the type of system. It shows that the fully mobile crushers have been used since the 1960s but the semi-mobile crushers have been mostly used since the 1980s.

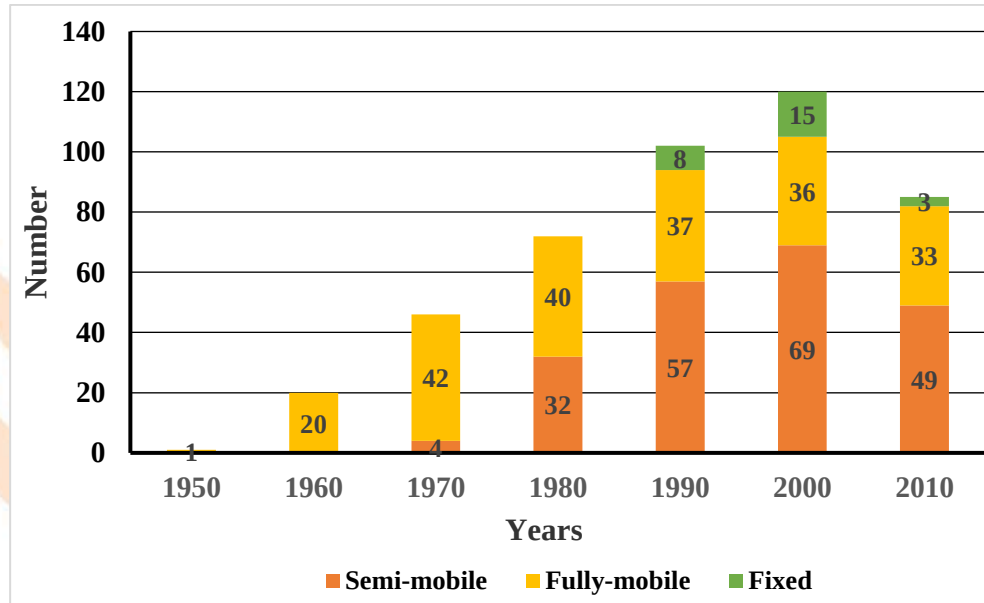


Figure 9 IPCC Installations by Decade and Type of System (Ritter, 2016)

Types of crushing systems: The crushers used in IPCC systems may be feeder breakers, gyratory crushers, hybrid crushers, impact crushers, jaw crushers, roll crushers and the sizers. To choose the right crusher for the system, Ritter (2016) recommends that the material properties (density, moisture, hardness and abrasiveness), application requirements (feed size, product size, product size distribution, content of fines, capacity) and costs must be considered. Appendix 2 summarises the parameters of primary conveyors while Tables 7 and 8 summarise the parameters for comparing the types of crushing systems including crusher types used.

Table 7 Comparison of In-Pit Crushing Systems

IPCC Crushing Options	Fully Mobile	Semi Mobile	Fixed
Throughput (t/h)	< 10,000	< 12,000	< 12,000
Truck Quality	None	Low	Intermediate
Crusher Type	Sizer, jaw/double roll crusher	Any	Any
Unit Crushing Cost	Higher	Intermediate	Lower

Source: (Mohammadi *et al.*, 2011)

Table 8 Comparison of In-Pit Crushing Systems

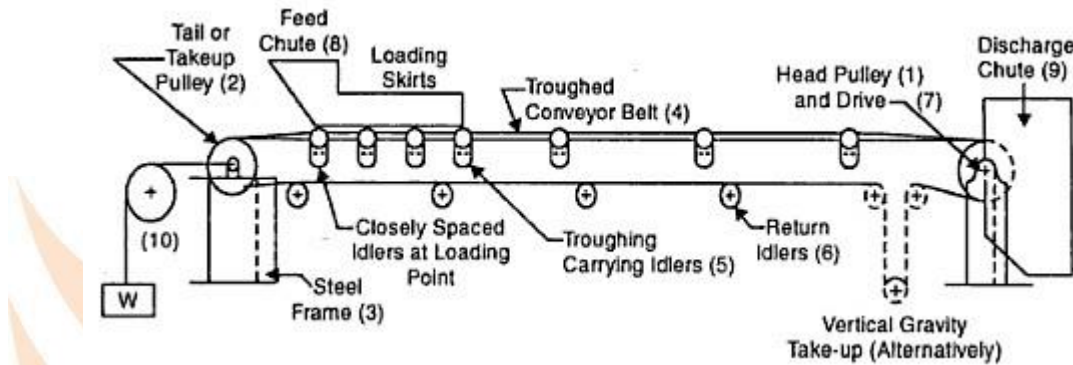
Type	Specification
Fixed	High capacity; Typical gyratory/jaw crusher; Rarely relocated; Commonly associated with transport tunnel
Semi-Fixed	High capacity; Typical gyratory/jaw crusher; Relocated every 3 - 5 years; Commonly associated with transport tunnel or wide truck ramp
Relocatable	Medium capacity; Typical twin roll crusher or sizer; Relocated every 6 - 18 months; Multiple crushing station with conveyor ramp and conveyor distribution point; Not common in deep hard rock mine
Movable	Medium - low capacity; Typical twin roll crusher or sizer; Relocated as required to follow shovel; Commonly feeds onto bench conveyor or conveyor bridge; Multiple crushing station with conveyor ramp and conveyor distribution point; To date. No application in large scale. Hard rock mine

Source: (Mohammadi *et al.*, 2011)

Conveyor belt systems: Conveyor belt systems are made up of a series of belt conveyors. A conveyor belt is basically a wide belt attached in a loop to two or more turning rotors driven by motors (Scott, 2017). The loop is actually the conveyor belt and it has two layers. The layer which gives the belt shape is called the carcass while the other layer is to help transport the belt material smoothly. Hill (2018) points out that a conveyor belt is composed of three main components namely the aluminium profile, the driving unit and the extremity unit. The aluminium profile is the conveyor belt support, the driving unit includes the motor brackets, counter-bearing and the electrical drive while the extremity unit is made up of the pulleys and clamping straps. Figure 10 is a schematic diagram of the components of a conveyor belt. It shows that a belt conveyor has a driving pulley at the head end and a take up pulley at the tail end. These pulleys are connected on both runs by an endless belt which is supported by suitable roller-type supports called idlers. The material is loaded into the belt by 1 or 2 feed hoppers at the tail end while the conveyed material is discharged over the driving pulley into the discharge spout (Anon., 2019b).

Paricheh and Osanloo (2018) state that conveyors from an in-pit crusher can exit the pit via an existing haul road, a dedicated (generally steep) conveyor ramp, a tunnel (i.e. inclined tunnel) or through a high angle conveyor. They note that the length of a conveyor, its design, and subsequently its operating and capital costs are quite different for the above alternatives.

Table 9 summarises the differences in these conveying systems. The dedicated ramp system and the conveyor tunnel system have high capital costs due to the fact that they add extra excavation and stripping costs to the project.



Legend: W = Tension counterweight

Figure 10 Different Parts of a Conveyor Belt (Anon., 2019b)

Table 9 Differences in Conveying Belt Systems

IPCC Conveying System	Dedicated Ramp	Conveyor Tunnel	Conveyor on Haul Road
Conveyor Angle	< 18°	< 10°	< 6°
Capital Cost	Intermediate	Highest	Lowest
Flexibility	Intermediate	Lowest	Highest

Source: (Paricheh and Osanloo, 2018)

Types of conveyor belt systems: Just like crusher systems, conveyor belt systems can be classified according to their mobility, design and work location. They include fully mobile, portable, shiftable, semi-fixed and fixed conveyors (Ritter, 2016).

Fully mobile belt conveyors: They are normally used with the fully mobile crushers to act as a connection between the crusher and the shiftable conveyors at the face. Fully mobile crushers which are commonly used in IPCC systems are belt wagons, conveyor bridges and horizontal conveyors.

Portable belt conveyors: These are inclined conveyors with maximum length of 42 m. They are usually used as a link between a fully mobile crusher and the further stages in the

conveyor belt system. They are also used to transport material at the downstream end of the system across active dump/heap areas where they are connected to a stacker.

Shiftable belt conveyors: They are made up of about 4 - 6 portable conveyor modules mounted on steel sleepers with steel frames to hold the carrying and return idlers. The steel rails allow the conveyor to be moved without dismantling. These conveyors can be located in-pit parallel to the operating face or at the dump face and can be moved periodically. Shiftable belt conveyors are usually associated with mobile or semi-mobile drive systems.

Semi-fixed belt conveyors: They are made up of about 4 - 6 portable conveyor belt modules mounted on concrete sleepers with steel frames to hold the carrying and return idlers. The concrete rails cause the conveyor to be dismantled during relocation. These conveyors are associated with semi-mobile or fixed drive systems.

Fixed conveyors belts: These conveyors are usually located ex-pit and used when relocation is not required throughout the mine life. They are associated with fixed drive systems.

3.4.3 High Angle Conveying

Dos Santos (2017) notes that when the overall slope of an open pit mine is less than 30°, troughed conveyor belts or smooth closed-troughed belts can be used to transport material out of the pit in conjunction with an in-pit crusher to reduce dump truck traffic in-pit. However, these conveyors require long conveying routes. He adds that in 1979 the U.S. Department of the Interior, Bureau of Mines funded a major study to develop high angle conveying systems that can haul material off pit over the shortest distance between the two end points, against the high wall. These conveying systems were to allow for material to be hauled off the pit over 30° - 90° angles against the high wall.

Sandwich belt conveyor: These conveyors are a result of the study that was funded by the U. S. Department of the Interior, Bureau of Mines in 1979. They incorporate all the advantages of conventional conveyors and have the ability to achieve high angle conveying. They achieve high angle conveying by hugging the bulk material between two belts and cause material friction which avoids gravitational slide back forces (Dos Santos, 2017). Their development was inspired by the 1970s loop belt shown in Figure 11 which was the first to introduce the concept of radial hugging pressure derived from the belt tension and

the curved profile. The development of the sandwich belt focused more on the C-profile of loop belts which did not provide a direct high angle path along a straight high incline that would be a typical path from the pit to the surface along the high wall in open pit mining (Dos Santos, 2017). Dos Santos (2012) also notes that multitude curvature reversals were introduced along the profile of a loop belt and produced the Snake Sandwich High Angle Conveyor shown in Figure 12 which allowed for vertical transport. He added that the Gently Pressed Sandwich (GPS) in Figure 13 was introduced for straight high angle portion of the profile. Its hugging pressure is applied onto the top belt by closely spaced, fully equalized pressing rolls.

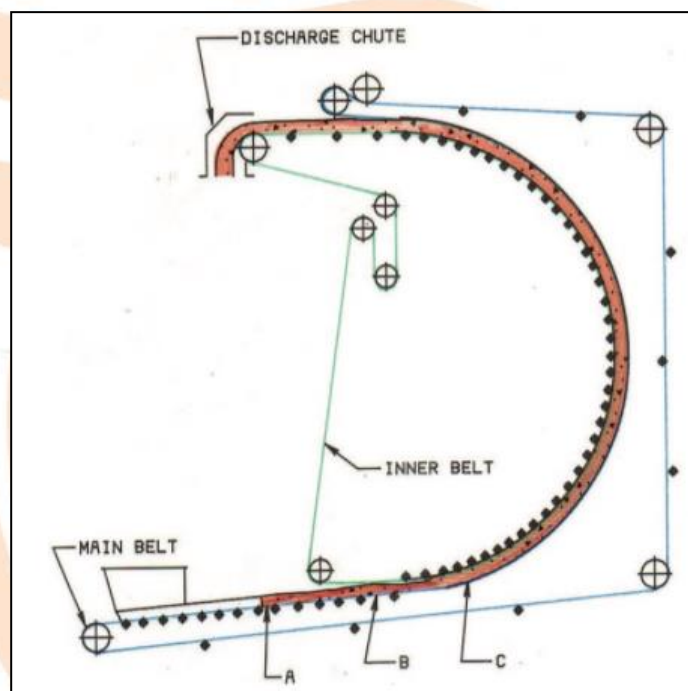


Figure 11 Schematic Diagram of a Loop Belt (Dos Santos, 2017)

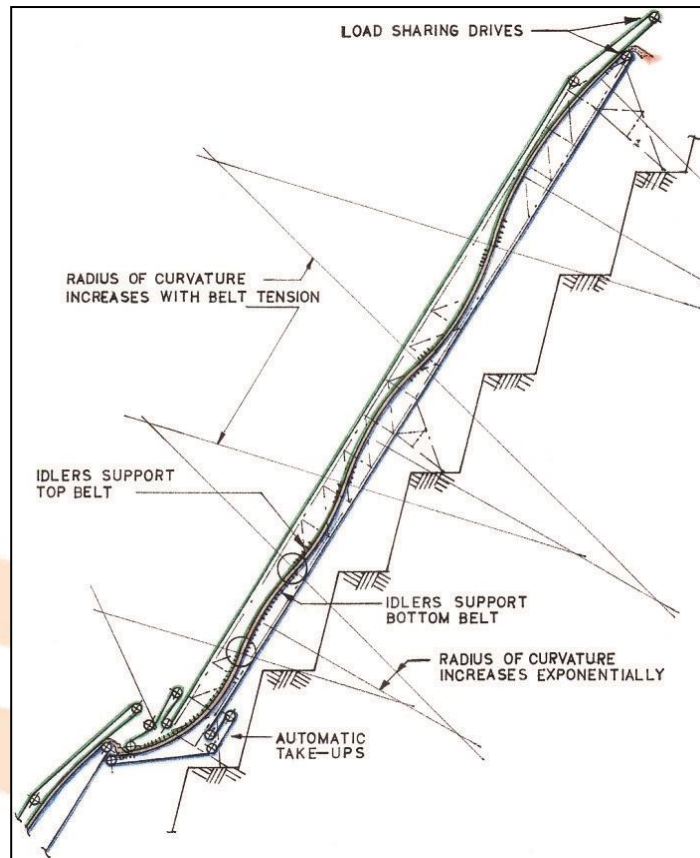


Figure 12 Schematic Diagram of a Snake Sandwich Conveyor (Dos Santos, 2017)

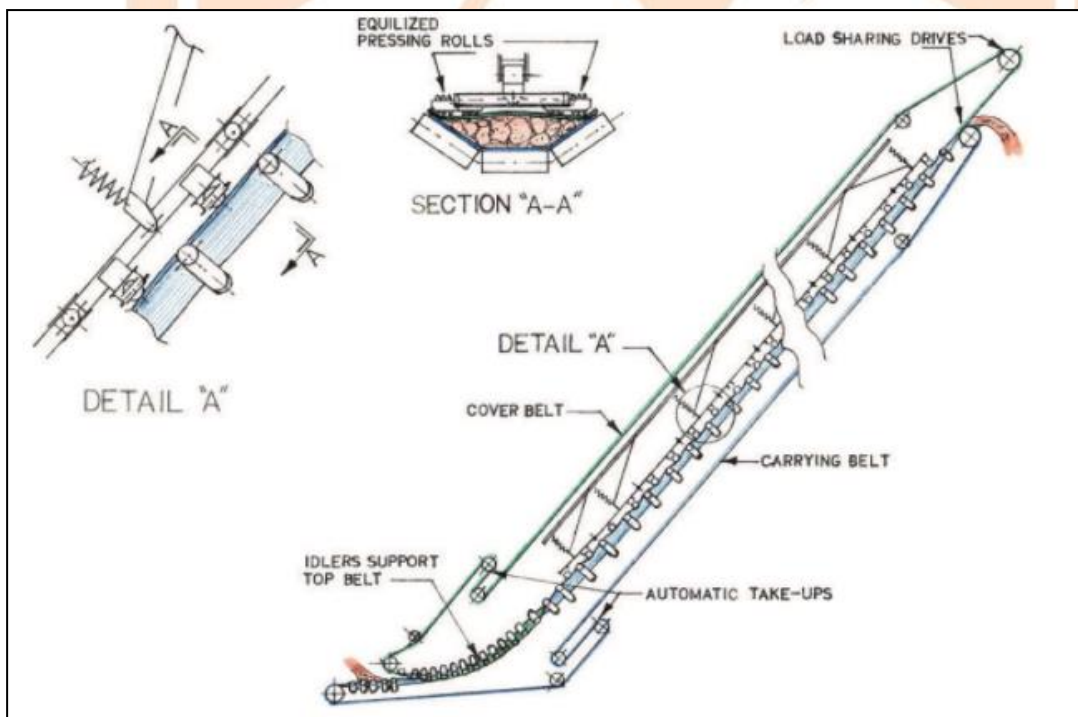


Figure 13 Schematic Diagram of a Gently Pressed Sandwich Conveyor (Dos Santos, 2017)

Chevron Mega-Pipe conveyor: This type of conveyor was founded by Contitech, Siemens and ThyssenKrupp. It is equipped with a belt conveyor with an outer diameter of up to 900 mm and nominal strengths of up to 9,500 N/mm. Figure 14 is a schematic diagram of a chevron conveyor. The belt can allow conveyance of 350 mm material size over 30° - 45° inclines. Driven by DirectDrives at the pit rim, the belt follows a vertical S-curve along the shortest route out of the open pit mine (Minkin *et al.*, 2016).

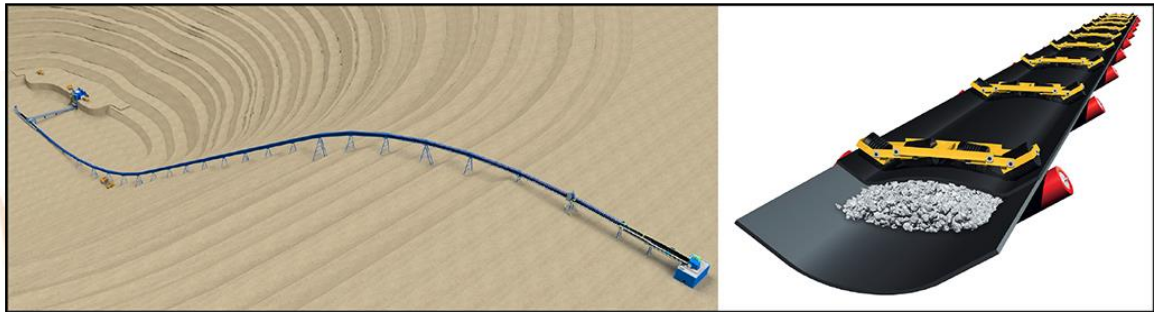


Figure 14 Schematic Diagram of Chevron Pipe Conveyor Belt (Minkin *et al.*, 2016)

Skip-way conveying system: This material haulage system was developed by ThyssenKrupp for small deep open pit mines of maximum tonnage of 5,000 mt/hr. and overall pit slopes up to 50°. It is based on cable crane technology used in large dam construction. Skips running in opposite directions are loaded by dump trucks in-pit and carry the material off pit along ropeways that are supported by a mast at the pit rim and anchored in the ground. The anchor stations off pit serve as tensioning and storage systems for excess rope that might be needed as mining progresses (Wolpers, 2016). Figure 15 shows a skip way conveying system.

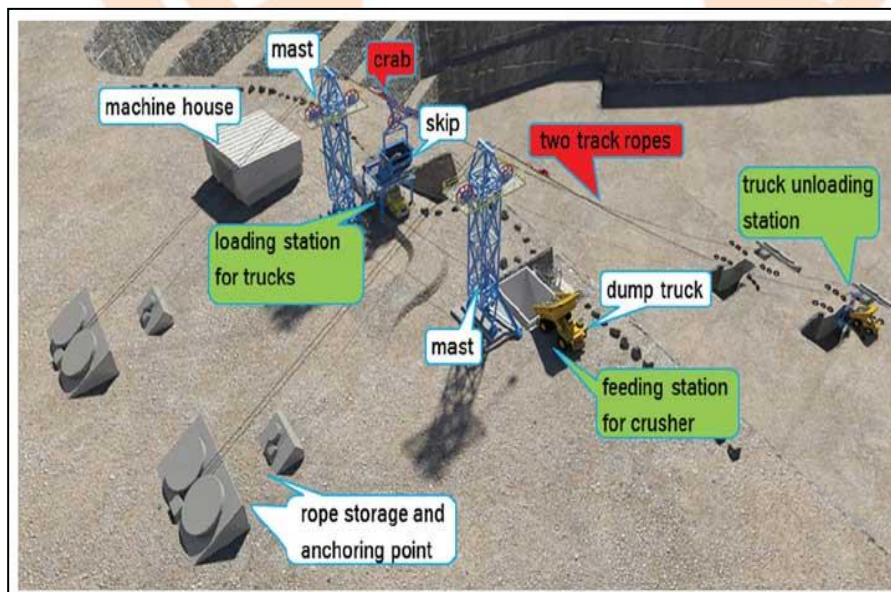


Figure 15 Schematic Diagram of Skip Way Conveying System (Wolpers, 2016)

3.5 Economic Evaluation Criteria

Mining projects involve huge capital investments and associated operating costs over the economic life of the project. Thus, they are high risk investment projects. Therefore, it is necessary to conduct economic analysis on existing and new projects to determine the acceptability or attractiveness of projects and decide on the best project to invest in (Suglo, 2010). According to Ocneanu and Cristian (2014), economic analysis is the procedure for assessing the opportunity of a project by considering the benefits compared to the costs in monetary terms. Different economic evaluation criteria such as the Net Present Value (NPV), Internal Rate of Return (IRR) and Payback Period (PP) are usually used individually or in combination to help in the analysis of projects for selection.

3.5.1 Payback Period (PP)

Payback period is defined as the time required to recover the initial investment in a project from operations (Hayes, 2020). It is established by calculating yearly returns from the start of the project until their cumulative value equals the initial project investment. The number of years at which this happens is called the payback, breakeven or recovery period. Femi and Olawale (2008) noted that this method only looks into how quick a project can return its investment and does not measure profitability. Therefore, it is not sensitive enough on its own to give the company's general investment worth. There is simple payback period which is calculated on a before tax basis ignoring the time value of money but only accounts for the absolute value of cash flows (Anon., 2021a). Discounted Payback Period, however, is an improved measure of liquidity and time risk over the simple payback period. Discounted payback period discounts future cash flows and considers the time value of money. This helps to calculate the risk involved in the project while considering the time factor and the value of money (Kagan and Drury, 2021).

3.5.2 Net Present Value (NPV)

The Net Present Value (NPV) is a discounted cash flow analysis used in capital budgeting and investment planning to analyse the profitability of a project (Kenton, 2020). It is used to test the reliability of future cash flows that an investment will yield by discounting the difference between the present value of cost inflows and the present value of cash outflows taking interest and inflation rates into account (Ong and Chun, 2013). That is, cash flows occurring at different points in time, from the time of investment to the projected future, are adjusted for the time value of money using a discount rate that is the Minimum Rate of

Return (MRR) required for the project to be acceptable. A positive NPV indicates that the projected earnings from a project are more than the anticipated costs. Therefore, a project with a positive NPV is said to be viable while the one with a negative NPV is not. If two projects have a positive NPV, the one with a higher NPV is said to be more profitable and worth investing on (Ong and Chun, 2013).

3.5.3 Internal Rate of Return (IRR)

Internal Rate of Return (IRR) is a discounted cash flow analysis that is also used in capital budgeting to determine the profitability of projects or investments (Hayes, 2020). It is the internal rate that makes all NPV of cash flows equal to zero (Christodoulou, 1996). That is, it makes the present value of estimated cash flows equal to the initial investment. IRR computes the breakeven rate of return showing discount rate. The use of "internal" refers to the omission of external factors, such as the cost of capital or inflation, from the calculation. Once the IRR is calculated, it is compared to the company's hurdle rate or cost of capital. If the IRR is greater than cost of capital, the company will accept the project as a good investment but if the IRR is less than the cost of capital then the project will be rejected. Hayes (2020) notes that the higher a project's IRR, the more attractive it is to undertake.

3.6 Sensitivity and Risk Analyses

The economic analysis of a project is based on uncertain future events. Anon. (2017) states that it uses the most likely forecast values of economic benefits and costs which are influenced by a wide range of factors and they may deviate from the forecasts. According to Suglo (2010), every mining investment venture faces many risks due to huge amounts of capital required, frequent and unpredictable changes in the market prices of commodities and equipment, type of legislations on taxes, laws on environmental protection, the prevailing political situation in the country where the project is conducted and the general consciousness of people. Anon. (2013) notes that sensitivity and risk analyses help engineers and planners to understand the critical factors that are subject to risks, the source of risks and the probable variation risk factors. This leads to improved project design and an outline of actions to mitigate the major sources of uncertainty.

Risk analysis: It incorporates the probabilities that the key variables will deviate from their forecast values and the associated risks to the project arising when these key variables vary simultaneously. It is a method of estimating the probability distribution of project outcomes.

Risk analysis helps to minimise future negative unforeseen outcomes. Anon. (2017) notes that risk analysis involves identifying key determinant factors or variables of project costs and benefits, establishing the probability distributions of these variables, randomly selecting the values of these variables from their probability distributions, combining these selected values with base case values of all other variables and parameters to estimate the NPV or IRR, repeating the last two steps numerous times to provide a large number of estimates of expected net present values (ENPVs) and expected internal rate of returns (EIRRs) and to establish their respective probability distribution; and estimating the probability of the weighted ENPV and EIRR, as well as the probability of the EIRR falling below the test rate.

Risk analysis can be either quantitative or qualitative. Quantitative risk analysis involves a risk model being built using simulation or deterministic statistics to assign numerical values to the risk. For any given input, the model generates a range of outcomes. A quantitative technique called Monte Carlo simulation can be used to generate a range of possible outcomes of a decision made or action taken. It calculates the results of the random input variables repeatedly, using a different set of input values each time. The resulting outcome from each input is recorded, and the final result of the model is a probability distribution of all possible outcomes (Hayes, 2017). According to Guthrie (2020) qualitative risk analysis is a subjective analysis which identifies project risks using a descriptive rating scale. It is done to find the probability of occurrence of project risks and their possible impact should they occur. The rating scale can describe risks as high, moderate or low depending on their severity relative to the project. Goodrich (2018) noted that qualitative risk analysis is quicker, easier and does not require any software or tools to perform it as compared to quantitative risk analysis.

Sensitivity analysis: It is a quantitative risk assessment that is undertaken to identify parameters that are uncertain and to which the project decision is sensitive, generally in terms of the NPV and IRR (Anon., 2013). It shows which task variables have the greatest impact on project parameters. Jovanovicâ (1999) describes it as a procedure that analyses how changes in certain input values influence certain output criteria values (e.g. the IRR and NPV) and the total investment project evaluation. These input values might be income, costs, mineral prices, value of investment, etc. According to Iloiu and Csiminga (2009) this analysis is aimed at helping to identify variables that influence the cost and benefit streams of an investment; investigating how much their change will be of effect; assessing whether

project decisions will be affected by such changes and identifying mitigation actions of these effects on the project. To achieve these aims, the analysis has to be conducted in a systematic manner. The steps that need to be followed are listed by Zenad (2015) as selecting key performance areas of the investment (IRR/NPV); identifying key variables to which the project decision may be sensitive, determining the possible extent of variation of these variables from the base-case and calculating the sensitivity indicator or switching value, analysing the direction and scale of the effect of different values on these variables on the project result by recalculating the NPV and IRR and interpreting the results and designing mitigating actions respectively.



CHAPTER 4

DATA COLLECTION AND ANALYSIS

4.1 Introduction

This chapter outlines the economic, risk and sensitivity analyses conducted on the materials handling systems in a deep open pit mine using data collected from Jwaneng Diamond Mine.

4.2 Data Collection

Data was gathered from previous mining reports by departments in Jwaneng Mine for TSHS. These include Load and Haul, Mine Planning, Geotechnical, Survey, Pit Services, Crushers and Conveyors. Data was also collected from the mine's service providers such as Komatsu, on the Komatsu 930E dump trucks used by the mine.

The type of data collected from load and haul, mine planning and pit services included the number of operators of shovels, dump trucks and support equipment, cycle times of dump trucks, haul road grades, purchase and operating costs of the mine's equipment. Mine Planning and Survey provided the tonnage and bank cubic meters (BCM) of materials moved and forecast over the years as well as the mine production schedules while the Geotechnical Department provided data on the rock mass properties and slope angles of the pit. Materials Crushing and Conveying Department provided data on the surface crusher and specifications of the conveyors and the number of personnel in that department. Data from Komatsu included information on the initial (purchase) prices, operating costs, maintenance costs and operating hours of the Komatsu 930E rear dump trucks.

For the IPCC system, data was obtained from the internet, journals and reports from mining companies which have either used or are currently using the system. Data collected from these sources included labour costs, operating costs and electricity costs incurred. Some online suppliers of IPCC equipment were also consulted for quotations on purchase prices, estimated maintenance and operating costs. The Materials Crushing and Conveying Department of Jwaneng Mine also provided labour costs, and the purchase, maintenance and operating costs of crushers and conveyors.

Data preparation

The data collected was divided into three categories for analysis using the TSHS and IPCC haulage systems. The data was used to calculate unit haulage costs, and to conduct economic, sensitivity and risk analyses on the two options.

4.3 Case Study

The lowest active bench at Jwaneng Mine is at a depth of 456 m below the surface. Currently, the mine has 3 active mining phases - Cuts 7, 8 and 9 (see Figure 16). Waste is mined from all the three phases while ore is only mined from Cuts 7 and 8. The annual production target of the mine is 9.3 million tonnes of ore and 105.7 million tonnes of waste (i.e. 34,000 tonnes of ore and 90,000 tonnes of waste daily). Dump trucks load ore and waste from the pit and haul it to the surface crusher or waste dumps. The surface crusher is located about 1,436 m from the pit while the waste dumps are located about 3,306 m from the pit.



Figure 16 Satellite Map of Jwaneng Open Pit Mine (Anon., 2021b)

The kimberlite pipes are about 100 m below the surface with the centre pipe and north pipe fully exposed by the Cut 7 and 8 extraction. At the end of Cuts 7 and 8 extraction, the final pit dimensions will be 2,000 m × 600 m with an overall pit slope angle of 42° and it will be 624 m below the surface. The analysis in this study started from 112 m below the surface at

Bench 8 to include benches that contain both ore and waste. Table 10 shows the production, bench design, material properties and operating parameters of the mine.

Table 10 Production Parameters at Jwaneng Mine

Material	Kimberlite	
Life of Mine (years)	16	
Pit Depth (m)	456	
Annual throughput (tonnes)	115 × 10 ⁶	
Daily Waste target (tonnes)	90,000	
Daily Ore target (tonnes)	34,000	
Bench Design Parameters		
Bench Width (m)	8	
Bench Height (m)	16	
Bench Angle	90°	
Material Properties		
Density (t/m ³)	Waste	2.8
	Ore	2.42
Uniaxial Compressive Strength (MPa)	Waste	273
	Ore	39
Haul Road		
Grade	10%	
Width	45 m	
Operating Parameters		
Annual Working Days	357	
Shifts per day	3	
Hours per shift	8	

In open pit mining, the dimensions of each bench resemble a truncated cone as shown in Figure 17. Therefore, the volume of material on each bench starting from Bench 8 was calculated using equation 1 (Bird, 2007).

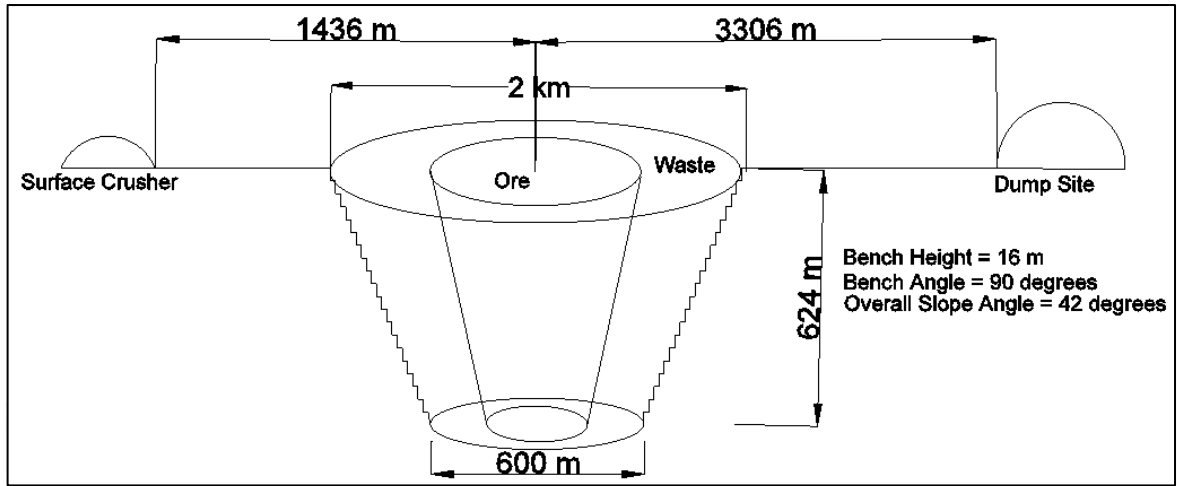


Figure 17 Final Pit Geometry of Cut 7 and 8 at Jwaneng Mine

$$\text{Volume} = \frac{1}{3} \times \pi \times [r_1^2 + (r_1 \times r_2) + r_2^2] \times h \quad (1)$$

where: r_1 = upper radius of the bench, r_2 = lower radius of the bench and h = bench height

The tonnage of materials on each bench was calculated using equation 2 (Gartneer, 2018).

$$T_o = V \times \rho \quad (2)$$

where: T_o = Tonnage of ore (tons)

V = Volume (m^3)

ρ = Density of in-situ material (tons/m^3)

The number of years required to mine each bench were then calculated using equation 3.

$$\text{Number of years per bench} = \frac{T_o}{T_d \times W_a} \quad (3)$$

where: T_o = total tonnes contained in bench

T_d = Daily production target (tons)

W_a = Total number of working days per year

Table 11 summarises the tonnage and years that will be used to mine per bench. It shows that it will take 35 years to mine Cuts 7 and 8. The number of years to mine the benches decrease with depth because the pit dimensions reduce as it deepens.

Table 11 Tonnage and Years Required to Mine per Bench

Depth (m)	Bench No.	Radius 1 (m)	Radius 2 (m)	Volume (Millions m ³)	Tonnage (Million tons)	Required Years	Cumulative No. Years
112	8	890	872	39.02	94.44	2.13	2.13
128	9	872	854	37.46	90.66	2.05	4.18
144	10	854	837	35.94	86.96	1.96	6.15
160	11	837	819	34.44	83.34	1.88	8.03
176	12	819	801	32.97	79.80	1.80	9.83
192	13	801	783	31.54	76.33	1.72	11.56
208	14	783	765	30.14	72.94	1.65	13.20
224	15	765	748	28.77	69.63	1.57	14.78
240	16	748	730	27.43	66.39	1.50	16.28
256	17	730	712	26.13	63.23	1.43	17.70
272	18	712	694	24.85	60.15	1.36	19.06
288	19	694	676	23.61	57.14	1.29	20.35
304	20	676	659	22.40	54.21	1.22	21.58
320	21	659	641	21.22	51.36	1.16	22.74
336	22	641	623	20.08	48.59	1.10	23.84
352	23	623	605	18.96	45.89	1.04	24.87
368	24	605	588	17.88	43.27	0.98	25.85
384	25	588	570	16.83	40.73	0.92	26.77
400	26	570	552	15.81	38.26	0.86	27.63
416	27	552	534	14.82	35.87	0.81	28.44
432	28	534	516	13.87	33.56	0.76	29.20
448	29	516	499	12.94	31.32	0.71	29.91
464	30	499	481	12.05	29.17	0.66	30.57
480	31	481	463	11.19	27.08	0.61	31.18
496	32	463	445	10.36	25.08	0.57	31.75
512	33	445	427	9.57	23.15	0.52	32.27
528	34	427	410	8.80	21.30	0.48	32.75
544	35	410	392	8.07	19.53	0.44	33.19
560	36	392	374	7.37	17.83	0.40	33.60
576	37	374	356	6.70	16.22	0.37	33.96
592	38	356	338	6.06	14.67	0.33	34.29
608	39	338	321	5.46	13.21	0.30	34.59
624	40	321	303	4.88	11.82	0.27	34.86

Production equipment

The details of the 930E dump trucks and other support equipment used by the mine in the TSHS are summarised in Table 12.

Table 12 Details of Mine Equipment Used at Jwaneng Mine

Dump Trucks		Support Equipment
Model	Komatsu 930E	Water bowser
Capacity (tons)	300	Graders
Operation Life (hr)	80,000	Crawler-mounted dozers (10)
Availability (%)	80	Trackless dozers
Utilisation (%)	85	
Excavators	Ore	Waste
Model	Shovel RH340	Shovel P&H 2800XPB
Bucket Capacity (m ³)	28	38
Operation Life (hrs)	60,000	150,000
Availability (%)	77	77
Utilisation (%)	80	80
Fill factor (%)	90	90

Table 13 shows the types and specifications of equipment proposed to handle the materials at Jwaneng pit using the IPCC system. Conveyors and spreaders of 1,500 tph capacity were selected to meet the mine's daily ore production target of 34,000 tonnes. In the IPCC system, the ore will be hauled by conveyor belts while waste will be hauled by the traditional TSHS.

Table 13 Proposed Equipment for IPCC at Jwaneng Mine

Conveyors		
Details	In-pit Conveyor	Overland
Capacity (tph)	1,500	1,500
Belt Width (mm)	1,200	1,000
Material Size range (mm)	< 250	< 250
Belt Availability (%)	85	85
Operation Life (years)	20	20
Discharge Systems – Spreader		
Capacity (tph)		1,500
Width (mm)		800
Availability (%)		85
Operation Life (years)		20

4.4 Unit Haulage Cost Calculations

The production parameters and capacities of the equipment in each system were used to determine the unit cost per tonne of ore of the materials handling systems considered in this study. The costs involved capital, maintenance and operating costs.

4.4.1 Unit Haulage Cost of TSHS

More dump trucks are required to meet the minimum daily production target as the pit deepens due to the increase in haul road distances. This results in an increase in unit haulage costs with the depth of pit. The cycle times of dump trucks were used to determine the number of trucks required on each bench. The cycle times were calculated by summing the loading, travelling, dumping, spotting and queuing times on each trip using equation 4.

$$\text{Cycle Time, } C_t = L_t + T_t + D_t + S_t \quad (4)$$

where: L_t = loading time;

T_t = travelling time

D_t = dumping time;

S_t = spotting and queuing times

The average spotting and queuing times for a Komatsu 930E dump truck at Jwaneng Mine was given as 4 min, average loading time is 3 min and the average dumping time is 1.03 min. Travelling times differ for waste and ore as the surface crusher is located 1,436 m from the pit while the waste dumps are located 3,306 m from the pit. Travelling times were calculated as a sum of the truck's travelling time loaded up hill, loaded on a flat ground, empty on a flat ground and empty downhill. As loading starts at the centre of a bench to the ends at Jwaneng Mine, the average flat distance to each bench was taken as the radius of that bench while the average inclined distance was calculated using equation 5. The speed limits for the dump trucks as summarised in Table 14 were used to determine travelling times using equation 6.

$$\text{Average Distance, } D_{av} = 10 \times H + D_{ds} \quad (5)$$

where: H = depth (m); D_{ds} = Distance to dumpsite (m)

$$T \text{ (min)} = \frac{D_{av} \times 60}{V} \quad (6)$$

where: D_{av} = average distance; V = Speed

Table 14 Speed Limits of Komatsu 930E Trucks at Jwaneng Mine

State of Dump Truck	Speed (km/hr)
Loaded Flat	40
Loaded Uphill	15
Empty Flat	40
Empty downhill	30

Equation 7 was used to calculate the number of trips that the dump trucks make in a day while equation 8 was used to calculate the number of trucks required per day (De Werk *et al.*, 2017). The number of trips made by a dump truck and its output per day were used to estimate the minimum number of dump trucks that will be required to meet the daily production targets of 34,000 tonnes of ore and 90,000 tonnes of waste. The results are summarised in Table 15.

$$\text{Number of trips per day, } N = \frac{SD \times U \times A}{C_t} \quad (7)$$

where: SD = Duration of shift

U = Truck utilisation

A = Truck availability

$$\text{Trucks required per day, } T_r = \frac{\text{Tons per day}}{T_c \times N} \quad (8)$$

where: T_c = truck capacity (tons)

N = No. of trips made by truck per day

Table 15 Number of Dump Trucks required per day per Bench

Bench No.	Ore Cycle Time (min)	Waste Cycle Time (min)	Cycles/day (Ore)	Cycles/day (Waste)	Ore Trucks Required/day	Waste Trucks Required/day
8	27.00	38.22	36	26	4	13
9	27.90	39.12	35	25	4	13
10	28.81	40.03	34	24	4	13
11	29.72	40.94	33	24	4	13
12	30.62	41.84	32	23	4	14
13	31.53	42.75	31	23	4	14
14	32.44	43.66	30	22	4	14
15	33.34	44.56	29	22	4	15
16	34.25	45.47	29	22	5	15
17	35.16	46.38	28	21	5	15
18	36.06	47.28	27	21	5	15
19	36.97	48.19	26	20	5	16
20	37.88	49.10	26	20	5	16
21	38.78	50.00	25	20	5	16
22	39.69	50.91	25	19	5	17
23	40.60	51.82	24	19	5	17
24	41.50	52.72	24	19	5	17
25	42.41	53.63	23	18	6	17

26	43.32	54.54	23	18	6	18
27	44.22	55.44	22	18	6	18
28	45.13	56.35	22	17	6	18
29	46.03	57.25	21	17	6	19
30	46.94	58.16	21	17	6	19
31	47.85	59.07	20	17	6	19
32	48.75	59.97	20	16	6	20
33	49.66	60.88	20	16	6	20
34	50.57	61.79	19	16	7	20
35	51.47	62.69	19	16	7	20
36	52.38	63.60	19	15	7	21
37	53.29	64.51	18	15	7	21
38	54.19	65.41	18	15	7	21
39	55.10	66.32	18	15	7	22
40	56.01	67.23	17	15	7	22

From Table 15 there were longer cycle times recorded for waste haulage than for ore haulage on the same bench due to the longer haulage distances to the waste dumps. Added to the fact that more waste has to be stripped daily than ore, more trucks are required for handling the waste than for ore haulage. Appendices 3 and 4 summarise the estimated cycle times per bench for waste and ore at Jwaneng Mine respectively. From Table 15, three additional dump trucks will be required for ore production as mining progresses from Bench #8 to Bench #40. One new truck will be added when mining reaches Bench Nos. 16, 25 and 34. Waste production will require 9 additional dump trucks to be added at Bench Nos. 12, 15, 19, 22, 26, 29, 32, 36 and 39.

The number of dump trucks required per day on each bench was used to calculate the haulage costs per hour of the TSHS. A typical equipment cost estimation form was used to calculate the unit haulage cost for TSHS (see Table 16). From Table 16, the total unit haulage cost per hour for a Komatsu 930E dump truck in Jwaneng Mine is BWP 8,056.73.

Table 16 Unit Haulage Costs of TSHS using Komatsu 930E Dump Truck

A. OWNERSHIP COSTS	
1. DEPRECIATION	
Purchase price (BWP)	61,405,008
Freight + Unloading and moving cost (BWP)	110,625
Delivered price (BWP)	61,515,633
Operating period (hr/yr)	5,040
Economic life (hr) or	80,000
Depreciation (BWP/hr)	769
2. INTEREST, TAXES, INSURANCE AND STORAGE	

Rate (%)	18
Interest (%)	14
Tax (%)	2
Others (Insurance, Storage, %)	2
Average annual investment rate (%)	53.15
Average annual investment (BWP)	32,695,559
Annual fixed charge (BWP)	5,885,201
Fixed charge (BWP/hr)	74
TOTAL OWNERSHIP COSTS	843
B. OPERATING COSTS	
1. TIRE REPLACEMENT COSTS	
Purchase price per tyre (BWP)	425,000
Purchase price for six tyres (BWP)	2,550,000
Tyre life (hr)	6,000
Tyre Cost (BWP/hr)	425
2.TIRE REPAIR COST	72.25
Tire repair factor (%)	17
3. REPAIRS AND MAINTENANCE (BWP/hr)	3,691
Repair and maintenance factor (%)	60
4. FUEL COST (BWP/hr)	1,971
Burn rate (l/hr)	260.0
Fuel cost (BWP/l)	7.58
5. LUBRICATION	985
Lubrication oil factor	0.50
6. LABOUR (BWP/hr)	69.83
Cost per month	12,000
Monthly hours (hr)	232
Benefits (%)	35
B. Total Operating Costs (BWP/hr)	7,214.22
C. Unit Haulage Costs (BWP/hr)	8,056.73

To determine the cost of moving a tonne of materials from the pit using TSHS, the unit haulage costs per hour were converted to cost/tonne using equation 9. The unit costs per tonne were then multiplied by the number of dump trucks required on each bench to determine the total cost per tonne per bench. Table 17 summarises the unit haulage costs per bench. Since it was not possible to calculate the cycle times of support equipment, their unit haulage cost in BWP/tonne was assumed to be 30% of the truck haulage costs on each bench.

$$\text{Unit Cost (BWP/t), } U_c = \frac{U_{hr}}{T_c \times \frac{60}{C_t}} \quad (9)$$

where: U_{hr} = unit cost per hour (BWP/hr); T_c = truck capacity (tonne)

C_t = cycle time of truck (min)

Table 17 Unit Cost of TSHS

Bench No.	Ore Trucks Required/day	Unit Cost Ore (BWP/tonne)	Waste Trucks Required/day	Unit Cost Waste (BWP/tonne)	Total TSHS Unit Cost (BWP/tonne)
8	4	65	13	299	364
9	4	67	13	306	373
10	4	69	13	313	382
11	4	72	13	320	392
12	4	74	14	353	427
13	4	76	14	360	436
14	4	78	14	368	446
15	4	80	15	402	482
16	5	103	15	411	514
17	5	106	15	419	525
18	5	109	15	427	536
19	5	111	16	464	575
20	5	114	16	473	587
21	5	117	16	482	599
22	5	119	17	521	640
23	5	122	17	530	652
24	5	125	17	540	665
25	6	153	17	549	702
26	6	156	18	591	747
27	6	160	18	601	761
28	6	163	18	611	774
29	6	166	19	655	821
30	6	170	19	665	835
31	6	173	19	676	849
32	6	176	20	722	898
33	6	179	20	733	912
34	7	213	20	744	957
35	7	217	20	755	972
36	7	221	21	804	1,025
37	7	225	21	815	1,040
38	7	228	21	827	1,055
39	7	232	22	878	1,110
40	7	236	22	890	1,126

From Table 17, the unit haulage cost for TSHS increases as the depth of the pit increases from BWP 364/tonne on Bench #8 to BWP 1,126/tonne at Bench #40. It can also be observed from Table 17 that while some benches (e.g. Bench Nos. 8 to 15) require the same number of dump trucks to mine ore their unit haulage costs are different due to the different haul distances. Also the unit haulage costs are noted to be significantly higher for waste than ore due to the much longer haul distances to the waste dumps.

4.4.2 Unit Haulage Cost of IPCC

The IPCC system involves mining and crushing the materials within the pit before hauling it out of the pit using conveyors. Komatsu 930E dump trucks will be used to transport materials from the excavator at the mining face to the in-pit semi-mobile crusher. The ore will be crushed and hauled by conveyors to the processing plant while the waste is hauled by dump trucks to the waste dumps. Therefore, the unit haulage cost of handling ore in this system will be the unit cost for hauling waste by dump trucks, the dump trucks that feed the in-pit semi-mobile crusher, in-pit conveyors, overland conveyors and spreaders. Assuming that the daily production of the IPCC system is the same as that of TSHS, the unit cost per tonne per bench for waste will be the same as that of waste in TSHS.

Since the semi-mobile crushers have to be moved every 3 - 5 years, the calculations were done based on moving the in-pit crusher every 5 years. Considering the number of years required to mine each bench, the crusher will be moved after every third bench at the beginning of the mining phase. The average travelling distance for IPCC dump trucks is the distance between the loading point and the in-pit crusher. The loading, dumping and spotting times of the trucks are assumed to be the same as those used for TSHS. Queuing time is assumed to be half of the TSHS queuing time due to fewer dump trucks being used. These cycle times were then used to determine the number of cycles a truck can make in a day between the loading point and the in-pit crusher, the number of trucks needed per bench and the resulting unit haulage cost per bench per tonne. The results are summarised in Table 18.

Table 18 Unit Haulage Cost of Dump Trucks using IPCC System

Bench	Cumulative Years	Average Uphill Distance (m)	Total Cycle Time (min)	Truck Cycles/day	Ore Trucks Needed/day	Ore Unit Cost (BWP/tonne)
8	2.13	160	11.66	83.98	2	14.04
9	4.18	320	12.57	77.92	2	15.13
10	6.15	480	13.47	72.68	2	16.22
11	8.03	160	11.50	85.15	2	13.84
12	9.83	320	12.41	78.93	2	14.94
13	11.56	480	13.31	73.55	2	16.03
14	13.20	160	11.34	86.35	2	13.65
15	14.78	320	12.25	79.96	2	14.74
16	16.28	480	13.15	74.45	2	15.83
17	17.70	160	11.18	87.59	2	13.46
18	19.06	320	12.09	81.02	2	14.55
19	20.35	480	12.99	75.36	2	15.64
20	21.58	160	11.02	88.86	2	13.27
21	22.74	320	11.93	82.11	2	14.36
22	23.84	480	12.83	76.31	2	15.45
23	24.87	640	13.74	71.27	2	16.54
24	25.85	800	14.65	66.86	2	17.63
25	26.77	160	10.75	91.07	2	12.94
26	27.63	320	11.66	83.99	2	14.04
27	28.44	480	12.57	77.93	2	15.13
28	29.20	640	13.47	72.68	2	16.22
29	29.91	800	14.38	68.10	2	17.31
30	30.57	960	15.29	64.06	2	18.40
31	31.18	160	10.43	93.86	2	12.56
32	31.75	320	11.34	86.36	2	13.65
33	32.27	480	12.25	79.96	2	14.74
34	32.75	640	13.15	74.45	2	15.83
35	33.19	800	14.06	69.65	2	16.92
36	33.60	960	14.97	65.43	2	18.02
37	33.96	1120	15.87	61.69	2	19.11
38	34.29	1280	16.78	58.36	3	30.30
39	34.59	1440	17.69	55.37	3	31.94
40	34.86	1600	18.59	52.67	3	33.57

From Table 18, the in-pit crusher will be moved 6 times during the life of Cuts 7 and 8. It will initially be stationed at Bench #7 and then moved to Bench Nos. 10, 13, 16, 19, 24 and 30 with time. Table 18 also shows that the number of dump trucks required per day will be the same from Bench #8 to Bench #37. At Bench #38, an additional dump truck will be required to meet daily production. The extra truck is required at this point due to longer cycle times that result from travelling over longer distances. The changes in cycle times are reflected in the unit haulage costs as well.

The relocation of the crusher will also affect the length of the in-pit conveyor and consequently affect its unit haulage cost. The length of the in-pit conveyor will be constant for the years that the crusher is stationery, and it will change only when the crusher is relocated. To determine the length of the in-pit conveyor on each bench, equation 10 was used with 1.5 as the ratio of bench height and bench face. The length of the overland conveyor will remain the same throughout at 1,500 m which is the distance between the crest of the pit and the processing plant.

$$\text{Length of in-pit conveyor, } L_c (\text{m}) = 1.5 \times C_d \quad (10)$$

where: C_d = Crusher location/depth (m)

Table 19 summarises the unit haulage costs of the IPCC system calculated based on the specifications in Table 13. Exchange rate used is USD 11.40 = BWP 1.

Table 19 Cost and Specifications of IPCC Equipment

Equipment	Width (mm)	Capacity (tph)	Purchase Cost (BWP/m)	Purchase Cost (BWP × 10 ⁶)	Unit Haulage Cost (BWP/hr)
In-pit Conveyor	1,370	1,814	59,703	35.82	4,115
Overland Conveyor	1,220	2,000	42,835	64.25	7,485
Spreader	914	1,995	-	148.13	9,417

Source: (CostMine, 2016)

The unit haulage cost of the overland conveyor and the spreader are constant as all their specifications and location are the same throughout. However, the unit haulage cost of the in-pit conveyor changes as its length changes with the crusher location. The calculated figures are summarised in Table 20.

Table 20 Unit Haulage Cost of Conveyors and Spreader per Bench using IPCC

Bench	In-pit Conveyor Length (m)	In-pit Conveyor Unit Cost (BWP/tonne)	Overland Conveyor Unit Costs (BWP/tonne)	Spreader Unit Costs (BWP/tonne)	Total Unit Cost (BWP/hr)
8	168	4.37	5.87	7.39	17.62
9	168	4.37	5.87	7.39	17.62
10	168	4.37	5.87	7.39	17.62
11	240	4.85	5.87	7.39	18.11
12	240	4.85	5.87	7.39	18.11
13	240	4.85	5.87	7.39	18.11
14	312	5.34	5.87	7.39	18.60
15	312	5.34	5.87	7.39	18.60
16	312	5.34	5.87	7.39	18.60
17	384	5.83	5.87	7.39	19.09
18	384	5.83	5.87	7.39	19.09
19	384	5.83	5.87	7.39	19.09
20	456	6.32	5.87	7.39	19.57
21	456	6.32	5.87	7.39	19.57
22	456	6.32	5.87	7.39	19.57
23	456	6.32	5.87	7.39	19.57
24	456	6.32	5.87	7.39	19.57
25	528	6.81	5.87	7.39	20.06
26	528	6.81	5.87	7.39	20.06
27	528	6.81	5.87	7.39	20.06
28	528	6.81	5.87	7.39	20.06
29	528	6.81	5.87	7.39	20.06
30	528	6.81	5.87	7.39	20.06
31	600	7.29	5.87	7.39	20.55
32	600	7.29	5.87	7.39	20.55

33	600	7.29	5.87	7.39	20.55
34	600	7.29	5.87	7.39	20.55
35	600	7.29	5.87	7.39	20.55
36	600	7.29	5.87	7.39	20.55
37	600	7.29	5.87	7.39	20.55
38	600	7.29	5.87	7.39	20.55
39	600	7.29	5.87	7.39	20.55
40	600	7.29	5.87	7.39	20.55

From Table 20, the maximum length of an in-pit conveyor at the end of the mining phase will be 600 m. Therefore, a total of 600 m will be purchased for in-pit conveyor and 1,500 m for the overland conveyor. The increase in unit haulage costs for the in-pit conveyor are because there will be increases in the length of the in-pit belt conveyor, maintenance costs increase and power consumption. Also more operators will be required for the conveyor belts. According to De Werk *et al.* (2017), the unit haulage cost of an in-pit conveyor increases by 0.21% of the total haulage cost per meter. Therefore, equation 11 was used to calculate the unit haulage cost of the in-pit conveyors using the capacities and availabilities of the belt conveyors given in Table 13.

$$U_{\text{in-pit}} (\text{BWP/tonne}) = \frac{U_{\text{hr}} (1 + 0.0021 \times L_c)}{C_{\text{cp}} \times B_{\text{av}}} \quad (11)$$

where: C_{cp} = Capacity of belt conveyor (tph)

U_{hr} = Unit cost/hr

B_{av} = availability of belt (%)

L_c = Length of in-pit conveyor (m)

The unit haulage cost of the overland conveyor and spreader were converted from cost/hr to cost/tonne using equation 12.

$$\text{Unit Haulage Cost} = \frac{\text{Unit Cost}}{\text{Belt Capacity} \times \text{Belt availability}} \quad (12)$$

Table 21 summarises the calculated unit haulage costs of the IPCC system. It is the sum of the cost to move a tonne of ore from the loading point to the in-pit crusher by dump trucks

and then by in-pit conveyors, overland conveyors and the spreader to the processing plant together with the unit cost of waste handling by dump trucks.

Table 21 Unit Haulage Cost for IPCC

Bench No.	Ore Unit Cost (BWP/tonne)	Waste Unit Cost (BWP/tonne)	Total Unit Cost (BWP/tonne)
8	31.66	299.05	330.71
9	32.75	306.14	338.89
10	33.84	313.24	347.08
11	31.95	320.33	352.29
12	33.05	352.61	385.66
13	34.14	360.25	394.39
14	32.25	367.89	400.14
15	33.34	402.36	435.70
16	34.43	410.54	444.98
17	32.54	418.73	451.27
18	33.64	426.92	460.55
19	34.73	464.11	498.84
20	32.84	472.84	505.68
21	33.93	481.57	515.50
22	35.02	520.95	555.97
23	36.11	530.22	566.34
24	37.21	539.50	576.71
25	33.01	548.78	581.79
26	34.10	590.88	624.98
27	35.19	600.71	635.90
28	36.28	610.53	646.81
29	37.37	654.82	692.19
30	38.46	665.19	703.65
31	33.11	675.55	708.66
32	34.20	722.02	756.22
33	35.29	732.94	768.23
34	36.38	743.85	780.24
35	37.47	754.77	792.24
36	38.57	803.97	842.53
37	39.66	815.43	855.08
38	50.85	826.89	877.74

39	52.49	878.27	930.76
40	54.12	890.28	944.40

In the TSHS, all materials are hauled from the pit using dump trucks while with the IPCC system, ore haulage is done by belt conveyors and waste hauling is done using dump trucks. Figure 18 shows the calculated unit haulage costs of TSHS and IPCC in Cuts 7 and 8.

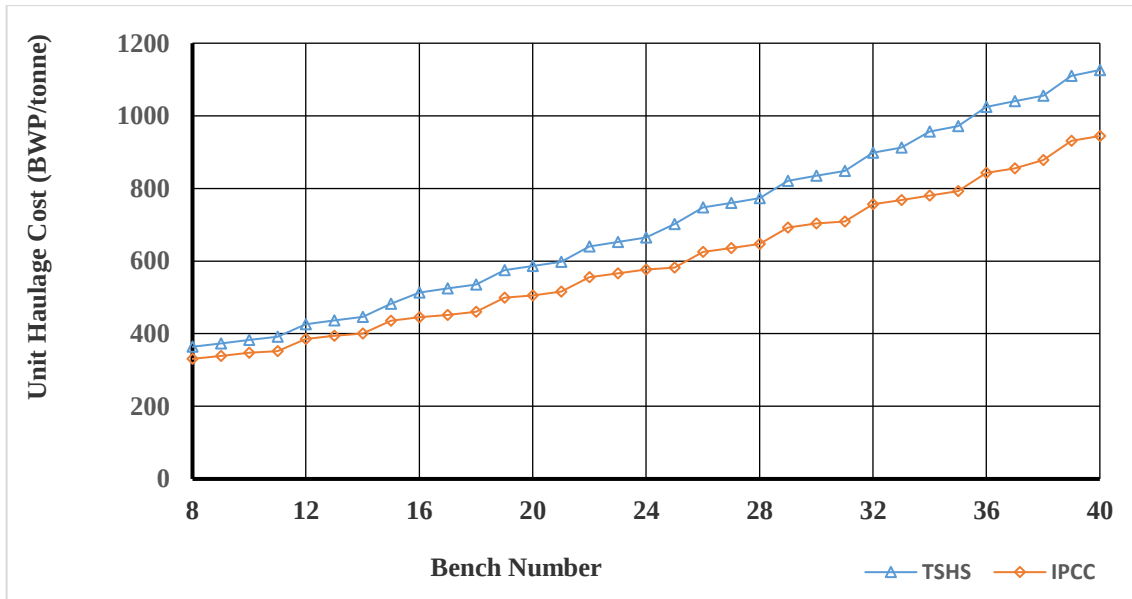


Figure 18 Estimated Unit Haulage Costs of TSHS and IPCC at Cuts 7 and 8

From Figure 18, the IPCC has lower unit haulage costs than the TSHS especially as the pit deepens from Bench #8 to Bench #40. At the beginning of the operations at Bench #8, the unit haulage costs are almost the same with a difference of BWP 33/tonne but as mining progresses, the difference in unit haulage costs for TSHS and IPCC increase to BWP 182/tonne at Bench #40.

Figure 19 also shows that the total unit haulage cost for TSHS increased by 68% from the beginning of operation at Bench #8 to the end of operation at Bench #40 while that of the IPCC system increased by 65%. This increase is due to the long haulage distances of the TSHS that require more trucks to meet the minimum daily production targets. Also from Figure 19 the costs of the TSHS option for ore alone increased by 72% from the beginning of extraction on Bench #8 to the end of extraction on Bench #40 while that of IPCC increased by 42%. These increases are shown in Figure 20 where the plots of the two

systems show an initial difference of BWP 33/tonne but a huge difference of BWP182/tonne at the end of by Bench #40.

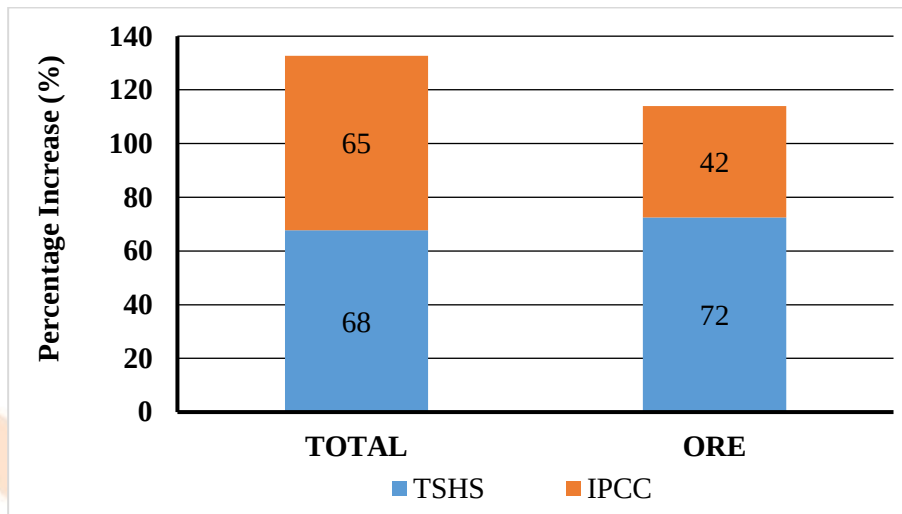


Figure 19 Percentage Increase of Unit Haulage Costs for TSHS and IPCC from Bench #8 to Bench #40

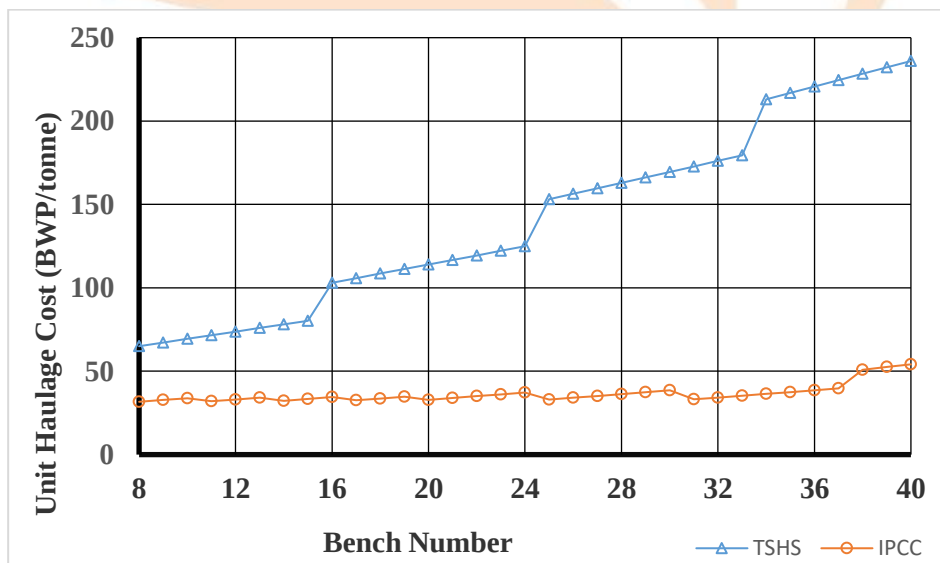


Figure 20 Unit Haulage Costs of Extracting Ore Using TSHS and IPCC at Cuts 7 and 8

The unit haulage costs of the IPCC system are lower due to the shorter haul distances of the dump trucks within the pit between the loading point (excavator) and the in-pit crusher. This resulted in shorter cycle times of the trucks, fewer truck requirements and consequently lower initial and maintenance costs, and less fuel consumed by the trucks. In addition, only one truck had to be added to the fleet at Bench #38.

Also from Figure 20, the unit haulage cost for ore by TSHS increases steadily from Benches #8 to #15 but there is a sudden jump at Bench #16, The next jumps in unit costs occur at Bench Nos. 25 and 34 otherwise the increases are steady over the years. The sudden increases in unit costs at Bench Nos. 16, 25 and 34 are due to the additional trucks that are added to fleet to meet production targets. The heights of the spikes in unit costs increase with depth (i.e. the spike in unit costs at Bench #16 is smaller than that at Bench #25 and which in turn is also smaller than the spike at Bench #34). This shows the impact of the increase in haul road distance on cycle times and hence on unit haulage costs.

4.5 Economic Analysis

The Net Present Value (NPV), Internal Rate of Return (IRR) and Discounted Payback Period (DPP) were used to assess the economic viability of TSHS and IPCC haulage system over the life of mine.

4.5.1 Net Present Value

The NPV for each system was calculated using the main production equipment of the systems. The NPV was calculated as the sum of the present values of cash flows over 35 years in Cuts 7 and 8 production. Cash outflows were calculated as a sum of operating and capital costs using equation 13 while cash inflows were calculated using equation 14.

$$\text{Cash Outflows, } C_o = (U_c \times D_T \times N) + (E_N \times P_p) \quad (13)$$

where: U_c = Unit cost (BWP/tonne)

D_T = Daily ore target (tonnes)

N = Annual production days

E_N = Number of equipment purchased

P_p = Purchase price (BWP)

$$\text{Cash Inflows, } C_i = (D_T \times N) \times G_R \times S_p \quad (14)$$

where: G_R = Average ore grade (cpht)

D_T = Daily ore target (tonnes)

S_p = Average carats sales price

N = Annual production days

In calculating the operating costs, consideration was given to the bench that would be mined each year by using the unit haulage costs of that bench while capital costs considered the number of equipment required in each year and their operating lives. An average grade of 160 carats per hundred tonnes (cpht) and an average sales price of BWP 1,707.38/carats were used to calculate the annual cash inflows assuming a mining recovery of 100%. Due to the same daily ore target production assumed for both systems, the annual cash inflows were the same for both systems as BWP 43.10 billion. The present value of these future cash flows were then determined by discounting the cash flows in each year using equation 15 at a discount rate of 15% used by Jwaneng Mine.

$$\text{Discounted Cash Flow (DCF)} = \frac{\text{Capital Cost}}{(1+n)^i} \quad (15)$$

where: n = discount rate and i = the year of investment

4.5.2 NPV of TSHS

From the calculations, the mine will spend BWP 246 million as the initial capital on dump trucks for ore haulage. As the pit deepens and cycle times increase, one new truck will be needed in years 16, 26 and 32 to meet daily production demands. The results are summarised in Table 22.

Table 22 NPV of TSHS for Ore Haulage

Year	Bench	Unit Cost (BWP)	Operating Costs (Billion BWP)	No. of Dump Trucks Purchased	Capital Costs (Billion BWP)	Cash Inflows (Billion BWP)	Cash Outflows (Billion BWP)	Cashflow (Billion BWP)	Discounted Cashflows (Billion BWP)
0	-	-	-	4	0.25	-	0.25	-2.46	-
1	8	65	0.79	0	0	43.10	0.79	42.31	36.79
2	8	65	0.79	0	0	43.10	0.79	42.31	31.99
3	9	67	0.81	0	0	43.10	0.81	42.29	27.80
4	9	67	0.81	0	0	43.10	0.81	42.29	24.18
5	10	69	0.84	0	0	43.10	0.84	42.26	21.01
6	10	69	0.84	0	0	43.10	0.84	42.26	18.27
7	11	72	0.87	0	0	43.10	0.87	42.22	15.87
8	11	72	0.87	0	0	43.10	0.87	42.22	13.80
9	12	74	0.90	0	0	43.10	0.90	42.20	11.99
10	12	74	0.90	0	0	43.10	0.90	42.20	10.43
11	13	76	0.92	0	0	43.10	0.92	42.18	9.07
12	13	76	0.92	0	0	43.10	0.92	42.18	7.88
13	14	78	0.95	0	0	43.10	0.95	42.15	6.85

14	15	80	0.97	0	0	43.10	0.97	42.13	5.95
15	15	80	0.97	0	0	43.10	0.97	42.13	5.18
16	16	103	1.25	1	0.06	43.10	1.31	41.79	4.47
17	17	106	1.29	4	0.25	43.10	1.53	41.57	3.86
18	17	106	1.29	0	0	43.10	1.29	41.81	3.38
19	18	109	1.32	0	0	43.10	1.32	41.78	2.94
20	19	111	1.35	0	0	43.10	1.35	41.75	2.55
21	20	114	1.38	0	0	43.10	1.38	41.71	2.22
22	20	114	1.38	0	0	43.10	1.38	41.71	1.93
23	21	117	1.42	0	0	43.10	1.42	41.68	1.68
24	22	119	1.45	0	0	43.10	1.45	41.65	1.46
25	23	122	1.48	0	0	43.10	1.48	41.62	1.26
26	24	125	1.52	1	61	43.10	1.58	41.52	1.10
27	25	153	1.86	0	0	43.10	1.86	41.24	0.95
28	26	156	1.90	0	0	43.10	1.90	41.20	0.82
29	28	163	1.98	0	0	43.10	1.98	41.12	0.71
30	29	116	1.99	0	0	43.10	1.99	41.69	0.63
31	30	170	2.06	0	0	43.10	2.06	41.03	0.54
32	32	176	2.14	2	0.12	43.10	2.26	40.84	0.47
33	34	213	2.59	4	0.25	43.10	2.83	40.28	0.40
34	36	221	2.68	0	0	43.10	2.68	40.42	0.35
35	40	236	2.87	0	0	43.10	2.87	40.23	0.30

From Table 22, four more trucks will have to be purchased in years 17 and 33. These trucks are to replace the initial trucks which would have reached their operating lives of 15.9 years. Table 22 also shows that the operating cost in the first year of production was BWP 0.79 billion but it increased over the years to BWP 2.87 billion by Year 35. These operating costs together with capital costs resulted in cash outflows being directly proportional with the years of production. However, huge increases in cash flows are noted in years 17 and 33 due to the new dump trucks purchased to replace the old ones. The NPV for ore haulage using the TSHS was calculated to be BWP 27.9 billion.

The mine will spend about BWP 1.04 billion as the initial capital on dump trucks for both ore and waste haulage using TSHS. Table 23 shows that one more additional truck will be required in years 9, 14, 16, 20, 23, 26, 27, 29, 32, 33 and 34 as the pit deepens. The mine will have to replace the initial dump trucks in years 17 and 33 resulting in high capital costs and cash outflows in those years. Trucks bought in years 9, 14 and 16 will also reach the end of their operation lives before year 35. Therefore, they will have to be replaced in years 25, 30 and 32. Table 23 shows that the operating costs increase with pit depth from BWP

16.11 billion to BWP 49.85 billion. The net present value of cash flows for ore and waste haulage using TSHS was BWP 169.48 billion.

Table 23 NPV of Ore and Waste Haulage using TSHS

Year	Bench	Unit Cost (BWP)	Operating Costs (Billion BWP)	Dump Trucks Purchased	Capital Costs (Billion BWP)	Cash Inflows	Cash Outflow (Billion BWP)	Cashflow (Billion BWP)	Discounted Cashflows (Billion BWP)
0	-	-	-	17	1.04	-	1.04	-1.04	-
1	8	364	16.11	0	0	43.10	16.11	26.98	23.47
2	8	364	16.11	0	0	43.10	16.11	26.98	20.40
3	9	373	16.51	0	0	43.10	16.51	26.59	17.48
4	9	373	16.51	0	0	43.10	16.51	26.59	15.20
5	10	283	12.53	0	0	43.10	12.53	30.57	15.20
6	10	283	12.53	0	0	43.10	12.53	30.57	13.22
7	11	292	12.93	0	0	43.10	12.93	30.17	11.34
8	11	292	12.93	0	0	43.10	12.93	30.17	9.86
9	12	426	18.86	1	0.06	43.10	18.92	24.18	6.87
10	12	426	18.86	0	0	43.10	18.86	24.24	5.99
11	13	436	19.30	0	0	43.10	19.30	23.80	5.12
12	13	436	19.30	0	0	43.10	19.30	23.80	4.45
13	14	446	19.74	0	0	43.10	19.74	23.35	3.80
14	15	483	21.38	1	0.06	43.10	21.44	21.66	3.06
15	15	483	21.38	0	0	43.10	21.38	21.72	2.67
16	16	514	22.75	1	0.06	43.10	22.82	20.28	2.17
17	17	525	23.24	17	1.04	43.10	24.29	18.81	1.75
18	17	525	23.24	0	0	43.10	23.24	19.86	1.61
19	18	535	23.68	0	0	43.10	23.68	19.41	1.36
20	19	575	25.45	1	0.06	43.10	25.52	17.58	1.07
21	20	587	25.99	0	0	43.10	25.99	17.11	0.91
22	20	587	25.99	0	0	43.10	25.99	17.11	0.79
23	21	598	26.47	1	0.06	43.10	26.53	16.56	0.67
24	22	640	28.35	0	0	43.10	28.35	14.75	0.52
25	23	652	28.88	1	0.06	43.10	28.94	14.16	0.43
26	24	664	29.41	1	0.06	43.10	29.47	13.62	0.36
27	25	702	31.07	1	0.06	43.10	31.14	11.96	0.28
28	26	747	33.08	0	0	43.10	33.08	10.02	0.20
29	28	774	34.26	1	0.06	43.10	34.33	8.77	0.15
30	29	821	36.34	1	0.06	43.10	36.41	6.69	0.01
31	30	835	36.96	1	0.06	43.10	37.03	6.07	0.08

32	32	898	39.75	2	0.12	43.10	39.88	3.22	0.04
33	34	957	42.36	18	1.11	43.10	43.48	-0.37	0.00
34	36	1,025	45.38	1	0.06	43.10	45.44	-2.34	-0.02
35	40	1,126	49.85	0	0	43.10	49.85	-6.75	-0.05

4.5.3 NPV of IPCC

The mine will spend BWP 0.35 billion as initial capital for ore haulage only and BWP 1.14 billion as the initial capital for both ore and waste haulage. The capital cost for ore is the sum of the cost for dump trucks and conveyors while that of waste is entirely for dump trucks. Tables 24 and 25 show the cash flows for ore haulage by IPCC only and for both ore and waste respectively.

When conveying ore using IPCC, the mine will have to purchase only 2 trucks at the beginning of operation to transport ore from the mining face to the in-pit crusher. One more truck will be required in year 34 to meet production. The two original trucks will require replacement in years 17 and 33 while the overland conveyor and spreader will require replacement in year 20 after reaching their operating life (see Table 24). Operating costs vary between BWP 0.29 billion and BWP 0.66 billion due to the distance of the haul road between the in-pit crusher and the loading point for dump trucks which keeps on changing as the crusher is relocated. The discounted cashflows resulted in an NPV of BWP 282.25 billion for the IPCC.

Due to the large number of trucks used for waste haulage, many dump trucks have to be purchased and replaced as shown in Table 25. A total of 15 dump trucks will be needed at the beginning of operation for ore and waste haulage. These trucks will have to be replaced in years 17 and 33 when they reach the end of their operating lives. Another replacement will occur in years 25 and 30 for dump trucks bought in years 9 and 14 respectively. The net present value of the system was calculated to be BWP 174.64 billion.

A summary of cash flows in Table 26 shows that both systems have positive net present values which indicates that both projects are viable but IPCC has higher NPV than TSHS. Thus the IPCC system is considered more favourable than the TSHS. Table 26 also shows that even though IPCC has lower operating costs than TSHS, its initial capital costs are much higher than TSHS. These high capital costs for IPCC is due to the fact that almost all the equipment used in IPCC system have to be bought at once at the beginning of operation

while for the TSHS, the dump trucks are bought in batches to meet production targets at the time.



Table 24 NPV of Ore Haulage using IPCC

Year	Bench No.	Unit Cost (BWP)	Operating Costs (Billion BWP)	Dump Truck & Conveyor Belt	Capital Costs (Million BWP)	Cash Inflows (Billion BWP)	Cash Outflow (Billion BWP)	Cashflow (Billion BWP)	Discounted Cashflows (Billion BWP)
0	-	-	-	2D & 168 m	345	-	0.35	-0.35	-
1	8	32	0.39	0	0	43.10	0.39	42.71	37.14
2	8	32	0.39	0	0	43.10	0.39	42.71	32.29
3	9	33	0.40	0	0	43.10	0.40	42.70	28.07
4	9	33	0.40	0	0	43.10	0.40	42.70	24.41
5	10	24	0.29	0	0	43.10	0.29	42.81	21.28
6	10	24	0.29	0	0	43.10	0.29	42.81	18.51
7	11	32	0.39	0	0	43.10	0.39	42.71	16.06
8	11	32	0.39	72 m	4	43.10	0.39	42.71	13.96
9	12	33	0.40	0	0	43.10	0.40	42.70	12.14
10	12	33	0.40	0	0	43.10	0.40	42.70	10.55
11	13	34	0.41	0	0	43.10	0.41	42.69	9.18
12	13	34	0.413	0	0	43.10	0.41	42.69	7.98
13	14	32	0.39	72 m	4	43.10	0.39	42.71	6.94
14	15	33	0.40	0	0	43.10	0.40	42.70	6.03
15	15	33	0.40	0	0	43.10	0.40	42.70	5.25
16	16	34	0.41	0	0	43.10	0.41	42.69	4.56
17	17	33	0.40	2D & 72 m	127	43.10	0.53	42.57	3.96
18	17	33	0.40	0	0	43.10	0.40	42.70	3.45
19	18	34	0.41	0	0	43.10	0.41	42.69	3.00
20	19	35	0.43	168 m	222	43.10	0.65	42.45	2.59
21	20	33	0.40	72 m	4	43.10	0.41	42.69	2.27
22	20	33	0.40	0	0	43.10	0.40	42.70	1.97

23	21	34	0.41	0	0	43.10	0.41	42.70	1.72
24	22	35	0.43	0	0	43.10	0.43	42.67	1.49
25	23	36	0.44	0	0	43.10	0.44	42.66	1.30
26	24	37	0.45	72 m	4	43.10	0.46	42.64	1.13
27	25	33	0.40	0	0	43.10	0.40	42.70	0.98
28	26	34	0.41	72 m	4	43.10	0.42	42.68	0.85
29	28	36	0.44	0	0	43.10	0.44	42.66	0.74
30	29	37	0.45	0	0	43.10	0.45	42.65	0.64
31	30	38	0.46	72 m	4	43.10	0.47	42.63	0.56
32	32	34	0.41	0	0	43.10	0.41	42.69	0.49
33	34	36	0.44	2D & 72 m	127	43.10	0.56	42.53	0.42
34	36	39	0.47	1 D	61	43.10	0.54	42.56	0.37
35	40	54	0.66	0	0	43.10	0.66	42.44	0.32

where: D = Dump trucks

Table 25 NPV of Ore and Waste Haulage using IPCC

Year	Bench	Unit Cost (BWP)	Operating Costs (Billion BWP)	Dump Truck & Conveyor Belt Purchased	Capital Costs (Million BWP)	Cash Inflows (Billion BWP)	Cash Outflow (Billion BWP)	Cashflow (Billion BWP)	Discounted Cashflows (Billion BWP)
0	-	-	-	15D & 168 m	1,143	-	1.14	-1.14	-
1	8	331	4.01	0	0	43.10	4.01	28.46	24.75
2	8	331	4.01	0	0	43.10	4.01	28.46	21.52
3	9	339	4.12	0	0	43.10	4.12	28.09	18.47
4	9	339	4.12	0	0	43.10	4.12	28.09	16.06
5	10	347	4.21	0	0	43.10	4.21	27.74	13.79
6	10	347	4.21	0	0	43.10	4.21	27.74	11.99
7	11	352	4.27	0	0	43.10	4.27	27.52	10.34

8	11	352	4.27	72 m	4	43.10	4.23	27.51	8.99
9	12	386	4.69	1D	61	43.10	4.74	25.95	7.38
10	12	386	4.69	0	0	43.10	4.69	26.01	6.43
11	13	394	4.78	0	0	43.10	4.78	25.66	5.52
12	13	394	4.78	0	0	43.10	4.78	25.66	4.80
13	14	400	4.86	72 m	4	43.10	4.86	25.39	4.13
14	15	436	5.29	1D	61	43.10	5.35	23.74	3.36
15	15	436	5.29	0	0	43.10	5.29	23.80	2.93
16	16	445	5.40	0	0	43.10	5.40	23.40	2.50
17	17	451	5.47	15D & 72 m	925	43.10	6.40	22.21	2.06
18	17	451	5.47	0	0	43.10	5.47	23.13	1.87
19	18	461	5.60	0	0	43.10	5.60	22.69	1.60
20	19	499	6.06	1D & 168 m	284	43.10	6.34	20.72	1.27
21	20	506	6.14	72 m	4	43.10	6.15	20.69	1.10
22	20	506	6.14	0	0	43.10	6.14	20.70	0.96
23	21	516	6.26	1D	61	43.10	6.32	20.22	0.81
24	22	556	6.75	0	0	43.10	6.75	18.49	0.65
25	23	566	6.87	1D	61	43.10	6.94	17.97	0.55
26	24	577	7.00	72 m	4	43.10	7.00	17.56	0.46
27	25	582	7.06	1D	61	43.10	7.12	17.28	0.40
28	26	625	7.59	72 m	4	43.10	7.59	15.43	0.31
29	28	647	7.85	1D	61	43.10	7.92	14.40	0.25
30	29	692	8.40	1D	61	43.10	8.46	12.40	0.19
31	30	704	8.55	1D & 72 m	66	43.10	8.61	11.87	0.16
32	32	756	9.18	0	0	43.10	9.18	9.63	0.11
33	34	780	9.47	16D & 72 m	987	43.10	10.44	7.58	0.80
34	36	843	10.23	2D	123	43.10	10.36	5.66	0.50
35	40	994	12.07	0	0	43.10	12.07	-0.91	0.00

Table 26 Summary of Cash Flows of TSHS and IPCC

Type of Cost	TSHS		IPCC	
	Ore	Ore & Waste	Ore	Ore & Waste
Capital Costs (BWP billion)	0.98	4.05	1.12	4.19
Operating Costs (BWP billion)	47.76	887.51	14.47	215.16
Total Cash Outflow (BWP billion)	48.74	891.56	15.38	219.13
NPV (BWP billion)	278.83	169.48	282.25	174.64

4.5.4 Discounted Payback Period (DPP)

DPP was used to assess the economic viability of TSHS and IPCC haulage system using the time that each system will be able to recover its investment costs taking into account the time value of money. Assuming that all the equipment that will be required for the production life in the two haulage systems are purchased in the same year (Year 0), equation 16 was used to determine the DPP of each system.

$$DPP = Y_n - \frac{L_{nc}}{DFC_n} \quad (16)$$

where: Y_n = The year cumulative of discounted cash flows was last negative (years)

L_{nc} = Last negative cumulative (BWP)

DFC_n = Discounted cashflow of the years after Y_n (BWP)

The results of DPP calculations are summarised in Table 27. The results show that the DPP of the TSHS and IPCC in ore and waste are 2.94 months and 2.13 months respectively. This indicates that both systems are acceptable as they have very short DPPs. However, IPCC is considered more economic than TSHS due to its shorter DPP.

Table 27 Discounted Payback Period of TSHS and IPCC

Factor	TSHS		IPCC	
	Ore	Ore & Waste	Ore	Ore & Waste
Investment Costs (BWP billion)	0.98	4.05	1.12	4.19
Year 1 Discounted Cash flow (BWP billion)	36.79	23.47	37.14	24.75
Discounted Payback Period (months)	0.36	2.94	0.31	2.13

4.5.5 Internal Rate of Return

Microsoft Excel IRR function was used to determine the rate of return for each system that makes the present value of cash flows equal to the initial investment. Table 28 shows that the IRR for the TSHS in ore and waste is 147% while that of IPCC in ore and waste is 185%. These are higher than the company's hurdle rate of 15% and show that both systems are economically feasible for use in materials handling. Table 28 also shows that IPCC is more economic than TSHS due to its higher rate of return.

Table 28 Internal Rate of Return for TSHS and IPCC Haulage Systems

Material Transported	TSHS	IPCC
Ore	1,964%	2,183%
Ore and Waste	147%	185%

It is noted that the results of the NPVs and IRRs are higher than the base values while the values of DPPs are very short due to the large differences in cash outflows and cash inflows. This is so because all the returns from Cut 7 and 8 ore production are used to cover all Jwaneng Mine's costs. In this analysis, the NPVs, DPPs and IRRs only cover materials handling costs, hence the high return rates, shareholders wealth and very short payback periods.

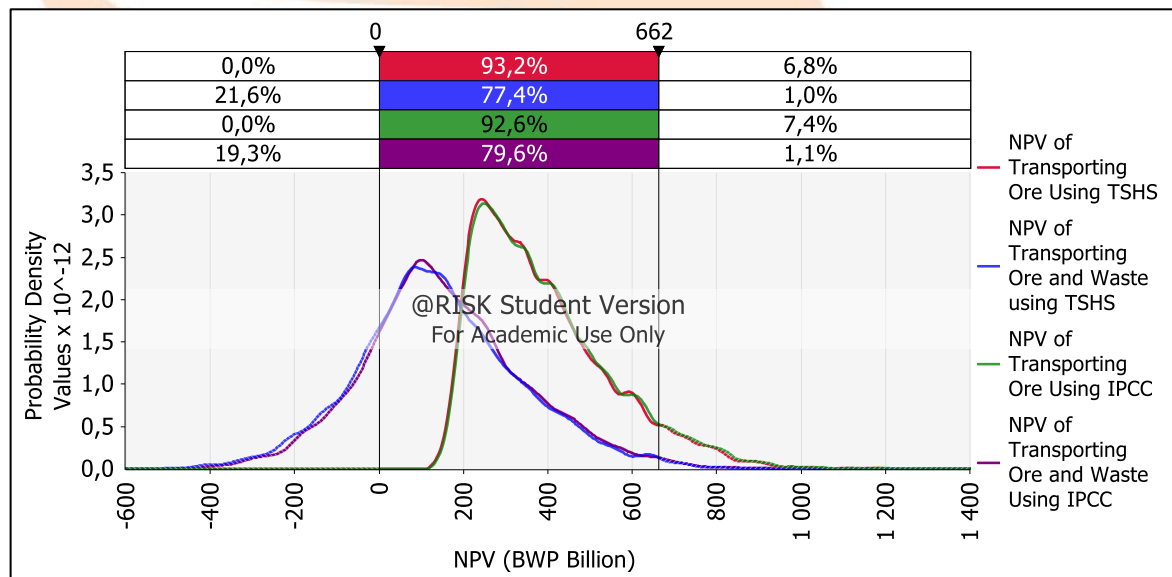
4.6 Risk and Sensitivity Analyses

Mining projects face many risks due to the large amounts of capital investments and operating costs involved. There are usually changes in production parameters with time as the market prices, government legislations and many other variations that occur during the mine's life. These increase the risk of investors not achieving the expected returns on their investment in the mining project. To characterise the risks involved, risk analysis was done by conducting a probabilistic simulation on the NPV of each haulage system using Monte Carlo Simulation with the @RISK software. The simulation used 10,000 iterations for each system to produce a probabilistic graph showing confidence levels of the NPV with varying assigned input values. The input values that were used in the simulation are summarised in Table 29. The range of variability for the input variables were chosen based on Jwaneng Mine's production reports and literature review. To determine the key drivers to the model, a sensitivity analysis was conducted using @RISK software and the resulting Spider graphs plotted.

Table 29 Input Variables Used in Risk and Sensitivity Analyses

Input Variables	Minimum	Most Likely	Maximum
Discount rate	12%	15%	18%
Ore Production rate (tons/day)	33,500	34,000	35,000
Number of working days	354	355	357
Cost of dump trucks (BWP million)	60	61	65
Average cost per carat (BWP)	1,446	1,707	2,175
Average Ore grade (cpht)	70	130	250

The probability density of the NPV of TSHS and IPCC is shown in Figure 21 while Table 30 summarises the possible variations of the NPV over the life of production using 10,000 iterations of the input variables.

**Figure 21 Probability Density of NPV for TSHS and IPCC Haulage Systems**

From Figure 21, there is 21.6% probability that the NPV for ore and waste haulage using TSHS will be negative while that of IPCC system is 19.3%. From Table 30 the minimum NPV for ore handling alone over the life of mine is BWP 114.10 billion for the TSHS while that of the IPCC system is BWP 116.40 billion. Both NPVs are positive which shows that both projects will be viable even in the worst-case scenarios of the input variables. However, the IPCC system is better than the TSHS in the worst case scenario as it has a higher NPV than TSHS.

Table 30 NPV of TSHS and IPCC Haulage Systems

	TSHS		IPCC	
	NPV of Ore (BWP billion)	NPV of Ore and Waste (BWP billion)	NPV of Ore (BWP billion)	NPV of Ore and Waste (BWP billion)
Minimum	114.10	-588.10	116.40	-544.70
Maximum	1,223.00	998.80	1,238.00	1,007.00
Mean	387.00	144.80	392.50	156.40
Std Dev	159.20	197.30	161.40	194.30
20%	246.60	-8.04	250.40	5.58
50%	353.20	130.50	358.20	140.00
70%	446.70	228.30	453.00	237.70
80%	513.40	302.30	520.80	312.00
90%	606.50	401.90	615.10	411.00
99%	862.70	660.70	874.00	668.90

It can be observed from Table 30 that both projects will not be viable when the discount rate is 12% when handling both ore and waste as the NPV values are negative. Table 30 also shows that at the mean and maximum discount rates the IPCC has higher NPVs than TSHS option.

A sensitivity analysis was conducted using @RISK software for both systems. Spider graphs were plotted to show the percentage change in the input values against the change in the output from the base output value. Figures 22 and 23 are spider graphs of NPV of the TSHS and IPCC systems respectively. Both graphs show that the NPVs are highly affected by variations in ore production rate, moderately affected by the price of diamonds and ore grade, and inversely affected by discount rate. This is because ore production rate has the steepest gradients in both graphs.

From Figure 22, a 5% change in ore production will give a NPV of BWP 203.80 billion while a 95% change in ore production will give an NPV of BWP 712.05 billion for the TSHS. However, Figure 23 shows that a 5% change in ore production will result in an NPV of BWP 201.15 billion while a 95% change in ore production will give an NPV of BWP 702.56 billion for IPCC system.

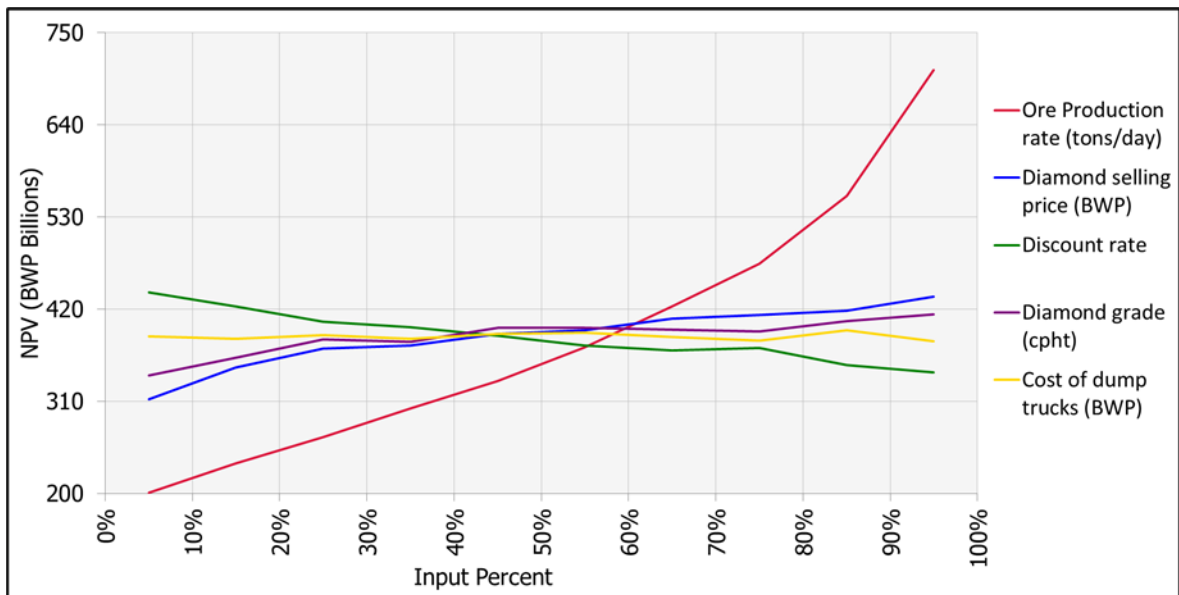


Figure 22 Spider Graph of NPV of Ore Haulage Using TSHS

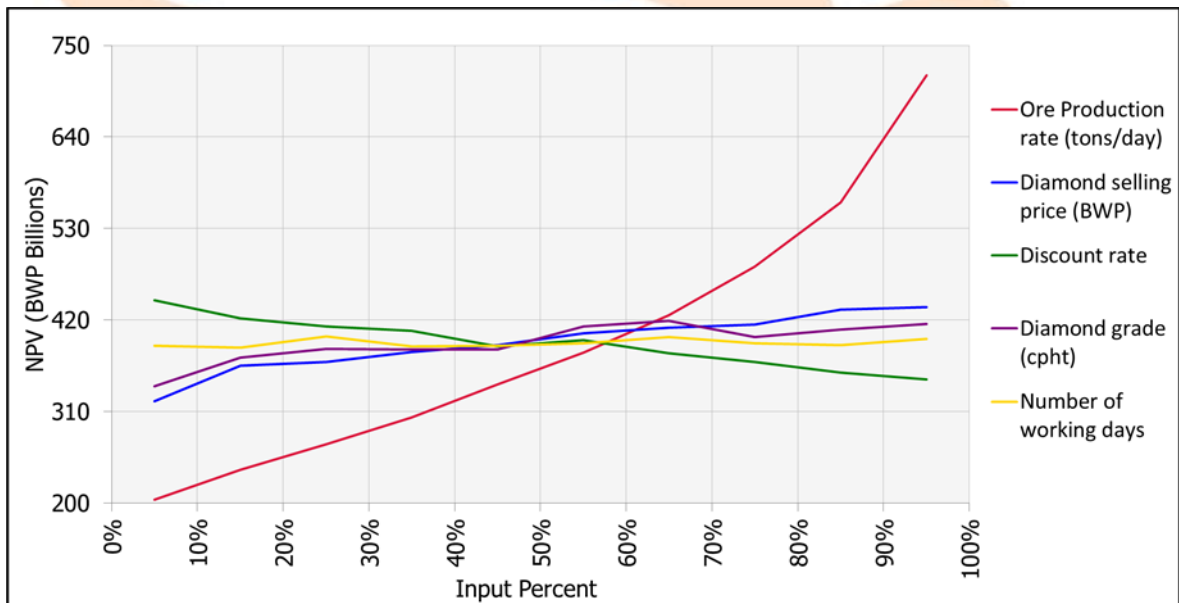


Figure 19 Spider Graph of NPV of ore Haulage Using IPCC Haulage System

CHAPTER 5

CONCLUSIONS AND RECOMMENDATIONS

5.1 Conclusions

Based on the work carried out and the results obtained, the following conclusions are drawn:

- i. The IPCC haulage system significantly reduces the number of dump trucks compared to TSHS. The results shows that a total of four dump trucks will be required to start production when using TSHS and this number will increase as mining progresses to seven dump trucks at the end of mine life. However, the IPCC method will require two dump trucks at the beginning of production. This number is constant throughout production until the last benches when three dump trucks will be required.
- ii. The IPCC haulage system requires high capital costs at the beginning of operation than TSHS. However, its operating costs are much lower than those of the TSHS system. For example, for ore haulage the capital costs of IPCC were BWP 0.14 billion higher than TSHS but its operational costs were BWP 33.29 billion lower than TSHS.
- iii. For ore haulage, the unit haulage costs of IPCC increased by 42% from the beginning of production at Bench #8 to the end of production at Bench #40 while unit haulage costs of the TSHS increase by 72%. This shows that the haulage costs of the IPCC system do not increase as much as the TSHS as the pit deepens.
- iv. The NPV of IPCC system was calculated to be BWP 282.25 billion while that of the TSHS was BWP 278.83 billion. As the NPVs of both systems are positive, it shows that both systems are viable to handle materials in the pit. However, it is noted that the IPCC is more economical than TSHS as it has a higher NPV than the TSHS.
- v. The results show that the discounted payback period of the TSHS and IPCC systems in ore and waste are 2.94 months and 2.13 months respectively. This indicates that both systems are acceptable as they have very short DPPs. However, IPCC is considered more economic than TSHS due to its shorter DPP.
- vi. The internal rate of return of the TSHS and IPCC systems in ore and waste are 147% and 185% respectively. These are higher than the company's hurdle rate of 15% and show that both systems are economically feasible for use in materials handling.

- vii. The results shows that the NPVs and IRRs are higher than the base values while the values of DPPs are very short due to the large differences in cash outflows and cash inflows. This is so because all the returns from Cut 7 and 8 ore production are used to cover all Jwaneng Mine's costs. In this analysis, the NPVs, DPPs and IRRs only cover materials handling costs, hence the high return rates, shareholders wealth and very short payback periods.
- viii. The risk analysis shows that there is a probability of 21.6% that the NPV of TSHS will be negative while that of IPCC system is 19.3%.
- ix. The results of the sensitivity analysis show that the NPVs of the two options are highly affected by changes in ore production rate, moderately affected by the price of diamonds and ore grade, and inversely affected by discount rate. The impact of variations of these input variables on the NPV are higher on TSHS than for IPCC system. For example, a 5% change in ore production costs will result in the NPV of TSHS being BWP 7.65 billion higher than that of IPCC and a 95% change in ore production rate results in the NPV of TSHS being BWP 9.49 billion higher than IPCC. This shows that IPCC haulage system is a better material handling system than TSHS.

5.2 Recommendations

Based on the results of this work, it is recommended that;

- i. The IPCC should be adopted by mine planners for use in deep open pit mines because it has a higher NPV and has lower risks than the TSHS. The IPCC system is also less affected by changes in the input variables than the TSHS.
- ii. Further studies should be done to determine how the IPCC system will behave against the TSHS in transporting waste out of the pit and how the two haulage systems will vary with changing production rates.
- iii. Further studies should be done to compare the economic viability of the two haulage systems considering all the mine's capital and operational costs.

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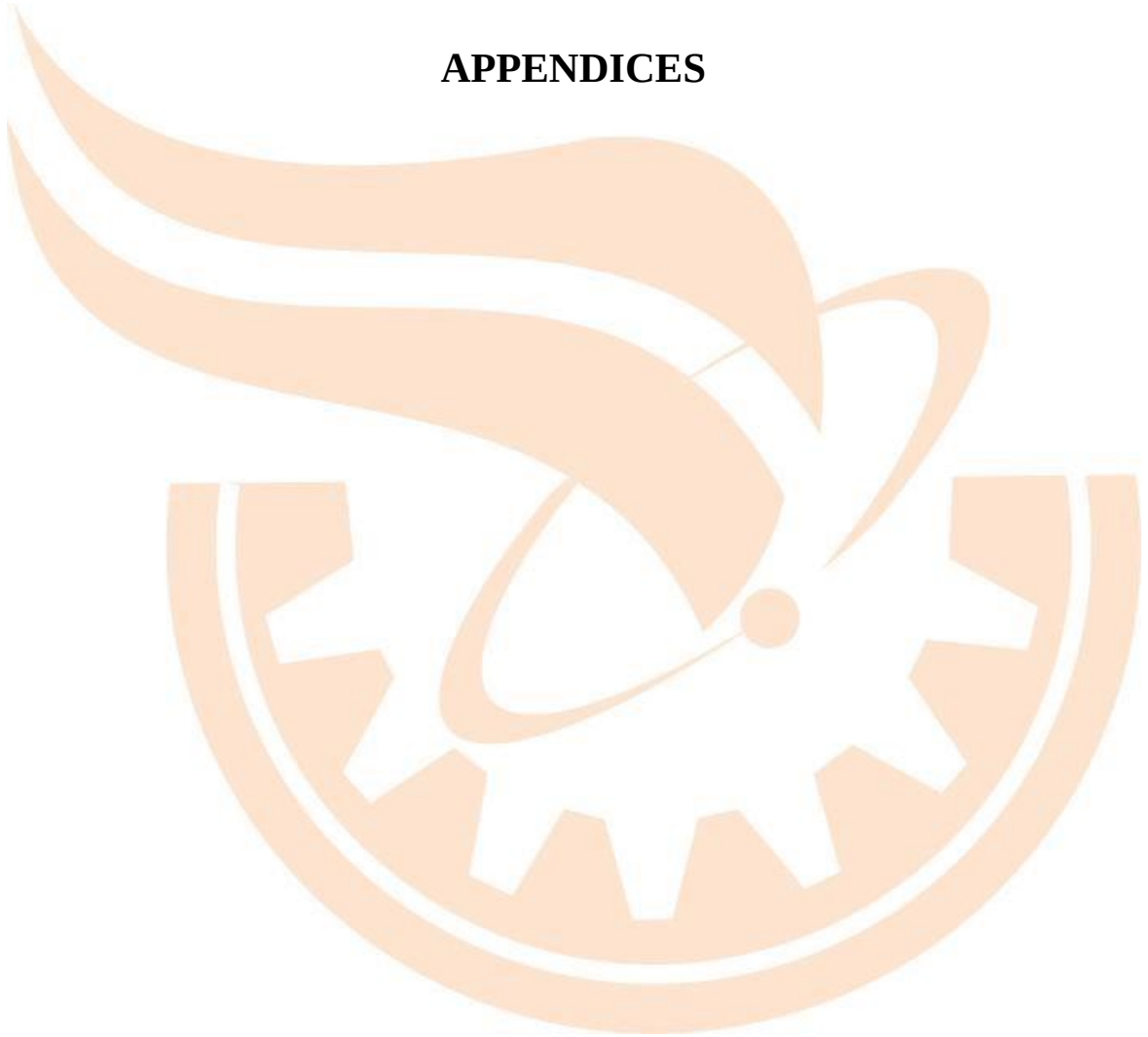
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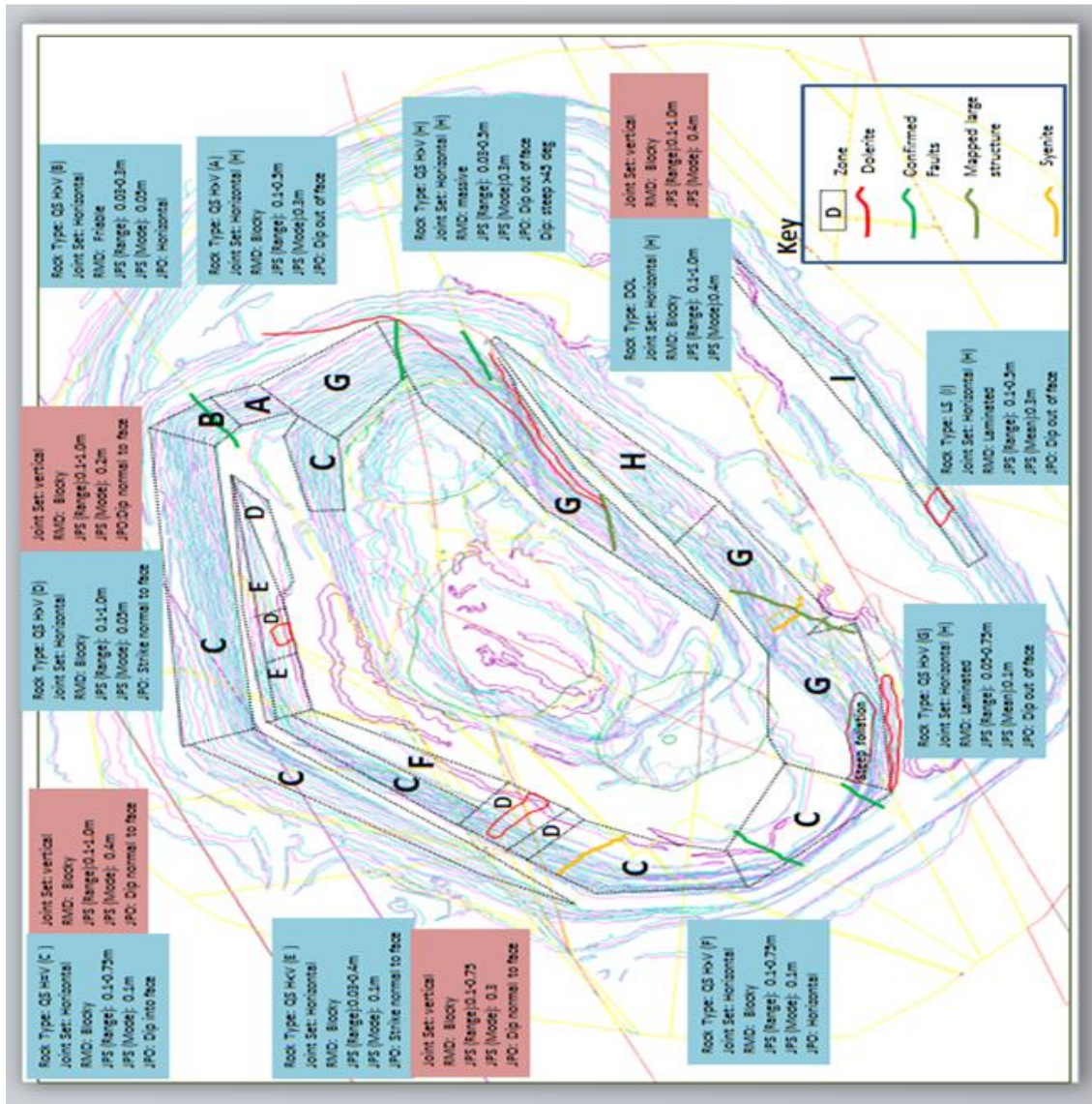


APPENDICES



APPENDIX 1

ROCK MASS PROPERTIES AT JWANENG MINE DK2 PIT



Source: (Goddie, 2014)

APPENDIX 2

PARAMETERS OF PRIMARY CRUSHERS

Crusher		Jaw	Gyratory	Roll Crusher	Impact	Feeder Breaker	Sizer	Hybrid
Year introduced		1858	1883	1910	1920	1960	1979	2005
Mechanical reduction method		Compression	Compression	Compression, impact & shear (for single roll)	Impact, attrition, shear	Compression, shear, impact	Shear, compression	Compression
Moisture content (%)		< 5	< 5	> 20	< 10	> 20	< 20	> 20
Application for high clay materials		Poor - fair	Poor	Good	Poor	Fair	Excellent	Very good
Abrasiveness		High	High	Low	Not applicable	Low	Low - medium	Low - medium
Fine generation		Low - medium	Low - medium	low	high	Low - medium	low	Low
Max. capacity (t/h)		1,250	10,940	14,000	4,500	6,000	12,500	12,000
Material compressive strength (MPa)		450	600	150	115	50	200	300
Max. feed size (mm)		1,500	1,830	1,600	3,000	1,500	2,000	2,500
Reduction ratio		1:4 - 1:9	1:3 - 1:8	1:5 - 1:10	1:10 - 1:50	1:2 - 1:4	1:2 - 1:4	1:4 - 1:6
Max dimensions (mm)	Height	5,400	10,800	3,500	8,100	2,000	1,800	2,000
	Length	5,200	6,450	9,700	5,500	6,500	10,100	9,300
	Width	4,200	6,250	8,200	5,700	4,500	4,050	7,000
Max. weight (t)		115	530	230	190	50	190	102
Max. installed power (kW)		400	1,200	2,000	2,800	300	1,200	2,500

APPENDIX 3

DUMP TRUCK AVERAGE CYCLE TIME PER BENCH - WASTE

Bench	Depth (m)	Average Flat Distance	Average Uphill Distance	Loaded Flat	Loaded Uphill	Empty Flat	Empty downhill	Spotting & queuing	Loading	Dumping	Total Waste Cycle Time (min)
8	112	890	4,586	1.34	18.34	1.34	9.17	4	3.00	1.03	38.22
9	128	872	4,746	1.31	18.98	1.31	9.49	4	3.00	1.03	39.12
10	144	854	4,906	1.28	19.62	1.28	9.81	4	3.00	1.03	40.03
11	160	837	5,066	1.25	20.26	1.25	10.13	4	3.00	1.03	40.94
12	176	819	5,226	1.23	20.90	1.23	10.45	4	3.00	1.03	41.84
13	192	801	5,386	1.20	21.54	1.20	10.77	4	3.00	1.03	42.75
14	208	783	5,546	1.17	22.18	1.17	11.09	4	3.00	1.03	43.66
15	224	765	5,706	1.15	22.82	1.15	11.41	4	3.00	1.03	44.56
16	240	748	5,866	1.12	23.46	1.12	11.73	4	3.00	1.03	45.47
17	256	730	6,026	1.09	24.10	1.09	12.05	4	3.00	1.03	46.38
18	272	712	6,186	1.07	24.74	1.07	12.37	4	3.00	1.03	47.28
19	288	694	6,346	1.04	25.38	1.04	12.69	4	3.00	1.03	48.19
20	304	676	6,506	1.01	26.02	1.01	13.01	4	3.00	1.03	49.10
21	320	659	6,666	0.99	26.66	0.99	13.33	4	3.00	1.03	50.00

22	336	641	6,826	0.96	27.30	0.96	13.65	4	3.00	1.03	50.91
23	352	623	6,986	0.93	27.94	0.93	13.97	4	3.00	1.03	51.82
24	368	605	7,146	0.91	28.58	0.91	14.29	4	3.00	1.03	52.72
25	384	588	7,306	0.88	29.22	0.88	14.61	4	3.00	1.03	53.63
26	400	570	7,466	0.85	29.86	0.85	14.93	4	3.00	1.03	54.54
27	416	552	7,626	0.83	30.50	0.83	15.25	4	3.00	1.03	55.44
28	432	534	7,786	0.80	31.14	0.80	15.57	4	3.00	1.03	56.35
29	448	516	7,946	0.77	31.78	0.77	15.89	4	3.00	1.03	57.25
30	464	499	8,106	0.75	32.42	0.75	16.21	4	3.00	1.03	58.16
31	480	481	8,266	0.72	33.06	0.72	16.53	4	3.00	1.03	59.07
32	496	463	8,426	0.69	33.70	0.69	16.85	4	3.00	1.03	59.97
33	512	445	8,586	0.67	34.34	0.67	17.17	4	3.00	1.03	60.88
34	528	427	8,746	0.64	34.98	0.64	17.49	4	3.00	1.03	61.79
35	544	410	8,906	0.61	35.62	0.61	17.81	4	3.00	1.03	62.69
36	560	392	9,066	0.59	36.26	0.59	18.13	4	3.00	1.03	63.60
37	576	374	9,226	0.56	36.90	0.56	18.45	4	3.00	1.03	64.51
38	592	356	9,386	0.53	37.54	0.53	18.77	4	3.00	1.03	65.41
39	608	338	9,546	0.51	38.18	0.51	19.09	4	3.00	1.03	66.32
40	624	321	9,706	0.48	38.82	0.48	19.41	4	3.00	1.03	67.23

APPENDIX 4

DUMP TRUCK AVERAGE CYCLE TIME PER BENCH - ORE

Bench	Depth (m)	Average Flat Distance	Average Uphill Distance	Loaded Flat	Loaded Uphill	Empty Flat	Empty Downhill	Spotting & queuing	Loading	Dumping	Total Ore Cycle Time (min)
8	112	890	2,716	1.34	10.86	1.34	5.43	4	3.00	1.03	27.00
9	128	872	2,876	1.31	11.50	1.31	5.75	4	3.00	1.03	27.90
10	144	854	3,036	1.28	12.14	1.28	6.07	4	3.00	1.03	28.81
11	160	837	3,196	1.25	12.78	1.25	6.39	4	3.00	1.03	29.72
12	176	819	3,356	1.23	13.42	1.23	6.71	4	3.00	1.03	30.62
13	192	801	3,516	1.20	14.06	1.20	7.03	4	3.00	1.03	31.53
14	208	783	3,676	1.17	14.70	1.17	7.35	4	3.00	1.03	32.44
15	224	765	3,836	1.15	15.34	1.15	7.67	4	3.00	1.03	33.34
16	240	748	3,996	1.12	15.98	1.12	7.99	4	3.00	1.03	34.25
17	256	730	4,156	1.09	16.62	1.09	8.31	4	3.00	1.03	35.16
18	272	712	4,316	1.07	17.26	1.07	8.63	4	3.00	1.03	36.06
19	288	694	4,476	1.04	17.90	1.04	8.95	4	3.00	1.03	36.97

20	304	676	4,636	1.01	18.54	1.01	9.27	4	3.00	1.03	37.88
21	320	659	4,796	0.99	19.18	0.99	9.59	4	3.00	1.03	38.78
22	336	641	4,956	0.96	19.82	0.96	9.91	4	3.00	1.03	39.69
23	352	623	5,116	0.93	20.46	0.93	10.23	4	3.00	1.03	40.60
24	368	605	5,276	0.91	21.10	0.91	10.55	4	3.00	1.03	41.50
25	384	588	5,436	0.88	21.74	0.88	10.87	4	3.00	1.03	42.41
26	400	570	5,596	0.85	22.38	0.85	11.19	4	3.00	1.03	43.32
27	416	552	5,756	0.83	23.02	0.83	11.51	4	3.00	1.03	44.22
28	432	534	5,916	0.80	23.66	0.80	11.83	4	3.00	1.03	45.13
29	448	516	6,076	0.77	24.30	0.77	12.15	4	3.00	1.03	46.03
30	464	499	6,236	0.75	24.94	0.75	12.47	4	3.00	1.03	46.94
31	480	481	6,396	0.72	25.58	0.72	12.79	4	3.00	1.03	47.85
32	496	463	6,556	0.69	26.22	0.69	13.11	4	3.00	1.03	48.75
33	512	445	6,716	0.67	26.86	0.67	13.43	4	3.00	1.03	49.66
34	528	427	6,876	0.64	27.50	0.64	13.75	4	3.00	1.03	50.57
35	544	410	7,036	0.61	28.14	0.61	14.07	4	3.00	1.03	51.47
36	560	392	7,196	0.59	28.78	0.59	14.39	4	3.00	1.03	52.38
37	576	374	7,356	0.56	29.42	0.56	14.71	4	3.00	1.03	53.29
38	592	356	7,516	0.53	30.06	0.53	15.03	4	3.00	1.03	54.19
39	608	338	7,676	0.51	30.70	0.51	15.35	4	3.00	1.03	55.10
40	624	321	7,836	0.48	31.34	0.48	15.67	4	3.00	1.03	56.01