University of Alberta

GEOMETRICAL MINE DESIGN AND MULTI-BENCH MATERIAL FLOW SIMULATION FOR AFS CHARACTERIZATION

by

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ABSTRACT

The economic extraction of the Athabasca oil sands is governed by the application of advanced technologies to reduce production cost, which is currently about 600% that of the cost of conventional oil. The pursuit of low-cost, bulk production methods in oil sands mining has led to the need to extend hydrotransport systems to the mining faces using flexible pipeline/belt conveyor assemblies called "At Face Slurrying" (AFS) technology. This technology will introduce a unique set of geometrical, material extraction and economic problems. Dynamic evolution of mine layouts is essential to developing efficient continuous extraction methods in surface mining. This is a complex process because of the underlying field constraints and the need to meet current production targets under a minimum cost envelope. Conventional mining practices have simplified the extraction processes and layouts using discrete functions which are appropriate for cyclic operations. However, in continuous AFS operations, discrete approximations may lead to significant errors in the geometric layouts of equipment components, and strategic and tactical mine plans.

This research initiative is a pioneering effort to develop continuous ellipsoid and spheroid processes for characterizing material extraction using the novel AFS technology. The dynamic evolution of material extraction is developed using 3D ellipsoid and spheroid processes that result in a system of parabolic partial differential equations with the appropriate boundary conditions. Continuous simulation of these stochastic processes with their field constraints is carried out within Visual SLAM, Matlab, Simphony and ADAMS simulation environments. The models are validated using data from Syncrude's operations in Fort McMurray with detailed experimentation in a laboratory environment. The continuous processes are compared with the current cyclic method at Syncrude.

The results of economic, dynamic and kinematic evaluation of the AFS technology show that it could result in a 44% reduction in the current production cost from $13.78/bbl to $7.72/bbl, with a strong long-term economic outlook compared to the 5-year industry target of $9.00/bbl. *The main novelty of this research was the development of continuous-time ellipsoid and spheroid processes for continuous surface mining operations, which provide solutions to strategic and tactical plans for materials handling with the AFS technology.*

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NOMENCLATURE

$\vec{a}_i$ acceleration of the center of mass of link *i*

$\dot{\omega}_i$ angular acceleration of the coordinate system (x_i, y_i, z_i) with respect to the base coordinate system (x_0, y_0, z_0)

ω_i angular velocity of the coordinate system (x_i, y_i, z_i) with respect to the base coordinate system (x_0, y_0, z_0)

$\dot{v}_i$ linear velocity of the center of mass of link *i*

$\vec{s}_i$ position of the center of mass of link *i* from the origin of the coordinate system (x_i, y_i, z_i)

$\Delta V'_{ij}$ volume excavated with each incremental pushback on any bench, m^3

ω angular velocity

θ bench slope angle, degrees

α dip of the deposit, degrees

γ inclination of belt conveyor, degrees.

β overall pit slope angle, degrees

ν, κ coefficients of viscous and coulomb friction respectively

α_1 angular acceleration

ρ bulk density of material, m^3/hr, (ft^3/hr)

ρ_f density of carrying fluid, m^3/hr, (ft^3/hr)

τ_j input torque/force for joint *j*

$\varphi_{i,n}(x, y, z, a_i)$ the toe co-ordinate set at i^{th} node point and j^{th} bench and function of x, y and z co-ordinates

$\varphi_{i,n}^x$, $\varphi_{i,n}^y$, $\varphi_{i,n}^z$; $\delta_{i,n}$ (x, y, z, a_i) the crest co-ordinate set at i^{th} node point and j^{th} bench and function of x, y and z co-ordinates; $\delta_{i,n}^x$, $\delta_{i,n}^y$, $\delta_{i,n}^z$

ρ_m density of the mixture, m^3/hr, (ft^3/hr)

ρ_{SP} density of stockpile material, kg/m^3

ρ_W density of waste material, kg/m^3

Δx incremental pushback, m

A equipment availability, %

a length of pit in north-south direction, m

A_b cross-sectional area of conveyor, m

A_{cf} availability of slurrification facility, %

a_f final length of pit along the major axes of the elliptical cross-section, m

a_o initial length of the major axes of the ellipse at the ultimate pit limit, m

A_s availability of shovel, %

B bench width that enables double back-up loading of trucks, m

b length of pit in north-south direction, m

b_b contact or "wetted" perimeter, m

B_b width of conveyor belt, m

b_f final length of pit along the minor axes of the ellipse at the ultimate pit limit, m

b_i viscous damping coefficient for joint *i*

b_o initial length of the minor axes of the elliptical cross-section, m

b_{rw} width of safety berm, m

C_{cov} capacity of conveyor belt train, tonnes/h

C_{hyd} capacity of hydrotransport system, tonnes/h

c_i Coulomb friction constant for joint *i*

CI_t capital investment at time t, $

CL clearance between trucks/safety berm, m

C_T cycle time of trucks, s

DR dumping radius of loader, m

F_C production capacity of fleet of trucks, t/h

f_i force exerted on link *i* by link *i-1* at the coordinate frame (x_{i-1}, y_{i-1}, z_{i-1}) to support link *i* and the links above it

F_i total external force exerted on link *i* at the center of mass

F_S fleet size of shovels

g acceleration due to gravity, m/s^2

g_o cut-off grade of mine, g/t or %

g_t grade of run-of mine ore at time t, g/t or %

H bench height, m

H_d depth of deposit below surface, m

H_S vertical reach of shovel, m

H_{ult} ultimate pit depth, m

i i^{th} bench

ICI initial capital investment, $

I_i inertia matrix of link *i* about its center of mass with reference to the coordinate system (x_0, y_0, z_0)

k slope width after projecting length of bench slope to plan view = $H/\tan\theta$, m

K_t force acting on component arms of CycEx CBCS, N

L_{AFS} length of AFS train, m

L_{cf} length of slurrification unit, m

L_{cw} length of one conveyor belt wagon, m

L_{fp} length of one flexible pipeline arm unit, m

L_{hc} length of hopper-crusher unit, m

$L_{M\text{-}AFS}$ maximum length of AFS train, m

L_N and L_E are the respective maximum cut lengths in the north-south and east-west directions, m

L_{PB} length of property boundary, m

L_{PBE} length of property boundary in east-west direction, m

L_{PBN} length of property boundary in north-south direction, m

L_{sh} length of shovel, m

M_h height of loading machine, m

m_i total mass of link *i*,

n total number of benches in open pit

NE_E number of loading equipment in active faces in the east-west direction

NE_N number of loading equipment in active faces in the north-south direction

NF_E number of concurrent active faces in the east-west direction

NF_N number of concurrent active faces in the north-south direction

n_i moment exerted on link *i* by link *i-1* at the coordinate frame (x_{i-1}, y_{i-1}, z_{i-1}),

N_i total external moment exerted on link *i* at the center of mass

n_t number of trucks spotting at same time to be loaded

OC_t operating cost at time t, \$

P_c horsepower required to drive conveyor belt, hp

p_i origin of the i^{th} coordinate frame with respect to the *i-1*th coordinate system

P_m price of mineral on world market, \$/g, \$/tonne, \$/m^3

P_t price of mineral used in mine economic evaluations, \$/g, \$/tonne, \$/m^3

q_i displacement of joint *i* at the coordinate frame (x_{i-1}, y_{i-1}, z_{i-1})

Q_m Mine production capacity, m^3/day, tonnes/day

Q_p Processing plant capacity, m^3/day, tonnes/day

R discount rate, %

R_{AFS} radius of production envelope of AFS train, m

REV_t operating income at time t, \$

R_g	Geotechnical properties of the rock
RLF	Radius of level floor, m
SF	Safety factor
SR	stripping ratio at time t
T	Mine life, yr
t	time at which the cut was made
T	torque on components of CycEx CBCS system.
T_O	tonnage of ore mined
T_{SP}	tonnage of stockpile material mined
T_W	tonnage of waste mined
TW	width of largest truck, m
U	equipment utilization, %
U_b	non-dimensional cross-sectional area shape factor of belt conveyor
U_{cf}	utilization of the slurrification unit, %
UC_O	unit operating costs for ore mining, \$/t (\$/m^3)
UC_{SP}	unit operating costs for mining stockpile material, \$/t (\$/m^3)
UC_W	unit operating costs for waste stripping, \$/t (\$/m^3)
U_s	utilization of the shovel, %
U_s	utilization of the shovel, %
V_b	speed of conveyor belt, m/s
V_{ij}	volume of material extracted from i^{th} bench, m^3
V_O	volume of ore material, m^3
V_T	total volume of ore material, m^3
z_p	generic length of pit in any direction

CHAPTER 1

INTRODUCTION

1.0 Overview

The evolution of surface mine layout configuration is a function of the schedules and sequence of material excavation and haulage within the optimized layouts. The evolution is also constrained by the technological, technical, economic, safety and operational factors that ensure total system ergonomics, efficiency and timely materials deliverability. These mine layout configurations must accommodate all the resources (production, ancillary equipment and personnel) required to achieve periodic production targets. The main challenge in new mining methods for bulk production capacities is to design the layouts and their subsequent evolution to sustain long-term production capacities with minimum engineering risks.

1.1 Background of the Problem

The primary extraction of the Athabasca oil sands uses surface mining technology with the shovel-truck production system. This system has several advantages which include production system flexibility, equipment mobility, equipment-centered availability and operating system efficiency. However, due to the ever-increasing costs of labor and energy (e.g. petroleum products) and as production faces advance further away from the extraction plants resulting in increasing haulage distances, significant production cost increases have emerged with the shovel-truck system. Production cost and efficiency optimization for oil sands is a key component of the strategic initiative to secure North America's energy supply in this century. Current oil sands production cost is about \$13.78/bbl compared with \$1.25/bbl for conventional crude oil (Anon., 2001). It is therefore necessary to reduce the unit operating costs in oil sands mining to make it competitive on the energy market. The effort to reduce production cost has focused on several fronts including new mining methods and advanced production technologies. Research and industry initiatives have focused on new production technologies to reduce truck haulage costs because these constitute a significant portion of the production cost ($\approx$ 26%), and together with shovels, they comprise 40% of the total cost (Bishop, 1968; Michaelson, 1979; Sullivan, 1990; Anon., 1993). In addition, the use of large diesel-powered dump trucks in oil sands mining leads to poor traction and high rolling resistances especially in the summer months and high emission levels of carbon dioxide (CO_2), oxides of nitrogen (NO_x), sulphur dioxide (SO_2) and other gaseous and particulate pollutants. Large trucks which carry higher loads require more efficient drive trains. However, it has been noted that for trucks with payloads greater than 150 tons, the payload-to-deadweight ratio tends to be lower than 1.6:1, meaning that whether empty or loaded, large (> 150 t) trucks actually move more deadweight than payload. This results in higher energy costs for larger trucks (Kutschera, 1994). Thus the general trend towards larger trucks with low payload-to-deadweight ratio to cater for the long haulage distances is not necessarily the answer to higher outputs in most material handling operations. These problems have necessitated the search for appropriate mining methods and materials handling systems to reduce production costs and increase efficiency within sustainable environments.

It has been noted that belt conveyor and hydrotransport systems are superior to the other bulk material handling systems in the transportation of materials under a wide variety of conditions (Anon., 1979; Michaelson, 1979; Frizzell and Martin, 1992). Due to the ever-increasing costs of labor and energy (e.g. prices of liquid fuels), belt conveyor and hydraulic transport systems offer competitive advantages in reducing the unit cost of mining and processing oil sands against truck haulage (Kutschera, 1994). Unlike the shovel-truck haulage system, the payload-to-deadweight ratio for bulk transport systems like belt conveyors are generally 3 to 6 times higher than those for trucks. This ratio increases with higher production capacities for belt conveyors. Generally, bulk transport systems have lower operating costs and much lower sensitivities to fluctuations in labor and energy costs.

The current haulage technology at Syncrude uses trucks to haul oil sands materials to a slurrification unit from where the slurry is pumped to the treatment plant through rigid pipelines in the hydrotransport system. The low production cost of bulk transport systems has generated interest in linking the fixed hydrotransport system with flexible pipelines or belt conveyor wagons to the face. The materials excavated by the shovel will be dumped into a hopper, crushed and slurried at the face before being pumped through either rigid or flexible pipelines to join the fixed hydrotransport system. This new technology is termed the "at face slurrying" (AFS) extraction method. The AFS technology is designed to optimize the efficiencies and costs of hydraulic and belt conveyor transport systems by extending the hydraulic transport or conveyor belt systems to production faces. The flexible pipeline or belt conveyor systems will allow the mobile train to adapt to the production face dynamics resulting from shovel excavation. The flexible pipeline or belt conveyor systems on a mobile train will introduce a unique set of mine design layout and configuration. If properly matched or linked, the movements of the mobile conveyor or hydrotransport trains will be synchronized with the shovel movements on the bench to ensure continuous movement of materials with minimum interruptions. To ensure that there is continuous feed of materials from the discrete shovels to the continuous belt conveyor or hydraulic transport systems, a number of shovels will be linked to the slurrification units by shovel/belt conveyor or shovel/flexible pipeline systems.

Currently, the use of hydraulic transportation is limited to fixed network systems, which are far removed from the production faces. In these systems, trucks are used to transport oil sands to fixed stations for slurrification and onward transport to the extraction plant. In order to optimize the hydraulic transport system efficiency and cost, there is the need to extend the hydraulic/bulk transport system to production faces using either a mobile train of flexible pipelines or belt conveyor wagons. The flexible pipeline or belt conveyor systems will allow the mobile train to adapt to the production face dynamics resulting from shovel excavation. The flexible pipeline or belt conveyor systems on a mobile train will introduce a unique set of mine design layout and configuration, soil foundation, equipment and hydraulic transportation problems within the production environments.

The layout design and configuration problems include the initial box cut, face advance and retreat, lower and upper bench interactions within common production envelopes, multi-bench, multi-face production systems and schedules, and sequence of materials extraction. The soil foundation problems include ground pressures, slope stability and degree of abrasiveness of materials. The equipment problems include the sequence of flexible and fixed pipeline or belt conveyor extensions, torsional stresses and bending moments, weight of equipment units and appropriate instrumentation (Changirwa et al., 1998). The hydraulic transportation problems include multi-phase flow characterization, transport efficiency and economics. These problems require appropriate design, rigorous modeling, experimentation and analysis to understand the total system production capacities and efficiency. This research study is focused on detailed mathematical and simulation modeling, experimentation and analysis to provide solutions to the layout design and configuration problems.

1.2 Statement of the Problem

The introduction of the "At Face Slurrying" (AFS) technology in primary oil sands production will be a significant milestone in minimizing costs and maximizing system efficiency. This novel technology, even though plausible on a conceptual basis, has a number of significant problems, which could make it impossible to implement. The conventional opening of an open pit mine is to make an initial box cut with a frustum geometry as illustrated in Fig. 1.1. The critical components (bench height and slope, minimum pit floor space) of the frustum geometry are functions of the production equipment size and operating dimensions, physical and mechanical properties of the material, topographical constraints and other site-specific problems. The entry haulage road is imposed on this layout based on the dump truck dimensions and gradeability within the layout.

The functional relationships among the various dimensions are relatively maintained or modified during incremental pushbacks depending on the material characteristics to meet production requirements and also ensure the safety of personnel and equipment. The pit floor dimensions in the box cut also depend on the dimensions of equipment and type of loading methods employed (e.g. single or double backup, conventional or modified driveby). The main challenge is to create an initial box cut that will accommodate a long train of flexible pipelines or belt conveyor wagons and a mobile slurrification unit. The layout must allow for minimum torsional stresses and bending moments (Changirwa et al., 1998).

The idea of a "borrow" pit for initial housing of the slurry equipment and its introduction at a later stage into the expanded layout must also be investigated to overcome these problems. An important consideration with the borrow pit is the timing associated with assembling and disassembling of the slurry making components. A borrow pit for materials in the initial pit could also be considered, whereby the material required to create enough room for assembling the main components is stored and processed after the system is assembled in the pit layout.

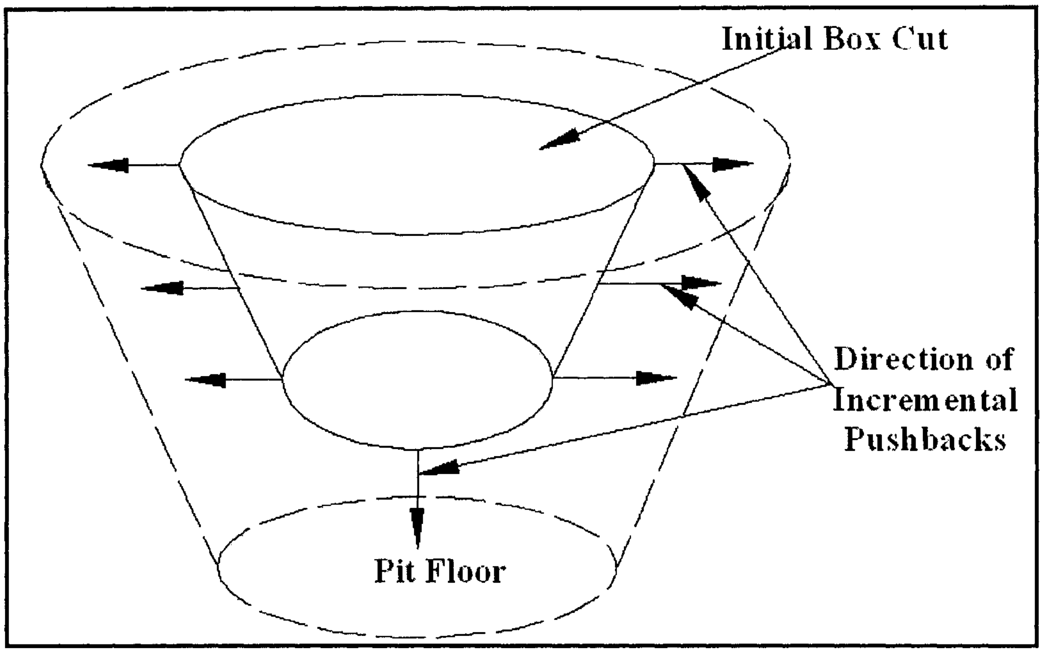


Fig. 1.1 Initial Box Cut and Incremental Pushbacks in Open Pit Mining

The interactions between the shovel and the mobile AFS train during the advance and retreat motions of the former must be well-coordinated to ensure adequate matching of their production capacities and to prevent accidental breakage or damage to the various equipment components. An automatic response system, which triggers the movement of the train in response to shovel movements, may be required for safe AFS operation. The movement of the shovel could occur on both the horizontal (bench movement) and inclined planes (bench-bench interactions). This will require that there is enough room in a circle of influence that may be deemed as a safe operating radius for the AFS production system. This safe operating radius may be determined based on the dimensions and components of the production equipment.

The mining layout configuration must also allow smooth interactions between equipment on lower and upper benches within similar production envelopes. Each AFS production unit will be used to service a production area called the "production envelope" as illustrated in Fig. 1.2.

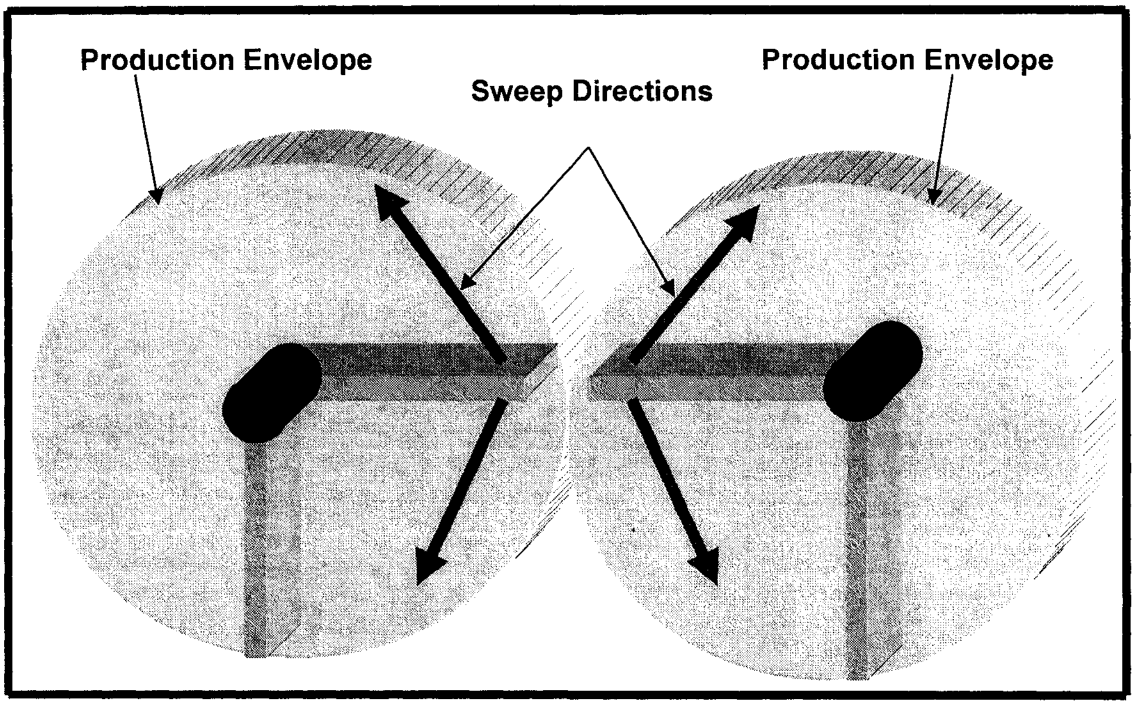


Fig. 1.2 Production Envelopes for two adjacent AFS Trains

The flexible pipeline train or belt conveyor wagon system and the slurrification unit, given a certain locus of attachment to the fixed hydrotransport pipeline, is constrained by its dimensions to certain aerial extent for production service. Beyond this envelope, the production units must be extended or must be serviced by other AFS systems as in two trains. The optimum production envelope that ensures minimum disruptions and maximum production capacity and productivity must be modeled and factored into the design of the layout of the flexible pipelines or conveyor belt wagons and mobile train in the AFS technology. The time to complete a production envelope must be estimated and planned for appropriately within the global production operation. For a number of AFS trains, the associated timing could be staggered in such a way that the units are adequately matched (allowing for surges during production) and production interruptions are minimized in the long term. The timing could be designed based on the production requirements within a shift, week or month depending on the optimum AFS size that will yield appropriate ground pressures and ensure mobility, portability and flexibility.

Reliable and efficient large-scale surface mine production systems are achieved using multi-bench and multi-face operations. In a shovel-truck operating system, this is easily achieved because the trucks are easily dispatched to the loaders and dump positions, and relocated (as required) to ensure smooth operations. In order to achieve reliable and efficient large-scale operations using the AFS technology, the layout design and configuration must allow the use of multi-bench, multi-face production systems within the operating constraints of the equipment. This may be achieved using small and mobile AFS units, which may have small production envelopes, frequent equipment relocations and reduced utilization factors. Another solution may also be the use of the "near face" technology, in which the AFS system is modified by using mobile or shiftable conveyor belt wagons to transfer materials from production faces to a central slurrification unit. The problems of the AFS system then include those of the mobile conveyor belt wagons. The design and layout configuration must ensure that materials from the benches are efficiently loaded onto the belts. Another solution may be the use of low bench height, which allows the gradeability of the AFS train on upper and lower benches and minimum torsional stresses and bending moments. The use of low bench height must also be balanced by the shovel digging height and thus its productivity.

The primary challenge is to design a system that will ensure that production targets are achieved with maximum safety, minimum risks and uncertainties. The production schedules and sequence of materials movement within the layouts from multi-bench and multi-face operations are difficult to plan and execute via a constrained technology. The AFS technology, its major physical components and their interactions with the designed layout environments and the production target determinants, present major challenges to the attainment of production goals. Realistic industrial-scale experiments will be costly and difficult to achieve for this technology. As a result, a production simulator will be required to provide a comprehensive laboratory scale testing of materials flow over a long period of time. The results will then be used to calibrate a scaled-up prototype system to examine the technical feasibility of the system with reduced associated risks and uncertainties.

The overall successful implementation of the AFS system for oil sands production will depend on the optimum layout that ensures total system ergonomics. These layouts must ensure the safe interaction of personnel, production and ancillary equipment in the production operations. The potential sources for ergonomic problems may include pipeline constraints to mobile equipment, mobile train and shovel interaction problems, possible breakages associated with flexible pipelines and potential flooding of the pit.

These problems could also threaten timely production target achievements. Comprehensive studies, equipment instrumentation and testing may be required to understand and/or mitigate their potential impacts on ergonomics and production target achievements.

1.3 Objectives of Study

The primary objective of this research study is to design, test and develop a new surface mining method that will allow the deployment of the AFS technology for oil sands production. The elements of this primary objective include the following:

i). Outline the conceptual AFS models for slurry transportation from production faces.

ii). Develop mathematical models governing the evolution of the surface mine layouts associated with the current mining system and the novel AFS method.

iii). Simulate the various mining options under the underlying field constraints and boundary conditions in a laboratory scale environment.

iv). Develop and test production schedules and sequence of material flow from multi-bench, multi-face surface mining operations.

The AFS technology must achieve periodic production targets, allow smooth interactions of various production and ancillary equipment and it must ensure total system ergonomics. The technology must be applicable all year round under varying field conditions and be environmentally friendly.

1.4 Scope of Study

This study deals with the development of a new mining method associated with the AFS technology. The study will develop a new mining method in which the evolution of the pit geometry will allow the efficient and timely extraction, transportation and slurrification of oil sands by slurrying units located within the pit. It is divided into four major areas including (i) the design, experimentation and development of the layout configurations and evolution of the current mining system and one AFS option; (ii) the development of a bench material flow simulator to simulate production schedules and sequences in continuous-time; (iii) comparative analysis of production-economic functions of the AFS technology versus the conventional shovel-truck haulage system; and (iv) comprehensive quantitative risk modeling of the AFS production system constrained by the layout configuration and its evolution through space and time.

The first area deals with the physical geometry of the mining layout that will accommodate the AFS equipment train. The current mining system (CMS) and one of the conceptual AFS options will be used as basis for this section. The first option focuses on the current mining system which comprises the shovel-truck system with short hauls to the slurrification facility while the second option comprises a shovel-hopper-crusher-belt conveyor-slurrification facility linked to the fixed hydrotransport system (HTS) by either rigid or flexible pipelines.

The second area focuses on developing continuous-time stochastic models of the excavation and handling processes associated with the CMS and the conceptual AFS option within the associated mining layouts. The stochastic models will capture the physical dimensions of the mining options within their working environments, the continuous multi-bench, multi-face materials excavation and handling operations, production and productivity determinants and the associated field constraints. These models will be simulated over an extended period of time, with changing environment covering the operating regimes of the AFS technology within the new mining methods. The results of the stochastic simulation models will be used to provide comprehensive quantitative risk characterization of the CMS and conceptual AFS option, which is the third area of the study. The probabilities associated with the CMS and AFS conceptual options in meeting certain production capacity targets will be generated and analyzed extensively to provide the various AFS efficiency and economic indicators. Finally, the efficiency, economic and risk spectra associated with the conceptual AFS option will be compared with conventional cyclic and continuous production systems of similar capacities.

1.5 Research Methodology

The research study combines a number of methodologies to create scientific and engineering environments for accomplishing the research objectives within the defined scope. Extensive literature survey and production systems review of existing hydraulic transportation systems at Syncrude and Suncor operations at Ft. McMurray have been used to provide a basis for developing the conceptual AFS options. The physical models of the CMS and conceptual AFS options and the associated mining layouts have been developed in Visual SLAM with AweSim, Simphony, Automatic Dynamic Analysis of Mechanical Systems (ADAMS) and Matlab development environments. Mathematical models that capture the evolution of the mining layout configuration have been developed using the 3D ellipsoid of extraction. The continuous processes have been captured using a system of parabolic partial differential equations (PDEs) in the northing, easting and depth coordinate system, and their associated boundary conditions. The PDE models have been numerically solved using implicit finite difference methods. Numerical simulators have been developed in Visual SLAM and Matlab environments and used to simulate the PDE models of the multi-bench, multi-face material flow processes. The CMS and conceptual AFS option were simulated under varying conditions in a laboratory scale environment to generate results for calibrating the simulators.

1.6 Expected Scientific and Industrial Contributions

The primary contribution of this research study is to advance knowledge and frontiers in surface mine layout design, materials excavation and handling engineering. The new knowledge and frontiers will also spur further research activities and technology, which will contribute to industrial growth. This is the first comprehensive research study to examine the design and experimental modeling of the novel AFS mining method for primary oil sands production and the evolution of the layout configuration. In addition, this research study will enhance the collaborative efforts and partnership of the University of Alberta and industry. While the University of Alberta undertook the study with Syncrude Canada Limited (SCL), other oil sands companies will benefit from a successful design and implementation of the technology. This will provide opportunities to the University of Alberta to remain a strong partner with these companies in further research to improve the method and other new research areas that may result from this study.

The AFS technology has the potential to replace the increasingly costly shovel-truck system in oil sands mining, reduce oil sands production costs, increase production capacity and efficiency. Cost reduction is a strategic goal for creating competitive oil sands production enterprises in Alberta and Canada. The study has also advanced the application of network, discrete event and continuous simulation modeling theories in complex industrial engineering systems. The use of the twinned stochastic-optimization modeling techniques to capture the stochastic processes associated with the production system for optimization has also advanced knowledge in this important subject area in production engineering. The numerical simulators could also be modified to simulate other production engineering systems to create understanding and knowledge in other areas of significance to the surface mining industry. If successful, this research study will provide an opportunity for the surface mining industry to reduce the number of large diesel-powered dump trucks. This will lead to significant reductions in the emission levels of gaseous and particulate pollutants resulting in highly sustainable environments. In the age of increased awareness of environmental stewardship, this technology will increase the capacity of companies in meeting regulatory policies and reducing the costs associated with compliance with environmental laws.

1.7 Structure of Thesis

Chapter 1 gives an overview of the research work. It outlines the background and statement of the research problems, the objectives, scope and methodology of the work. As well the expected scientific and industrial contributions of the study are given. Chapter 2 gives a comprehensive literature review of the subject area. It deals with mine planning, scheduling and design of production systems in open pit mines. Materials handling by cyclic, continuous and combined handling systems are elaborated upon. Chapter 3 elaborates on the possible mining methods that can be employed in oil sands mining. The current mining method and one of the most promising AFS options have been exhaustively studied, modeled and simulated to provide solutions to the layout and design of the various AFS configurations. Integral calculus, solid geometry using 3D ellipsoid and parabolic partial differential equations are used to model and analyze the continuous flow of materials by volume in the CMS and conceptual AFS models in Chapter 4 of the report. Chapter 5 covers the computer modeling and verification of the mining models. Chapter 6 contains the AFS performance simulation modeling and analysis while Chapter 7 contains the conclusions, contributions and recommendations of the work. The relevant references in this work arising out of the literature review and Appendices follow Chapter 7. All terms and symbols used in equations in this thesis are defined in the Nomenclature section in the Prefatory pages.

CHAPTER 2

DETAILED LITERATURE SURVEY

2.0 Introduction

This chapter focuses on an extensive literature survey underlying the major work in the areas of surface mining methods and production systems engineering. A critical review that highlights the major problems, methodology, scope, limitations and contributions is carried out to trace the evolution of knowledge and research frontiers in these important areas of mining. Surface mine design, production planning and scheduling involve the geometrical design of the mine layouts, schedule and sequence of extraction of ore and waste blocks, lateral as well as vertical movements of pit faces and scheduling of mine equipment. Bulk materials transport may be achieved by means of continuous and cyclic equipment like belt conveyors, hydraulic transport systems, ships or barges, bucket wheel excavator-conveyor belt systems and bucket ladder dredges.

2.1 Surface Mine Design and Production Planning

The earliest relatively large scale mining for outcropping native copper which occurred some time between 5000 and 15000 BC was the precursor to today's open pit mines. However, large scale open pit mining began in the period 1904 – 1907 when D. C. Jackling designed and developed the Bingham Canyon Copper pit in Utah (Michaelson, 1979). Materials handling systems evolved from human and animal carriage/haulage through steam powered locomotives to diesel, electric or diesel-electric powered trucks, trains and belt conveyors. The capacities of haulage units ranged from less than one tonne side dump rail cars through relatively small rear-dump mechanical drive trucks in the early 1960s to the present day diesel/electric powered large dump trucks of up to 500 tonne capacities.

There have been no basic changes in the fundamentals of early manual pit design methods. However, with the introduction of improved borehole drilling and data logging techniques together with the introduction of computers and modeling software into open pit mine planning and design, there have been phenomenal changes in open pit planning, design and modeling. With the advent of computers and modeling software, highly reliable geological, mineralogical and rock mechanics information needed for open pit design and computerized modeling, simulation, calculating and plotting are easily maintained and accessed. These capabilities, coupled with improved and more detailed capital and operating cost information, can now provide mine operators with frequent and cheaply produced short, medium and long-range alternative pit designs for making mine planning and scheduling decisions (Michaelson, 1979). With increasing stripping ratios, higher energy and labor costs, accurate mine planning, design and scheduling, equipment selection and optimization techniques are required to maintain and improve upon the productivities and efficiencies of the systems, reduce costs and ensure the maximum profitability of the operations.

2.1.1 Open Pit Mine Design and Optimization

The design of an open pit is normally conducted in a series of steps which range from the exploration stage, through the conceptual stage to the design and finally the evaluation stage. The factors involved in open pit design include those that the mining engineer can control such as equipment selection, stripping ratios, production rates, and those which the engineer has no control (e.g. orebody geometry, ore dispersion, allowable slopes; topography and location) (Bohnet, 1989, Wilke et al., 1996). The development of a mine design for an open pit mining operation occurs in three stages. These stages include long-range, intermediate-range and short range plans based on the mineralization inventory of the resource. The mineralization model is built from the borehole data collected during exploration and development drilling and the geological interpretation of the data (Kahle and Scheaffler, 1979).

As open pit mines are generally excavated to greater depths than strip mines, they involve more complex designs and layout. Layout design requires geological data and surface topography, which are critical to the establishment of pit size, layout, production rate and the process flow sheet. It also includes geotechnical and hydrogeological data that affect pit wall stability. Mine optimization is dependent on the interaction of the contributing parameters which leads to maximizing the net present value (Dohm, 1979).

2.1.2 Mine Production Planning and Scheduling

Mine production planning is primarily aimed at ensuring that an optimum production plan is developed for consecutive mining periods to ensure that a resource is extracted in a safe, efficient and profitable manner (Wilke et al., 1996). It comprises several interrelated systems such as excavation and handling, strata control, operating and service support (Bise, 1986). Excavation and handling include the selection of all the required equipment for the economic extraction of the resource from its natural environment and for the subsequent transportation of the excavated material from the mining face. Apart from the strata control system, which deals with the maintenance of the safety and stability of the surface mining environments, the remaining systems mainly supplement the other systems to ensure that mine production targets are met at maximum safety and minimum cost (Bise, 1986).

In the planning stage, the targets for mine production, capital expenditures and unit costs are set at long, medium and short-ranges. The grade and quality standards of the mine and processing plant are also determined with some flexibility built into the plans to meet the ever-changing economic and technological circumstances. Mine production planning aims at securing a uniform and homogenous product throughout the planned period, which meets the processing plant requirements and satisfies the overall profit maximization objective of the mine (Wilke et al., 1996; Smith, 1998). Mine production scheduling allows the scheduling of the waste stripping and ore production to keep the equipment requirements constant. Production scheduling also analyses the production capacity of the existing mine equipment. In this stage, the expected equipment, fleet sizes, projected equipment availabilities and utilizations, planned haulage profiles and conditions, anticipated mining conditions, weather and labor constraints are reviewed (Kahle and Scheaffler, 1979). The overall objective of production scheduling is to maximize the net present value and the return on investment. These measures are derived from the extraction, processing and sale of some commodity from the deposit within the target period (Bohnet, 1989). The method and sequence of extraction of ore and waste blocks, cut off grade and production strategy are affected by many exogenous and endogenous factors. These factors include the physical and chemical characteristics of the mineral, its grade distribution, operating costs, initial capital requirements, economic factors such as commodity prices, market and capital constraints, political and environmental factors (Bohnet, 1989; Wilke et al., 1996).

Production planning and scheduling provide projections of the future mining progress and time requirements for the development and extraction of the resource (Kahle and Scheaffler, 1979). These schedules and plans are used by management: (i) to maintain and maximize the expected profit; (ii) determine the future investment in mining; (iii) optimize the return on investment; (iv) evaluate alternative investment options, and (v) conserve and develop the mine's resources. This may be daily, weekly or monthly schedules. Surface mine production scheduling is complicated by the fact that most open pit mines work with multiple benches and often involve the simultaneous excavation of both ore and waste from a large number of working faces. The ability to maximize the profit or net present value of an orebody is largely dependent on the mining schedule. The mining schedule will determine the life of the mine, the projected cashflows (revenues, operating costs, capital costs), and the investment requirements over the production life of the mine. The effective scheduling of an open pit operation ensures production maximization, production costs minimization; maintaining a correct balance between waste removal and ore mining to ensure smooth stripping ratios; providing adequate working room/faces in the pit; making available exposed ore for mining and if necessary blend; and that reclamation can be effected in a timely manner and at the lowest cost.

Some of the factors considered and delineated in a complete mine plan include shovel mining sequence, (moves, ore and waste schedule), production, number of operating units required; drill sequence, footage requirements by hole size and blast site development. Other factors include the construction, types and lengths of haulage routes, stripping requirements to uncover the ore, and the type and quantity of explosives required based on rock types. As well, dewatering requirements, provision of sufficient power and its efficient distribution to the proposed mining areas; design of waste dumps and their capacities to meet the requirements of the stripping operation; scheduling of ore production sequence to fulfill the needs of the beneficiation plant in terms of time and cut-off grade are factors that are considered.

2.2 Mine Equipment Selection and Scheduling

Most open pit mines operate a multitude of operating faces on different benches. Major mining equipment for both ore and waste removal has to be flexible enough to be capable of rapid redeployment in the mine to suit the demand of the mining schedule. The general trend in open pit mining is to reduce operating costs by using larger and more productive materials handling equipment. Factors which affect equipment selection on a long range basis are the required mine production, haul distance, available operating room, availability and costs of power, fuel, weather conditions, material type, mine life versus capital required for a specific mining system, and the operating characteristics of the equipment (Hendricks and Dahlstrand, 1979). Thus, special attention has to be paid to the selection and scheduling of the mining equipment to ensure maximum productivity and utilization in the mine.

2.3 Conventional Open Pit Mine Planning and Design

The stability of an open pit mine slope is a critical component of the safety of personnel and equipment within the confines of the pit. Pit walls with low bench slopes will increase safety but at increased costs in removing large amounts of waste to mine a given amount of ore. Conversely, steep pit walls reduce the amount of waste stripping required to extract the ore but reduce the safety of the pit walls due to possible failure from ground stresses. In practice the ultimate pit slopes are the steepest possible angles, which minimize the amount of waste material excavated and ensures the safety of personnel and equipment (Hustrulid and Kuchta, 1995). Fig. 2.1 shows the longitudinal section of an open pit layout with three benches showing the initial box cut, incremental pushbacks, bench height and width as well as bench slope angle and the overall pit slope angle.

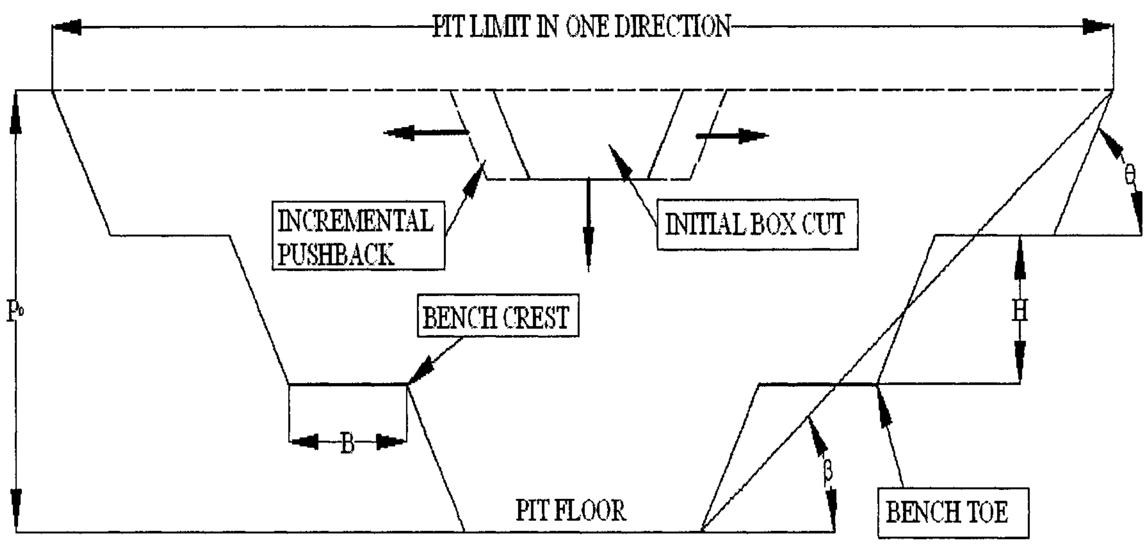


Fig. 2.1 Longitudinal Section of an Open Pit Layout

Depending on the geotechnical characteristics of the pit walls and the perceived instability, the slope of an open pit mine may be analyzed either by stability evaluation on the bench scale or on the overall pit wall. Simplified instability models (dictated by the type and structural control) have been employed over the years to predict the slope geometry of open pit mines. Some of the common forms of instability models, which are amenable to engineering analysis are wedge failure, plane failure, toppling failure, circular failure and other special failures (Hoek and Bray, 1981; Hustrulid and Kuchta, 1995).

Due to the different shapes, sizes, dips, depths below surface, mineralogical composition, geotechnical characteristics of mineral deposits, and the technological and economic conditions, open pit layouts have different geometric shapes. Material characteristics, equipment type and technology greatly influence pit geometry. Key factors that affect the planning and design of an open pit mine, and hence its shape, are the geology, grade and localization of mineralization, aerial extent of the deposit, topography and property boundaries. They also include production rates and costs, bench height, pit slopes, road grades, mining and processing costs, mill recovery, marketing considerations, strip ratios and cut off grades (Robb, 1979; Armstrong, 1990).

Three major components of open pit mine design are bench configuration, inter-ramp angle and overall pit slope angle. The design of open pit slopes involves specification of the functional requirements (i.e. safe access to benches) with acceptable maintenance and economic conditions. It comprises the determination of design sectors and preparation of the appropriate design layouts to satisfy the functional requirements. It also involves the enumeration and preliminary design of all the solutions which could possibly satisfy the functional requirements. In addition, major components of the open pit mine slope design involve stability analysis of the chosen design(s) to estimate probability of failure and the expected volume of failed material for bench, inter-ramp and overall pit slopes, the development of the maximum inter-ramp slopes based on catch bench criteria; and the design of the optimum pit slopes (Keaton and Beckwith, 1996).

The dimensions of an open pit depend on the configuration of the designed benches, ramps and pit wall slopes. The bench configuration comprises the bench height, bench width, width of safety berms and bench slope angles. Their dimensions are usually chosen to ensure the integrity of pit walls safety and stability. The bench height is affected by ore grade and potential for dilution, the required degree of selectivity of ore from waste, production rate, physical and geological characteristics of the deposit, equipment type and size, climatic conditions and safety of personnel and equipment (Armstrong, 1990). The determination of the configuration of an open pit mine is a fundamental requirement for the geometrical mine design and simulation of bench material flow under the AFS method. The exact location of the excavators can be determined by global positioning satellite (GPS) technology. This allows for the determination of the elevation of the face at which the excavator is loading materials, the grade and volume (weight) of the materials being excavated at any particular time (Paine and Wright, 1998). Though the general tendency is to design the bench height as high as possible, the size and type of drilling, loading and haulage equipment may limit the bench height. The vertical reach of the excavator should be able to scale the crest of the working bench, to prevent hang-ups in the upper section of the pit wall and to achieve a smooth wall surface (Hendricks and Dahlstrand, 1979; Robb, 1979). As well, high bench heights in combination with high pit wall slopes could lead to high haul road grades that can seriously affect the performance of the materials transport system (e.g. dump trucks and belt conveyors). The transport of ore or waste out of the pit by means of belt conveyor systems may require the pit to have many benches of low height and flatter bench slopes (Anon., 1988).

The rock conditions and type of mobile equipment used in production operations determine the widths and grades of the haul roads. Haul roads are generally designed in areas where slope angles are gentler with minimum or no stability problems. Spiral or zigzag haul road layouts are commonly used in pits depending on the width, depth and slope angles of the pit walls. Pits with tight walls allow for zigzag road layouts. However, pit walls with high slope angles may not allow for a zigzag layout as the amount of excavation required may increase the stripping ratio tremendously. Hazards due to slope instability could prevent the placement of haul roads on certain sections of the open pit. This applies to sections of a pit where high water pressures exist or the rock mass is characterized as poor. In the latter case a spiral layout is recommended. The length of the haul roads has to be minimized to reduce construction costs and travel time. Well-designed haul roads ensure the safest, quickest and cheapest access to mining operations within the pit. However, regulatory and production constraints may prevent the placement of haul roads in certain areas of the open pit due to road slope limitations and/or surface disturbance regulations.

2.4 Open Pit Mine Planning and Design Software

Large integrated mine planning and design software have been available for over 30 years. These software packages have given mining engineers useful tools for orebody modeling, mine planning, design and scheduling. An integrated system software provides capabilities such as database management, data analysis tools such as statistics, modeling, mine planning, and production scheduling (Gibbs, 2003). The most common computer-based mine design systems available today are Vulcan, Gemcom, Datamine Studio, Whittle Four-X Analyser, Medsystem, Mincom, Lynx/Microlynx, Minex and Surpac Vision. Minex is a geology modeling and mine planning software for coal mining, incorporating efficient modeling, flexible mine planning and advanced innovative solutions. Surpac Vision is a software system for orebody evaluation, open pit and underground mine design and production scheduling. Surpac has drillhole and data interpretation, geology modeling and plotting tools. It includes leading drillhole logging and data management tools (Anon., 2003a). Both Minex and Surpac Vision are produced by Surpac Minex Group (SMG). Apart from its wide usage in geological modeling of orebodies, Surpac can be used to model many variables in the orebody and in grade forecasting (Paine and Wright, 1998). As well, it is able to report on quality data from sized geological or mining blocks within the mine, which enables mine planners to respond to changes in bench geometry and ensure that ore from different mine faces are blended to the required levels before being sent to the processing plant. Vulcan is another software package that has been designed to streamline all aspects of spatial modeling and analysis in diverse fields ranging from mining to environmental management, and urban planning to defence (Anon., 2003b).

Both Gemcom and Whittle Four-X Analyser™ software are produced by Gemcom Software International Inc. GEMS is used for mine planning and optimization, designing ultimate pits and pushbacks, mine production scheduling and cut-off grade strategies. As well, GEMS is used in underground mine planning and design and is particularly suitable for underground blast design and block caving management systems. Like GEMS, Whittle Four-X Analyser™ software for open pit optimization uses the Lersch-Grossman graph theory algorithm. It incorporates a wide range of functions for designing ultimate pits and pushbacks, mine production scheduling and optimizing stockpiles and cut-off grade strategies. The Whittle Four-X has several modules which include the multi-element, mining width, the Milawa algorithm®, Express NPV output, pushback 50, buffer stockpiles, pushback chooser, stockpile and cut-offs, blending, discounted pit shells, multi-and-advanced analyses modules (Anon., 2003c).

Datamine Studio is generally used by mining engineers and exploration geologists in geological exploration and modeling of ore blocks, mine planning, design and optimization of open pits, underground mines and quarries, production scheduling and blending (Anon., 2003d). Datamine also has enhanced geostatistical modeling modules for calculating ore reserve values in blocks, grades and their variations in different directions from geological exploration data. It can be used to optimize the mineable reserves in ore blocks in both underground and open pit mines. The important modules in Datamine Studio include drill hole data handling and display, block modeling, enhanced geostatistics, wireframe modeling, 3D visualization, open pit mine design, underground mine design, schedule and achieve, stereonet viewer, short term planning and floating stope optimizer (Anon., 2003d). Using the theories of regionalized variables, advanced geometry, graph, solid and computer modeling, these software packages are used to estimate the average grades, volumes and tonnages of ore or waste materials within mining blocks and the global volume or tonnage in a particular deposit. However, the use of Datamine, Gemcom and Surpac are limited to orebodies where there is a detailed and comprehensive database of drill hole records from exploration drilling showing details of the rock types and faulting intersected by each drill hole. In addition, the high cost of these software limits their use to only large scale operating mines.

2.4.1 Simulation Software

The computer-aided design (CAD) aspects of these software enable mining engineers to develop geological and block models of orebodies, design various layouts of mining excavations and output the results in diverse graphical formats. Automatic Dynamic Analysis of Mechanical Systems (ADAMS), Algor, Visual SLAM with AweSim, Simphony and SIMAN are software that are often used for modeling discrete and continuous units and simulating bulk materials flow on benches, through pipelines or on belt conveyors. These software are based on Fortran 77, Visual Basic or C languages. Visual SLAM is an advanced simulation language that allows the modeler to select the "world view" that is most applicable to the system under study. Visual SLAM supports three world views of network, discrete event and continuous modeling. User inserts and event routines can be developed in either Visual Basic or C language. To support the entire process resolution in Visual SLAM, AweSim provides a database, project maintainer, interactive execution environment (IEE), standard textual and graphical reports, and concurrent, and post-process animation facilities (Pritsker et al., 1997). Simphony is a special purpose simulation language for building and experimenting with construction simulation models, based on the existing modeling element library. It is a tool that can be used by engineers to automate, formalize and verify decision-making processes in construction management (Anon., 2000a). Simphony's biggest strength is its ability to allow for the construction of re-usable simulation models. This is accomplished through two approaches: parameterized models and structural reuse. With parameter-based models, certain defined parameters can be changed by engineers at run-time to simulate slightly different scenarios and generate different outcomes. Structural reuse takes advantage of object-oriented simulation to accomplish more advanced methods of reuse. A simulation model based on this method consists of a set of elements, possibly with their own parameters, connected together. The simulation outcome is a function of the parameter values of each element and the manner in which these two elements are defined and connected together. These supported concepts allow for the generation of extremely flexible and reusable models that can be used to model a wide range of scenarios within the target domain. Simphony is a software based mainly on Visual Basic (Anon., 2000a). ADAMS software is a mechanical dynamic virtual prototyping software. It can be used to evaluate complete system motion and force by employing Newton-Euler equations (Synge and Griffith, 1959; Fu et al., 1987; Anon., 2003g).

A number of available packages for belt conveyor transport systems are BELTSIM (Bucklen et al., 1969), Continuous Materials Handling Simulator (CMHS) (Tan and Ramani, 1988), Coal Mine Belt Capacity Simulator (CMBCS) (Thompson and Adler, 1988), BETHBELT T-1 (Newhart, 1977) and Underground Materials Handling Simulator (UGMHS) (Manula et al., 1974). These software are mainly used for simulating production operations which employ continuous miner-belt conveyor systems underground. Most of these software packages have been developed for in-house use within certain mining operations only and are not commercially available to other users.

Discrete event and continuous simulation techniques have been widely applied in open pit production systems for detailed analysis of equipment interaction. This is because these approaches can easily accommodate the strong dynamic and stochastic nature of the systems. As well, great levels of detail can readily be incorporated in the simulation models (Yingling, 1992). Unfortunately, most of the available software for modeling such discrete and continuous events can hardly be employed to give accurate and real time results of bulk material flow on benches when discrete and continuous mining units are interfaced as envisaged in the conceptual AFS methods. In addition, they cannot be used to model the differential lateral and longitudinal changes in a 3D open pit configuration as the loader excavates the materials.

2.5 Material Handling Systems

The development of material handling systems in surface mining operations has been largely based on the theories of cyclic (discrete) and continuous flow processes. Discrete material loading and transport systems are predominantly employed in most open pit mines because they allow high flexibility in planning and scheduling the operations, and easily cope with the frequent changes in the pit face configuration (Martin et al., 1982). They comprise the combination of discrete loading equipment such as shovels (cable and hydraulic), backhoes and front-end loaders with trackless trucks, shuttle cars, trains, aerial tramways. Other mining equipment which combine the loading and haulage operations are draglines, large stripping shovels, load-haul-dump machines and tractor scrapers. The production capacities of such discrete loading and haulage units are characterized by a cycle time. An accurate estimation of the cycle time of a discrete unit equipment leads to the determination of the production capacity of the equipment (Crawford, 1979; Sweigard, 1992).

In shovel-truck systems, a given load of material is delivered to the processing plant, stockpile or waste dump by the trucks. Material flow is not continuous but depends on the total production rate of the fleet of trucks in the system. The production capacity of a fleet of haulage trucks depends mainly on their rated capacities, cycle times and operating conditions within the pit (Hendricks and Dahlstrand, 1979). In this system, the link between the shovel at the face and the treatment plant (crusher), waste dump or stockpile is the fleet of trucks that travel from one destination to the other. Due to the operating flexibility, mobility from one operation to another and resale value, truck haulage is the favored method for moving both ore and waste in open pit mines (Frizzell and Martin, 1992). The cycle times of trucks and cost of haulage depend on the length and nature of the haul roads to the crusher, waste dump or stockpile. As the pits deepen and haul roads lengthen, more truck units have to be added to meet the production targets. Thus, conventional truck haulage costs usually increase dramatically together with the cycle times. This often leads to lower production rates per hour and lower system efficiencies. Due to the fact that trucks are the most costly portion of the shovel-truck system, some mining companies are simply eliminating them by employing either mobile or semi-mobile in pit crusher/conveyor belt systems, cross-pit conveyors or side casting operations (Sullivan, 1990).

2.5.1 Cyclic Material Handling Systems

Cyclic materials handling systems involve all loading and haulage equipment that deal with discrete units and are generally described by a cycle time. Knowledge of the cycle time of the equipment enables the calculation of its production capacity. Depending on the targeted production required by the mine per day or per hour, the fleet size (number and type of equipment units) required to meet the production target can be determined. The production rate of discrete unit loaders that require no tramming (e.g. backhoes, cable and hydraulic shovels) which are used to load haulage vehicles such as trucks is a function of the heaped capacity of dipper or bucket (tonnes), number of cycles or trips per hour, fill factor and job efficiency factor (utilization × availability). The cycle time of the unit comprises the most common components of a discrete unit loading cycle i.e. time to load bucket, bucket swing time (loaded), time to dump load and bucket swing time when empty (Crawford, 1979; Yingling, 1992).

The production rate of discrete unit haulers is the product of the capacity of the truck (tonnes), number of trips it makes per given time, fill factor and job efficiency factor. The cycle time of the trucks comprises the loading time, haul time of truck when loaded to crusher, stockpile or waste dump, dumping time at crusher, return time of truck (empty) to shovel, spotting time and other variable delays in the cycle (Crawford, 1979). Simulation of discrete materials haulage units provides a basis for the detailed analysis of the units in combination with the loading equipment. Issues addressed in simulation of discrete haulage units in combination with discrete excavation equipment include (1) the nature and profile of the haul roads; (2) type, size and number of haulage units in a fleet; (3) type of real-time fleet control strategies (type of dispatching and management of fleet); (4) detailed analysis of the equipment interactions on overall system performance

(e.g. amount of maneuvering required before trucks can be spotted for loading at the mine face); and (5) analysis of the section and pit layout options (Yingling, 1992).
The challenges of the system center on optimizing production economics and efficiency by improving upon haul roads quality factors such as grades, radii of curves, widths, surface conditions and the number of crossings. Other challenges include the optimum loading of haulage trucks to ensure useful economic tire life and long-term truck integrity and the attainment of economies of scale with bulk production (Martin et al., 1982).

2.5.2 Continuous Materials Handling Systems

The common continuous materials handling systems involve bucket wheel excavators (BWEs), bucket chain excavators (BCEs) or bucket ladder, suction or jet lift dredges as the primary excavation units. These units are matched with bulk materials transport systems like belt conveyors, screw conveyors, chain conveyors or with hydraulic or pneumatic transport systems. Problems that are addressed using simulation models include the design of the system throughput capacity, assessment of conveyor network configuration, impacts on the availability of the transportation system to production system, and testing control strategies for belt monitoring and control systems (Yingling, 1992). Ships, barges and/or bucket ladder dredges may also be used to handle bulk materials in placer or ocean mining environments. The production capacity of each hauling unit is a function of material characteristics (lump sizes, density, wetness, stickiness, abrasiveness, corrosiveness and temperature), environmental effects, operating costs, distance, topography and availability of water. Belt conveyor and hydrotransport systems are superior to the other bulk material handling systems in the transportation of materials under a wide variety of conditions (Anon., 1979).

2.5.3 Production Capacity of Continuous Flow Loaders

This category of equipment includes BWEs, BCEs and bucket dredges (Sweigard, 1992). BWEs are limited to materials with a cutting resistance below 70 kg/cm (4,700 lb/ft) and compressive strengths below 12 MPa, and are adaptable to all hauling and materials handling systems (Sagner, 1990). BWEs with theoretical outputs of up to 20,000 m^3/h have been reported to be operating in surface mines (Durst and Vogt, 1988). However, they require strong soil bearing characteristics over which the excavators, mobile crushers and slurrification units can travel. In addition, adverse digging conditions, shifting of bench and face conveyors, preparation of ramp for BWE can seriously hamper the production of a BWE (Morey, 1989). The theoretical output of a BWE is based on the bucket size, the number of bucket discharges per minute and slewing speed of the bucket wheel (Durst and Vogt, 1988). The amount of material cut by the wheel at any given time depends on the height and the depth of cut of the bucket wheel.

2.5.4 Conveyor Belt Transport Systems

The concept of belt conveying goes back almost two centuries. The first reference to the commercial use of flat belt conveyor in the USA was for grain handling in 1795. Development of belt conveying took place at a slow pace during the nineteenth century with most of the belt conveyors constructed during the period being used for grain handling. It was not until the 1890's that the use of belt conveyors for materials heavier than grain commenced (Anon., 1966; Brooks, 1971). Belt conveyors are predominant in transporting dry bulk solid materials because of their operating economics and safety, reliability, versatility and practically unlimited range of capacities (Anon., 1988). The characteristics include continuous material flow, high material capacity, fairly rapid movement, economies of scale, and flexible application (small portable units to long hauls). Commercially available conveyors include the conventional belt conveyors, screw conveyors and chain conveyors. Conventional troughed belt conveyors have been widely used in industry mainly for the movement of materials within processing plants and also over long distance overland transportation. Modern belt conveyor systems include single flight lengths of 10 to 15 km, increased belt speeds (5 – 9 m/s) and increased tonnages (2,000 to 6,000 t/h). Cable belt systems can have up to 30 km of singe flight lengths. As indicated by Marlatt (1977), one of the longest belt conveyors in the early nineteenth century was installed for handling phosphate in the Spanish Sahara. It was about 100 km long and made up of ten sections with a maximum individual length of 20 km.

Research has shown that belt conveyor widths ranging from 800 to 2000 mm are most suitable for most situations. Other types of conveyors include high lift sandwich type conveyors, pipe conveyors, enclosed suspended type belt conveyors, flexowell conveyors and air supported belt conveyors (Morey, 1989). High angle sandwich conveyors constrain the bulk solids between two belts during transport. High angles of conveying up to 90° are possible making the sandwich conveyor particularly useful where bulk materials are required to be elevated in narrow operating areas. High angle conveyors provide an economic alternative to large dump trucks in open pit mines with highwalls. In process plants, other types of conveyors for handling materials widely used include bucket elevators, screw conveyors, vibratory conveyors and drag chain conveyors (Morey, 1989; Kutschera, 1994).

The types of conveyor drives include conventional belt drives with the driving and return drums at the end of the conveyor, intermediate booster conveyor drives, linear booster drives and multi-type booster drives (Roberts and Hayes, 1980). Belt conveyors have low maintenance and operation costs. They do not require highly skilled workers, are able to cross over adverse terrain, have practically unlimited haul range, high reliability and excellent safety record (Anon., 1988). The factors that affect the carrying capacity of a belt conveyor include the cross-sectional area of material, belt speed, bulk density of the material and the angle of inclination of the belt to the horizontal (Roberts and Hayes, 1980). When the angle of inclination of the conveyor belt is low, the effect of slope on the capacity of the conveyor is minimal and is thus omitted in the calculations (Brook, 1971; Roberts and Hayes, 1980; Sweigard, 1992; Roberts, 1994).

The main challenges with belt conveyor systems include the choice of the right type of conveyor belt widths and speed, and the drive systems and components. They also include how to design and install the support structure for the belt conveyors to avoid excessive belt tensions, avoid belt slippage, ensure proper alignment and prevent spillage of materials and dirt on the belt return side. Other challenges, particularly with mobile and semi-mobile conveyor belt systems, are focused on the movements of the train of conveyor belt wagons to minimize material flow interruptions as well as ensure the safety of workers and equipment at the face.

2.5.5 Hydraulic Transport Systems

According to Bain and Bonnington (1970), the earliest hydraulic transport from 1913 to 1924 conveyed coal up to 12.7 cm in size from a wharf on the River Thames to Hammersmith Generating Station, over a distance of 603.5 m at the rate of 50 tons/hr. Hydraulic transport is predominantly employed in small installations for conveying solids between processes in the chemical industry, disposing of solid waste from mineral beneficiation plants, and in the distribution of conventional oil and gas. However, a number of large scale hydraulic transport installations have been successfully operated in dredging, land reclamation, disposal of rock and overburden in phosphate mining in Florida, USA and for oil sands slurry transportation at Syncrude and Suncor mines in Ft. McMurray, Canada (Bain and Bonnington, 1970; Kizior and Payne, 1991). Hydraulic transportation has also been employed to transport coal and copper concentrates in some mines. Presently, 25 million tons per year of coal is transported as slurry coal in pipelines up to 1667 km. (Snoek et al., 1976; Wasp et al., 1979; Jacques et al., 1982). Hydrotransport of limestone, iron and copper concentrates, and oil sands have been successfully achieved over distances up to 407 km (Wasp et al., 1979; Alexander and Shaw, 1984). In addition, solid waste materials have been transported by pipelines to waste disposal sites in some municipalities (Kundu and Peterson, 1986).

The size of particles conveyed in such hydraulic systems range from very fine particles (< 1 μm) to as high as 25 cm. Centrifugal pumps are mainly used to transport larger particles in heterogeneous slurry over fairly short distances (Kundu and Peterson, 1986). Apart from water, other types of fluids have been used as carrier fluids in hydrotransport systems. These include oil agglomeration method used in Australia (Rigby and Thomas, 1983), liquid carbon dioxide (Santhanam et al., 1985) and methanol (Keller, 1979). The flow of a solid-liquid mixture through a pipeline is a complex phenomenon with the flow characteristics and subsequent pipe friction being dependent upon size distribution, shape, density, concentration of solids and pipe diameter. Slurries are generally classified as settling (heterogeneous) and nonsettling (homogenous) depending on whether the settling velocity of a 62 μm quartz sand grain is less than or greater than 1.5 mm/s in water at a temperature of 20 °C. If the settling velocity is greater than 1.5 mm/s, the slurry is said to be a settling slurry while a settling velocity below 1.5 mm/s denotes a nonsettling slurry (Addie, 1982). The flow characteristics of settling and nonsettling slurries are quite different.

The most efficient slurry transport is achieved when the specific energy consumption, E_s, is a minimum. The specific energy consumption, which is dimensionless, is the energy required to move one kilogram of solids through a horizontal distance of one meter. E_s is affected by the friction pressure gradient in meters of water per meter of pipe length, specific gravity of the solids and the delivered volume concentration of solids (fraction). Nonsettling slurries flowing in a pipe have a uniform distribution of particles across the flow section and axisymmetric velocity distribution. The flow of nonsettling fluids is often treated as that of a pseudofluid where the density of the mixture is equal to that of the carrying fluid. Homogenous slurries like drilling mud, sewage sludge and concentrated suspensions of fine limestone often exhibit a non-Newtonian rheology at normal pipeline velocities. Heterogeneous mixtures have lower solids concentration and larger particle sizes than homogenous slurries (Wasp et al., 1979). Transport of two-phase mixtures is known to take place in only two flow regimes – as a heterogeneous suspension and by saltation. Both flow regimes could occur simultaneously in a pipeline (Bain and Bonnington, 1970; Gillies et al., 1991; Gillies and Shook, 2000).

Settling slurries are generally heterogeneous mixtures in which a portion of the solid particles is carried as suspended load while the remainder is moved along the bed of the pipe. The bed-load or stratification ratio, R, which is the ratio of the bed-load transport to the total transport, is used to characterize the flow conditions in settling slurries. Since turbulence is a function of mean velocity, V_m, R is also a function of V_m (Addie, 1982). At a sufficiently high mixture velocity, all the solid particles are conveyed as suspended load or as a pseudo-homogeneous suspension for which R = 0. At lower velocities, the solid particles tend to settle towards the bottom of the pipe resulting in bed-load transport in which the particles bounce, roll and slide along lower portion of the pipe. This results in large resistance to the solid/solid bed-load transport together with a little additional resistance from the suspended-load transport. Under those conditions, the flow characteristics of settling slurries differ significantly from Newtonian fluids (e.g. water). Thus with settling slurries, the conveying velocity is required to be far above the velocity at which the solid particles begin to form a bed at the bottom of the pipe to maintain operating stability of the pump (Gillies et al., 2000).

Rabinowitsch and Mooney (1982) show that the rheologic properties of a fluid can be determined by means of experimental measurements of steady and uniform, laminar flows in a circular conduit. For all laminar flows in a pipe, they have shown that rheologic properties of a fluid is affected by the inside diameter of pipe, the mean velocity of fluid and the slurry consistency property. For turbulent flows, the Darcy-Weisbach and Colebrook equations are used to measure the friction pressure gradient. The Colebrook equation for boundary-drag coefficient for a single-phase fluid is assumed to apply to nonsettling slurries. For many nonsettling slurries in which turbulent flow conditions exist, the viscosity of the slurry mixture, μ_m, is considered the same as that of the carrying fluid, μ_f, and the densities of the mixture and carrying fluid are assumed to be the same. For other nonsettling fluids, the viscosities of the mixture and the carrying fluid are considered different (Addie, 1982; Gillies et al., 1991; Sanders et al., 2000).

The basic problem with hydraulic transport is the determination of the fluid forces acting upon the solids and the effect of these forces upon the behaviour of the solids and upon the resultant energy losses. Bain and Bonnington (1970) have provided a model for estimating the behaviour of solids and the energy losses expected in pipelines conveying solid-liquid mixtures and how the quality of both the water and the material being transported are affected. Some other problems with hydraulic transport include the impact on the environment when the solids and liquids have to be separated at the other end of the pipeline, treated or disposed off and/or when there is a spillage due to pipe bursts (Jacques et al., 1982). The success of hydraulic transport is, however, reckoned in terms of its rate of solids throughput, simplicity, reliability and low manpower requirements and costs (Bain and Bonnington, 1970; Gillies et al., 1991).

In horizontal pipelines, a characteristic of the flow of slurries is the fact that the solid and liquid phases remain identifiable as there is no increase in viscosity of the liquid phase on account of its association with the solid particles. In cases where some of the solids are ultra-fine and are held in homogenous suspension by the water or carrying medium, the medium thus formed must be evaluated in terms of its density and viscosity. The suspended fine particles are regarded as constituting part of the new carrying medium and are not considered as part of the solids burden to be conveyed by this medium. The new medium and the larger solid particles, which are not components of the medium still behave respectively as liquid and solid in two distinct phases (Bain and Bonnington, 1970). The increased buoyancy of the solid phase due to the increased density of the liquid phase helps to reduce the pressure losses within the pipeline. The carrying capacity of a hydraulic transport system is governed by the cross-sectional area and density of solids and velocity of fluid (Sweigard, 1992). Sweigard (1992) has noted that the velocity required to cause solids to float in an upward stream of fluid is determined by the internal diameter of pipe, density of the fluid and the density of the solids.

2.5.5.1 Multiphase Slurry Transportation

Hydraulic transportation of oil sands and coal over long haulage distances and rugged terrain has economic advantages over other types of ore haulage systems. Hydrotransport of oil sands slurry is a multiphase flow system in which the water, sand and bitumen phases co-exist with their superimposed behaviors affecting the entire system rheology (Frimpong et al., 2002a). Some theoretical approaches have been developed which consider multiphase mixtures as pseudo single-phase fluids with variable densities (Shook and Daniel, 1969; Shook, 1986). However, due to the boundary conditions adopted in such approaches for simplicity, it is difficult to apply the models to actual flow situations. Thomas (1963) used the pseudo single-phase approach to model a two-phase flow system. Pseudo single-phase techniques enable the conservation of energy equations to be developed for single-phase fluid flows. These are used with appropriate modifications to cater for the rheological properties of the overall mixture especially in the computation of the various parameters. Depending on the application, the most important parameter computed varies from pressure drop in the slurry transportation pipeline to the deposition and the critical velocity in solid-liquid media.

Multiphase slurry systems are assumed to be homogeneous in most theoretical and experimental works. In most heterogeneous and homogenous slurry systems, the solid phase is usually categorized by size and properties (Durand, 1953; Kaskas, 1971; Wasp et al., 1977). Multiphase slurry transportation systems have been investigated beyond the pseudo single-phase stage treating the phases as distinct and interacting. Hinze (1963) employed this approach for dispersed multiphase flows. The stress tensor was treated as a composite velocity made up of the dispersed solids and the slurry vehicle phase velocities. In this way he accounted for the velocity slippage between the two phases and neglected the effect of particle concentration on the shear viscosity of the stress tensor. Later, Murray (1965) developed a model for the entire slurry system that contained a virtual term that does not actually follow Newton's third law.

In addition, multiphase flows have been studied by combining the second law of thermodynamics with three Lagrangian multipliers to account for the entropy inequality common in two-phase flows. Thus, the established restrictions on the form of the constitutive equations (particularly those at the fluid-particle interface) for the fluid-particle mixtures have been incorporated (Stewart and Wendroff, 1984). The end result of this approach was the introduction of a distinct hydrostatic pressure for each phase together with an interfacial pressure. Properetti and Jones (1994) developed several mathematical models for solving various two-phase flow problems on the basis of one-dimensional stratified flow of two inviscid fields (or fluids), with distinct pressures ascribed for the two fields.

Nunziato (1983) proposed a different approach to formulate the governing equations for disperse two-phase flows. In this case of high particle concentrations, collisions among the particles also give rise to growth in the rapidly varying component of the pressure field. Unlike the models by Murray (1965), Stuhmiller (1977), Nunziato (1983), Stewart and Wendroff (1984), Properetti and Jones (1994), the rapidly varying pressure component appeared only in the equation of motion for the continuous fluid phase. An attempt was made to characterize the rheological behavior of dilute suspensions, due to non-uniform particle distributions, by extending Nunziato's model to plane Poiseuille flow (McTigue et al., 1991). Poiseuille flow of a suspension through a vertical, two-dimensional channel was modeled using two different models. These models include (i) models with distinct pressures for each of the solid and fluid phases and, (ii) a model with one pressure (Hwang and Shen, 1991). As such, the inter-phase interaction was modeled differently. Approximate analytical solutions were obtained for both models (Frimpong et al., 2002a).

2.5.5.2 Generalized Assumptions in Multiphase Flow Systems

The basic assumptions in multiphase flow system modeling are that (i) the various phases in multiphase flow systems (fluid and solid phases) are immiscible and incompressible, and the presence of trapped air is neglected as illustrated in Frimpong et al., (2002b); (ii) the various phases are uniformly distributed throughout the pipeline; (iii) the solid particles in multiphase systems are dispersed and do not settle at the bottom along the pipe length to form a stationary bed (Addie, 1982); (iv) the density of each phase is constant and different from those of the other phases; (v) the velocity and mass/volume flux for each phase are constant over the entire flow region; (vi) the shear stress along the pipe wall is constant irrespective of the phase that is in contact with the pipe surface (wall); (vii) in oil sands slurries, the hot water and bitumen are considered as the continuous immiscible phases; (viii) the temperature of the bitumen is assumed to be constant throughout the pipe length, and the average sizes of sand particles are less than 5.08 cm. Further assumptions made on the mixture properties of oil sand slurries as: (i) density of sand (solid) particles = 2650 kg/m^3, bitumen (API <10° and density = 995 kg/m^3) and density of hot water = 1000 kg/m^3 (Ghazi, 2002; McDonell, 2002); (ii) the flow of the slurry mixture is continuous from one end of the pipeline to the other; (iii) the flow is considered as laminar or turbulent depending on the rheological conditions, pipe diameter and velocity of the slurry (Frimpong et al., 2002b).

Fig. 2.2 is an elemental portion of a pipeline showing the possible 3D slurry flow patterns. As the slurry enters the pipeline, the flow stream is governed by velocity spectra that consist of axial, radial and tangential components. The velocity spectra are a function of the pipe wall conditions, the components of the slurry, the pipe configuration and layout. The velocity spectra in pipe flow are important in determining the overall flow performance (Frimpong et al., 2002a). The friction losses, total head, velocity spectra and volumetric flow rates must be determined to capture the productivity of the hydrotransport system. Considering the velocity components in Fig. 2.2, the axial flow component is usually the predominant one as the radial and tangential components are often very small compared to the axial flow component.

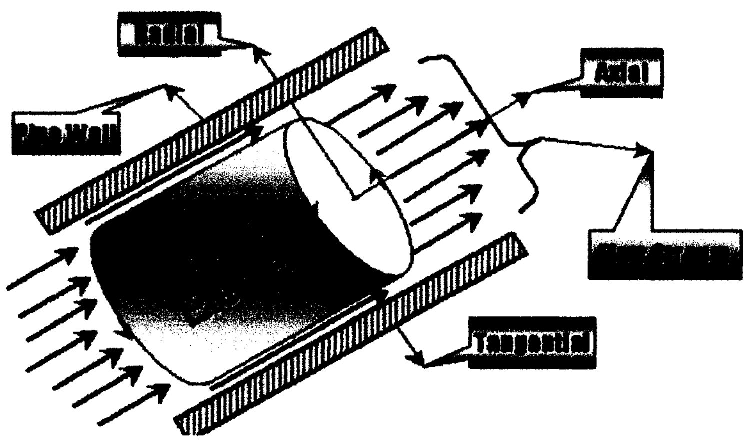


Fig. 2.2 Model of 3D Elemental Flow through a Pipe showing Velocity Components

The flow between two ends of the pipeline can be modeled with the continuity and momentum equations, which give the conservation of mass between the two ends. This can be extended to predict flow pattern at every point along the pipes. The two ends must be connected by a streamline where the conditions are known and the continuity equation for the control volume of solid particles and two continuous phases can be derived (Greenspan and Nigam, 2001). However, detailed information on the local mass flux, velocity and density is seldom available to solve equations on momentum. The data on the profiles depend on the volumetric flow rates of the different phases, their physical properties and pipe geometry. The initial and boundary conditions of the multiphase slurry flow system are then applied to equations to determine the shear stresses, average momentum flux and volumetric flow rates in multiphase flow systems using finite difference approximation methods (Bain and Bonnington, 1970; Frimpong et al., 2002b).

2.5.5.3 Volumetric Flow Rate

The volumetric flow rate, Q_s, in hydraulic transport systems may be found by integrating the axial velocity across the pipe's cross-section. The average flow velocity is obtained by dividing the volumetric flow rate by the pipe's cross-sectional area (Frimpong et al., 2002a). For two-phase flow systems (e.g. solids and water as the carrier fluid), the volume flow rate of the solids is given by Bain and Bonnington (1970). The volume flow rate, Q, is the ratio of the volume flow rate of the mixture, Q_s, to the concentration of the solids in the mixture, C_s (Bain and Bonnington, 1970). The challenges involved with hydraulic transport systems include (1) determination of the fluid forces acting upon the solids and the power required to convey them through the pipelines, (2) ensuring that the quality of the carrier fluid (usually water) and the material being transported are not affected, and (3) when necessary, separating the solids from the fluid and disposing of the unwanted materials in a manner that is environmentally acceptable.

2.5.4 Combined Materials Handling Systems

Discrete unit mining equipment that combine the excavation and handling of materials include draglines, large waste stripping shovels, tractor scrapers, load-haul-dump (LHD) machines and dozers. While the tractor scrapers, LHDs and dozers tram their load over some distance before dumping it, draglines and large stripping shovels have fixed bases and do not combine tramming in the materials handling operations. The production capacity of combined loader-haulers like LHDs and tractor scrapers is the product of the carrying capacity of bucket or dipper per cycle or trip, number of trips per hour, fill factor and the job efficiency factor. With regard to discrete combined loader-hauling equipment with fixed bases like draglines and large stripping shovels, the production capacities are generally calculated on the basis of annual overburden stripped or ore mined (Anon., 1976). The production capacities are governed by bucket or dipper capacity, cycle time, scheduled yearly operating time, bucket fill factor, swell factor of material, and the availability and utilization of equipment unit. The production capacities of such a unit per day, per shift or per hour are then calculated from the scheduled number of operating days per annum and the number of shifts per day and hours per shift.

2.6 Summary

Open pit production planning, design and scheduling involve the geometrical mine design of the pit layouts, sequence of extraction of ore and waste blocks, lateral and vertical expansions in pit faces and scheduling of mine equipment. The ultimate objective of mine production planning and scheduling is to maximize the net present value and the return on investment that can be derived from the mining project within a given period. To achieve these goals, it is necessary to design and schedule the extraction of the ore and waste blocks using discrete and continuous mining equipment such that the stability of the pit slopes is maintained to ensure the safety of personnel and equipment within the pit. There are several computer-based software such as Gemcom, Datamine, Vulcan, Whittle Four-X Analyzer and Surpac Vision that are used to generate 2D and 3D models of open pits in mine planning, design and optimization of mines. Simulation software such as Visual SLAM with AweSim, Simphony and SIMAN are used for modeling discrete and continuous units and events, and for the simulation of bulk material flow on benches by trucks, through pipelines or on belt conveyors. ADAMS and Algor software are used for calculating the torsional and bending forces, displacement, projection angles of the equipment components and parts. Discrete event and continuous simulation techniques have been widely applied in open pit production systems for the detailed analysis of equipment interaction.

Materials handling in surface mining operations is mainly by discrete or continuous flow units and processes. Discrete materials loading and transport systems (e.g. the shovel-truck system) are predominantly used in open pit mines because they allow high flexibility in planning and scheduling of operations and easily cope with frequent changes in the pit configuration. As most open pits become wider and deeper, haul roads lengthen leading to dramatic increases in conventional haulage costs and cycle times, lower production rates and system efficiencies. Thus, most mine operators are searching for cheaper mining and haulage systems that will ensure the optimum profitabilities of their operations. Bulk materials transport systems such as belt conveyors and hydraulic transport systems have been noted to offer lower operating costs. They are also versatile and have practically unlimited range of capacities and are increasingly being employed in the bulk transportation of materials in large surface mines (e.g. oil sands mines). They offer competitive advantages over other materials handling systems (e.g. truck haulage) in reducing the unit production costs of oil sands and other materials.

The AFS technology is intended to take advantage of the lower unit operating costs, higher payload-deadweight ratio, and higher efficiencies of hydraulic and belt conveyor transport systems. The AFS technology extends the hydraulic or conveyor belt transport systems to the production faces in the pits using mobile and semi-mobile trains of flexible/rigid pipelines or belt conveyor wagons. The AFS technology will introduce a unique set of mine design layout, configuration and ergonomic challenges by allowing the mobile train of flexible pipeline or conveyor belt systems to adapt to the production face dynamics resulting from the shovel movements during excavation. The main challenges include the design of a system that will ensure that production targets are achieved with maximum safety, minimum risks and uncertainties. The interactions between the shovel and mobile AFS train during the advance and retreat motions of the shovel have to be well-coordinated to ensure adequate matching of their production capacities and also prevent accidental breakage or damage to the various components. The layouts of the AFS options must ensure safe interaction of personnel, production and ancillary equipment in the production operations. The potential sources for ergonomic problems with the AFS technology may include pipeline constraints to mobile equipment, mobile train and shovel interaction problems, possible breakages associated with flexible pipelines and potential flooding of pit layout and weather conditions.

CHAPTER 3

CONCEPTUAL MODELS OF AT FACE SLURRYING OPTIONS

3.0 Introduction

The geometrical mine design and bench material flow simulation systems are based on a number of different conceptual models of the "At Face Slurrying" equipment. These options involve a combination of shovels, bucket wheel excavators (BWEs), dredges, hoppers, crushers, slurrification units, pumps, flexible and rigid pipelines, and conveyor belt wagons. These components are required to ensure a continuous flow of materials from the bench faces to the processing plant. The movement of the shovel or BWE can occur on both the horizontal (bench movement) and inclined planes (bench-bench interactions). This will require that there is enough room in a circle of influence (production envelope) that may be deemed the safe operating radius for an AFS production system. This safe operating radius may be determined based on the size and components of the production equipment. The mining models examined in this chapter are the current mining system (CMS), and seven other conceptual AFS options. These are (1) cyclic excavator conveyor belt control system (CycEx CBCS), (2) cyclic excavator flexible pipeline control system (CycEx FPCS) (3) cyclic excavator mobile slurry pipe wagon articulating system (CycEx PWA), (4) BWE conveyor belt control system (BWE CBCS), (5) dredging with flexible pipeline control system (DRE FPCS), (6) hydraulic mining (HYDS) and (7) in situ recovery techniques (INS REC).

3.1.1 Current Mining System (CMS)

The shovel-truck system is widely used in most surface mines throughout the world. Due to the operating flexibility, mobility and resale value, truck haulage is the favored method for moving both ore and waste in open pit mines (Frizzell and Martin, 1992). The shovel-truck system is currently being used in oil sands mining and waste stripping operations by both Syncrude Canada Ltd. and Suncor Energy, Inc.

The CMS comprises shovels as the primary loaders with diesel powered dump trucks that are dispatched or allocated to each excavator. The loaded trucks transport the material over short hauls to dump sites at the crusher-slurrification facility located close to the face. The system comprises discrete loading and haulage units whose outputs per hour are characterized by their cycle times. Fig. 3.1 is a schematic diagram of a typical shovel-truck-hydrotransport system of the CMS.

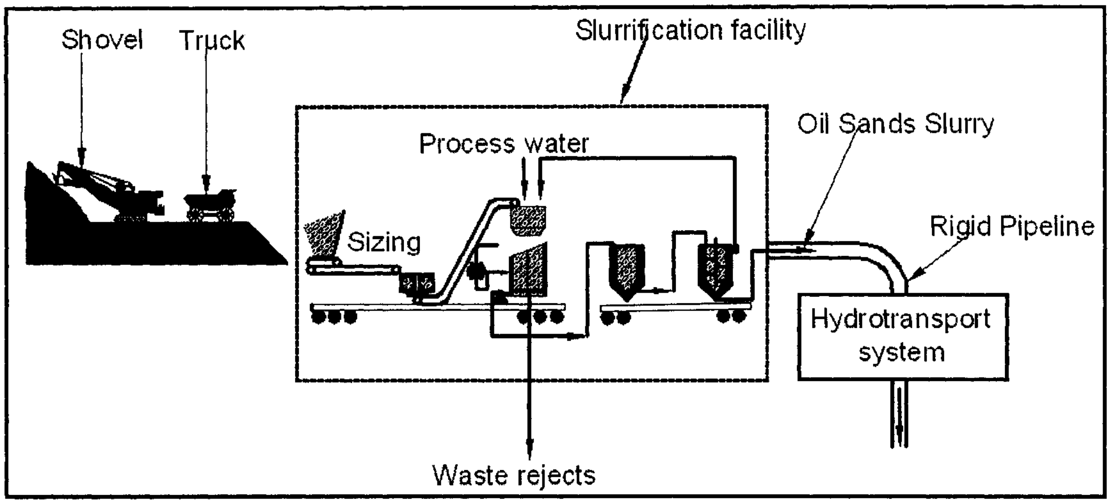


Fig. 3.1 Schematic Diagram of Current Mining System

The advantages of the CMS include reduced haulage distance from the pit face to the slurrying unit and reduced number of trucks (fleet sizes) that will be required to meet production targets leading to lower capital and maintenance costs. This in turn will lead to lower levels of emission of carbon dioxide and other gaseous fumes and particulate matter from the few diesel-powered trucks. In addition, the inherent flexibility of the shovel-truck system will be maintained; fewer operators leading to lower labor costs due to the reduction in fleet sizes; and the creation of better environmental conditions within the pit for workers.

The associated disadvantages include the use of diesel or electrically powered shovels and trucks for excavating and hauling materials to the slurrying unit. The problems associated with the use of large diesel-powered dump trucks in oil sands mining will continue to prevail. The problems include poor traction and high rolling resistances, especially in the summer months, and high emission levels of carbon dioxide (CO_2), oxides of nitrogen (NO_x), sulphur dioxide (SO_2) and other gaseous and particulate pollutants. As well, wide haul roads will have to be constructed and maintained to enable the system work efficiently and ensure maximum safety of men and equipment. The anticipated frequent relocations of the slurrification facility to cut down on the haulage distances of trucks will lead to frequent interruptions in production and high shovel idle times.

The main challenge with this option will be the need for frequent relocation of the crusher-slurrification facility on the bench so as to maintain short haul distances of the truck units and ensure high productivity for the discrete loading and haulage units. Another challenge is the design and maintenance of good haulage roads for the large (> 200 tonnes) capacity haulage trucks. This will ensure long tire life and high reliability, high operator comfort, good traction within the pit (e.g. in summer) and avoid undue stresses on the bodies and undercarriages of the trucks and other mobile units.

3.1.2 Option 1: Cyclic Excavator Conveyor Belt Control System (CycEx CBCS)

Option 1 comprises a shovel, a crawler-mounted mobile crusher, belt conveyor wagons, a mixing tower and pump (slurrification unit). The shovel will be located at the face and load its material into a nearby crawler-mounted hopper. A series of apron feeders will transfer the materials to sizers then into a double roll crusher unit for reduction in size. The crushed material will be sized and conveyed on a series of crawler-mounted belt conveyor wagons to a surge facility from which apron feeders will transport it to a slurrification facility. The slurrification facility will then condition the material into slurry for transport through the main hydrotransport pipeline to the main processing plant (Changirwa et al., 2000; Coward, 2000). The slurrification unit will also receive materials from other faces in a multi-bench multi-face mining and material flow system. This will ensure that a good match is created between the production from the cyclical shovel units and the continuous hydrotransport system. The desire is to meet the required production targets, and avoid downtimes of the slurry unit because of a problem at any mining face. Accordingly, a surge bin will be provided to accommodate all surges in daily production from the shovels and even out the feed to the slurrification facility. The component parts of the equipment for Option 1 are currently available. Custom engineering is required for modification and hybridization of the mobile crusher and belt conveyor wagons. Fig. 3.2 is a schematic diagram of the CycEx CBCS while Figs. 3.3 and 3.4 show the details of the unit components of the system and the belt wagon units respectively.

The potential advantages with this option are (i) reduced haulage costs due to the elimination of trucks; (ii) limited use of rigid/flexible pipelines; (iii) low operating and processing costs due to the reduced number of operators required to run the system and and, (iv) the inherent low haulage costs associated with the use of belt conveyors.

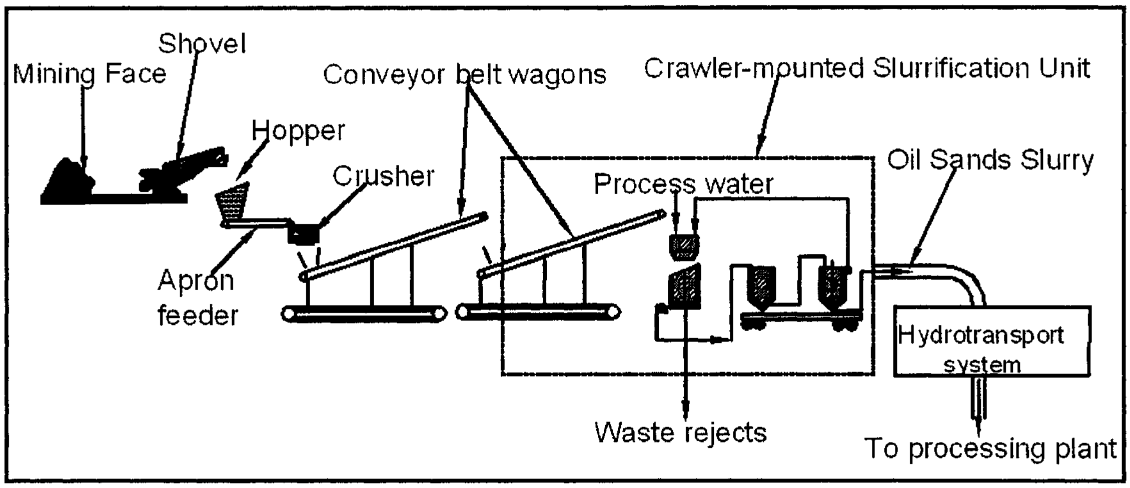


Fig. 3.2 Schematic Diagram of Cyclic Excavator Conveyor Belt Control System

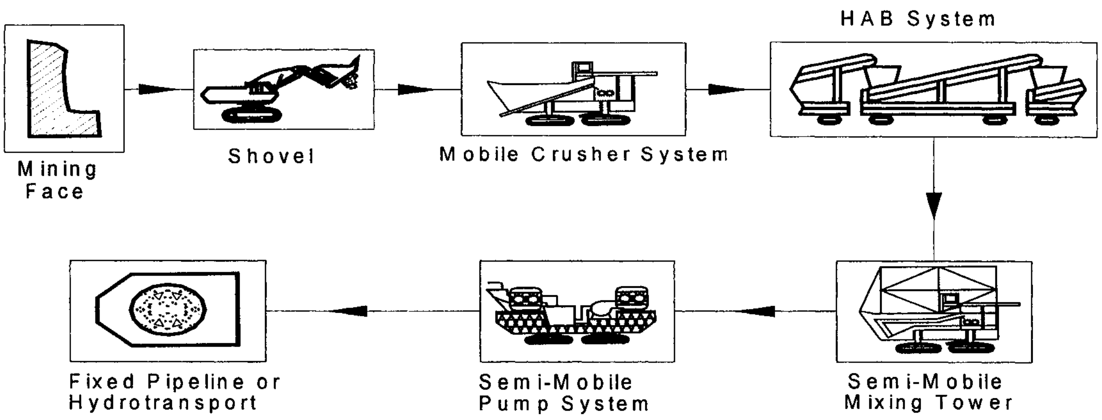


Fig. 3.3 Conceptual Shovel-Mobile Crusher Conveyor Belt Wagon Slurrification System (Source: Changirwa et al., 2000)

In addition, comminution and slurrying of materials at the mine face eliminates the need to transport dry oil sands materials over long distances (Anon., 1988). The possibility for automating and robotizing the components of Option 1 could lead to increased productivity, higher system utilization because of less downtime and enhanced safety of men and equipment. The continuous operation of equipment will lead to lower maintenance costs and also higher system availabilities.

Option 1 will be easy to operate and maintain after a short period of training and with decentralized control systems, tele-operation is possible. The mobile hopper and conveyor belt wagons will constantly follow the shovel ensuring continuous feed to the

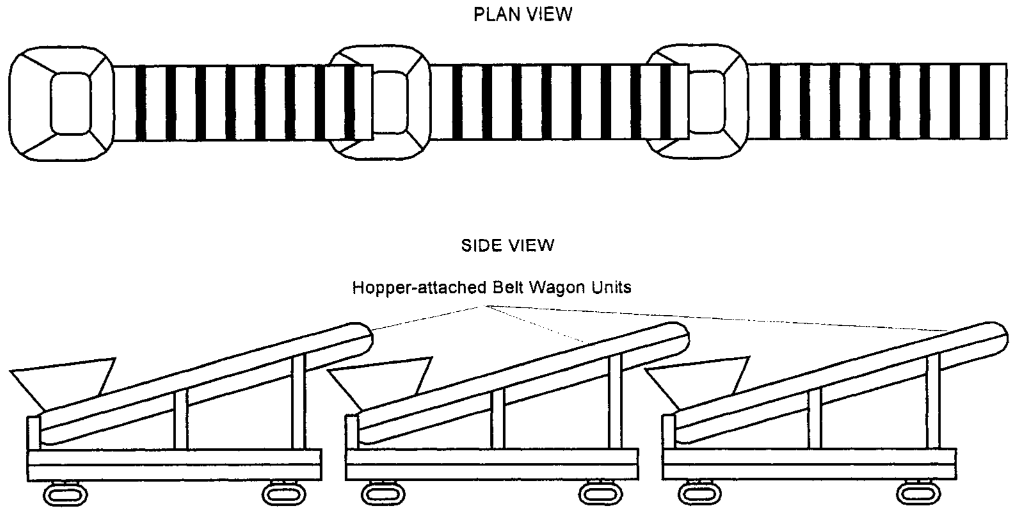


Fig. 3.4 Conceptual Views of the Belt Wagon Units
(Source: Changirwa et al., 2000)

double roll crushers and thus reduce the idle times of the shovel. As well, the excavated oil sands lumps will be crushed and screened to suitable sizes for belt conveyance leading to low wear and tear on the belt conveyors and eliminate costly downtimes. Option 1 will also ensure an even and continuous flow of materials through system as the production from several shovels will be channeled to one slurrification unit. Any breakdown or problems with on any of the shovel units will not adversely affect the output of the slurrification unit and the HTS system. In addition, any spillage of materials from the conveyor belt wagons can be easily cleaned up by bulldozers or front-end loaders in combination with trucks. Finally, adaptation of this new technology will enable Syncrude to remain competitive in the new millennium.

The potential disadvantages of Option 1 are the possible oil sands spillages that will result in wear and tear in the belts that will lead to increased downtimes and lower production with the consequent high costs and reduced efficiencies of the system. Also trucks and front end loaders will be required to clean up and recycle the spilled oil sands materials from the conveyor belts. Longer relocation times due to semi-mobility of slurrification facility will also increase the costs of operation.

The main challenges with this option will be how to match the production capacities of the shovels, conveyor belts and slurrification systems to ensure continuous material flow through the system. It is also necessary to ensure that the movements of mobile hopper-crusher and slurrification units are synchronized with that of the shovel movements on the bench to minimize interruptions to production operations at the face. Extensions or modifications to the train of belt conveyor wagons must not interfere with bench material flow. Also the design must allow for the possible blending of ores to achieve the required mill head grades.

In addition, the wear, torsional stresses and bending moments on the rigid pipelines used will have to be minimized to prevent sudden pipe breaks that could flood the pit with slurried materials. As well, the rheology of the oil sands slurry must be such that there is no sedimentation of the material in the pipes and that the energy required to pump the slurry through the pipelines is minimized by the proper choice and installation of pipelines and pumps in the HTS system.

3.1.3 Option 2: Cyclic Excavator Flexible Pipeline Control System (CycEx FPCS)

Option 2 comprises a shovel, crawler-mounted slurrying system (slurrification facility), and a flexible pipeline or ground articulating pipeline (GAP) system. A shovel loads the oil sands materials into a nearby crawler-mounted hopper. The feed from the hopper will then be transported by short apron feeders to a double roll crusher where it will be crushed and screened to suitable sizes before being fed to a surge facility, then to the slurrification unit by means of apron feeders. The oil sands materials are conditioned and slurried in the slurrification facility to the required concentration by weight. The slurry will be pumped through ground articulating flexible pipelines to a central station from where it will be transported via the main hydrotransport system (HTS) to the base processing plant. The double roll crusher and slurrification-pump box units will all be housed in a crawler-mounted structure that will be linked to the receiving hopper at the face (see Fig. 3.5). Fig. 3.6 shows the details of the CycEx FPCS option.

The ground articulating pipeline consists of a series of pipe arms interconnected by steel helix reinforced rubber hoses and ball joints. One of the GAP joints is anchored to the mobile slurrying system while the other end is connected to the main HTS pipelines. It is necessary to carefully consider the location of the GAP system downstream so that it can be safely used for discharging purposes to the HTS. Upstream location of the GAP (i.e. for suction purposes) could be detrimental to the system as it could cause necking (Changirwa et al., 2000).

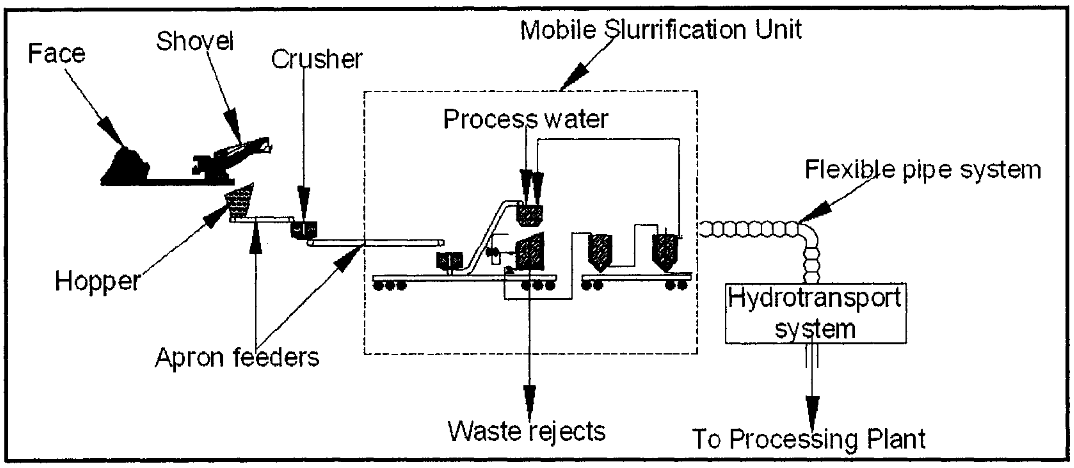


Fig. 3.5 Schematic Diagram of CycEx FPCS

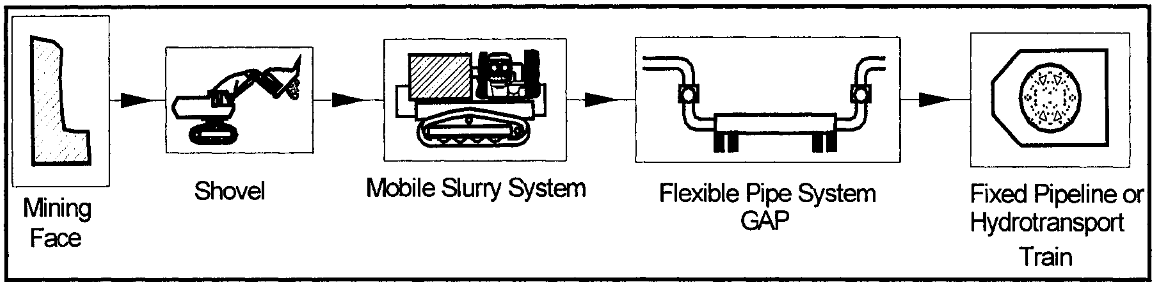


Fig. 3.6 CycEx FPCS Option
(Source: Changirwa et al., 2000)

Option 2 differs from Option 1 because flexible pipelines are used to link the AFS unit with the main hydrotransport facility. Also a separate crusher-slurrification unit will be required for each loader at the face. A breakdown of the shovel unit will lead to increased downtimes of the slurrification facility attached to it since there will be no dry feed to it (except from the attached surge bins). In both Options 1 and 2, the bulk materials are crushed, screened and slurrified before it is pumped through the main hydrotransport system to the base processing plant.

Option 2 also involves a technology that is already known, well proven and currently available, except for the GAP system. Option 2 will adopt a shovel, hybrid tumbler and the GAP system. The GAP system and the hybrid tumbler have a potentially high degree of accuracy in the handling and processing of oil sands. The merits of the CycEx FPCS include reduction in mine production costs due to the elimination of haulage trucks and increased safety due to the possibility of remote operation of system. Other advantages include lower oil sands losses and lower processing costs because up to 15% of the oil sands will be conditioned and processed within the pipelines en-route to the processing plant. The sizing of oil sands lumps (to -80 mm) will lead to low wear in the pipe walls and pumps, saltation and bedding in the GAP system and in the downstream pipes. As well, the system has high mobility during periodic relocations. More importantly, the adaptation to new technology will enable Syncrude to remain competitive in the new millennium. Other advantages are increased productivity because automated and robotized components of the system could lead to higher utilization and availability.

The potential disadvantage of the CycEx FPCS is that each cyclic hydraulic and electric shovels will require its own hopper-crusher-slurrification facility. This will lead to higher capital costs of equipment than in Option 1. Besides, the intermittent excavating nature of shovels will render the entire mining system discontinuous or cyclical. The intermittent behaviour of shovels subject the booms to varying stresses. Wear and corrosion in the pipe walls will reduce their life spans and increase the costs of the operation and maintenance. Additionally, the wriggling of the GAP system during mining operations will cause wear in the ball joints. This could lead to frequent leakages and downtimes of the system, and possible problems with the sensors that may be installed on the equipment due to the harsh operating and temperature conditions in the pits (Frimpong et al., 2001a). In addition, any problems with Option 2 such leakages from the flexible slurry pipelines or water pipes could lead to the flooding of the pit floor which will take a much longer time to clean up than in Option 1 where dry bulk material is involved.

The main challenge with this option is how to match the production capacities of the shovels to the crusher-slurrification unit and flexible pipeline system. Another challenge is to ensure that the movements of mobile hopper-crusher and slurrification units are synchronized with that of the shovel movements on the bench to ensure continuous flow of materials from the face with minimum interruptions. It is also important to ensure that the torsional stresses and bending moments on the flexible pipelines used are minimized to prevent sudden pipe breaks that could flood the pit with slurried materials and minimize the power required to transport the materials through the system. Interference with bench material flow must be minimized during extensions or modifications to the flexible pipelines.

3.1.4 Option 3: Shovel-Pipe Wagon Articulating System (CycEx PWA)

The CycEx PWA system comprises a shovel, mobile slurrification system and pipe wagon articulating (PWA) system. A shovel excavates and feeds the oil sands material into the mobile slurrification facility via hoppers and apron feeders. Hot water will be added to the oil sands to produce the oil sands slurry. The resulting oil sands slurry is then pumped through the PWA system to join the HTS train. A schematic diagram of the CycEx PWA system is shown in Fig. 3.7 while Fig. 3.8 shows the plan and side views of the PWA system.

The PWA system comprises a series of crawler-mounted pipe wagons connected to each other by helix reinforced rubber hoses. It has a series of rigid truss frames on castors which are allowed to swivel relative to each other. Each frame will support a 0.61 m (2 ft) diameter slurry pipe and a 0.46 m (1.5 ft) diameter fresh water line of the PWA system. The interconnecting joints are interfaced as shown in Fig. 3.8.

As in Options 1 and 2, the component parts for Option 3 are also available and well proven on the market. Option 3 will adopt a shovel, hybrid tumbler and PWA system. Its potential advantages include (i) lower haulage costs due to the elimination of trucks; (ii) lower requirements for operators; (iii) creation of a safe and environmentally-friendly atmosphere for mining operations; (iv) lower processing costs because the oil sands will be conditioned and processed online in the PWA system en-route to the processing plant; (v) integrated primary and secondary crushing in the hybrid tumbler reduces the downtime caused by oversize lumps choking the system and, (vi) oil sands lumps will be sized to a fraction of -80 mm making it suitable for slurry pipe transportation. Thus, there will be low wear on the pipe walls, minimal or no saltation and bedding in the PWA system and in the downstream pipes and stations (Changirwa et al., 2000). The crawler-mounted units will ensure good traction and mobility of the system on the soft pit floors during periodic relocations giving a clean and level pit floor. In addition, there will be increased productivity because the possible automation of the system could lead to higher availability and utilization of the system. Accordingly, there will be less downtime due to delays at shift changes and the medium degree of adaptation to new technology will enable Syncrude to remain competitive in the new millennium.

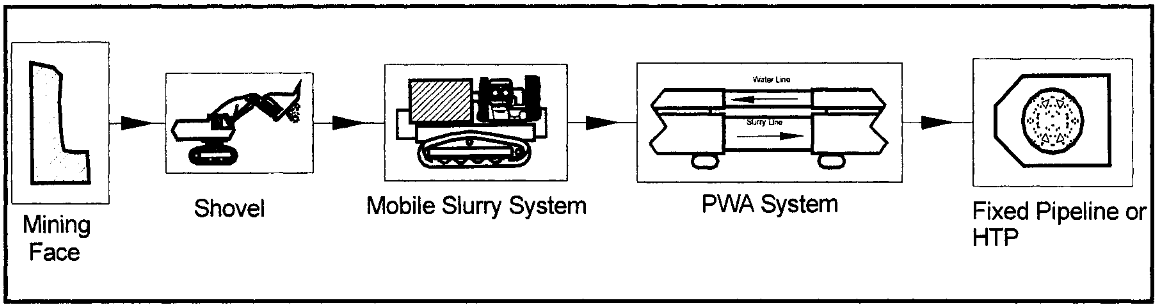


Fig. 3.7 Schematic Diagram of CycEx PWA
(Source: Changirwa et al., 2000)

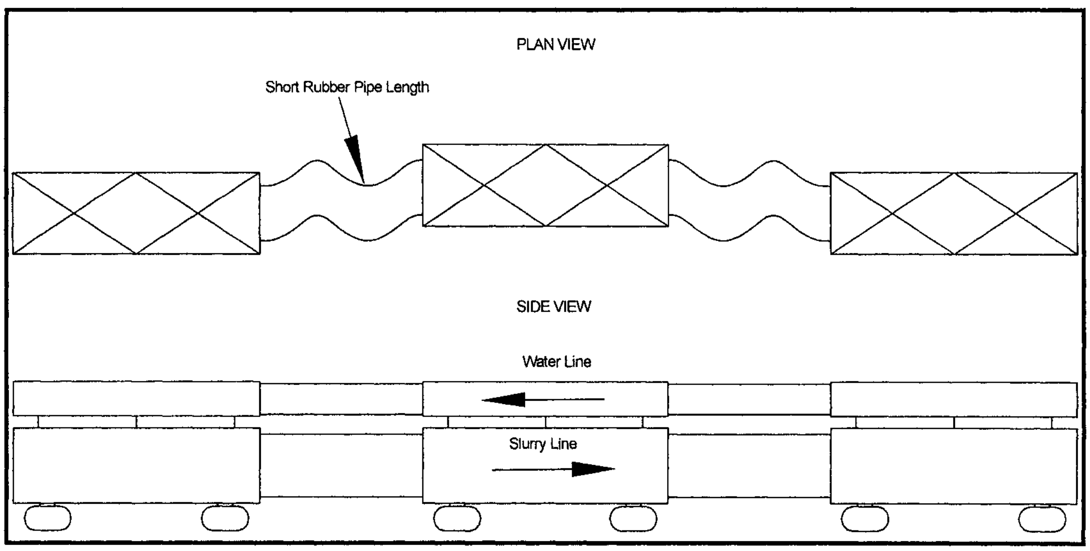


Fig. 3.8 Plan and Side Views of the PWA System
(Source: Changirwa et al., 2000)

The potential demerits of Option 3 include (i) wear and air-entrainment-caused corrosion in the pipes will reduce their life spans; (ii) wriggling of the PWA system during mining operations will cause wear in the short rubber hose connections; (iii) opposite streams of water and slurry are likely to leak in the short rubber hose connections due to repeated articulation thus causing downtime in dismantling and replacing new parts in the system. Conversion to sensor controlled and automated equipment will be expensive. In addition any installed sensors will require protection in the harsh temperature and operating environments leading to additional capital and operating costs (Changirwa et al., 2000). As well each loader in Option 3 will require its own slurrification facility. This will lead to higher capital cost of equipment in Option 3 than in Option 1. Besides, the intermittent excavating nature of shovels will render the entire mining system discontinuous or cyclical. Finally, any leakages in the pipelines, could easily flood the pit floor leading to longer periods of equipment shutdown since it will take a much longer time to clean up than with Option 1.

The main challenges with this system include how to synchronize the movements of the shovel, the mobile slurrying unit and the PWA system so that there will be minimum interruptions in production. Also it is necessary to monitor the tensile, bending and shear stresses imposed on the PWA system such that the short rubber hose connections are not damaged during movements at the face as this could lead to the flooding of the pit with either water or slurried materials.

3.1.5 Option 4: BWE-Conveyor Belt Control System (BWE CBCS)

Option 4 comprises a bucket wheel excavator-belt conveyor-crusher-slurrification unit-HTS system. The BWE will excavate the materials from the face. The materials will be conveyed by high capacity, low temperature resistant belt conveyors to a tumbler/trommel-slurrification unit where it will be crushed, sized, slurried and pumped through the HTS line to the main processing plant. The application of this option assumes that the material can be easily excavated and handled by a new generation of BWEs and belt conveyor systems. Fig. 3.9 is a schematic diagram of the BWE-conveyor belt system.

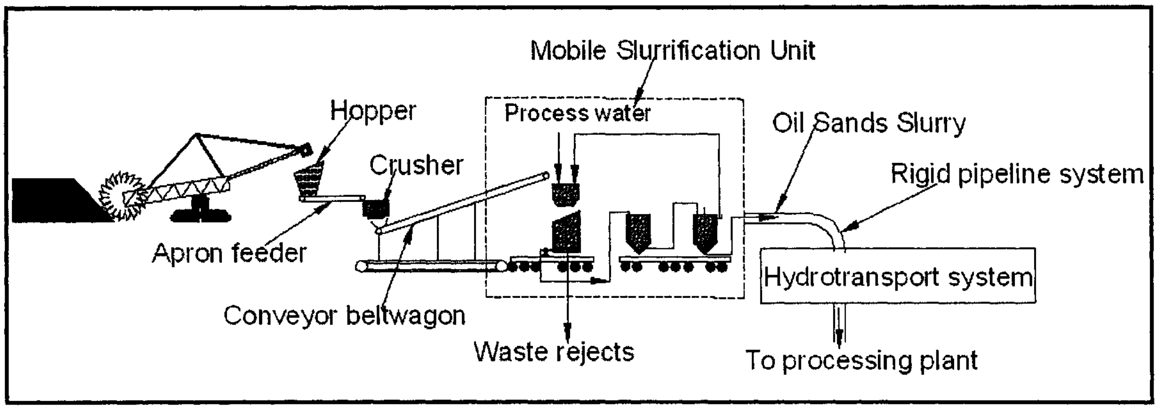


Fig. 3.9 Schematic Diagram of BWE-Conveyor Belt System

BWEs were originally employed at Syncrude Canada Ltd. in conjunction with field conveyors to mine tar sands from its Base and North Mines and to transport the oil sands to the main treatment plant. There were many problems associated with the BWEs and field conveyor belts and resulted in low equipment availabilities and productivities and was eventually replaced by the present shovel-truck system (Kershaw, 1987). These problems included (i) the large weights of the BWE units; (ii) their inability to handle the sticky oil sands materials; (iii) cracking of conveyor belts at temperatures below -40 °C; (iv) frequent failures of conveyor belt splines; and (v) high wear on parts of ore handling equipment due to abrasion; (vi) frost conditions in windrow piles due to the effects of the extremely cold temperatures led to numerous shutdowns. However, new and more powerful BWEs and low temperature resistant belt conveyors have since been developed to solve these problems. The material will be crushed by a double roll crusher, sized and slurried at a slurrification facility and pumped through the main hydrotransport pipeline to the processing plant.

This option which interfaces a continuous excavating system with a continuous materials handling system, has the potential for high productivity, creating an environmentally-friendly atmosphere, low unit operating costs and the requirement for few operators. However, the oil sand deposits in the Athabasca formation are characterized by interburden waste pockets, overburden materials and displaced deposits. In major fault zones within this formation, displaced orebodies could be as far off as 20 m which will make it extremely difficult for selective mining using BWEs. Currently, interburden waste bodies (5 m or less in thickness) are mined as part of the ore resulting in dilution and additional cost for handling and processing waste materials. Due to the varying nature of the McMurray formation, the BWEs will be exposed to complex forces particularly due to the embedded shales and limestones in the oil sands. Thus the BWE bucket, cables and component parts will be exposed to unexpected high forces leading to high wear and tear, and damage to the moving parts and prolonged downtimes.

BWEs are designed to handle materials with a cutting resistance of less than 70 kg/cm (4,700 lb/ft), and compressive strengths below 12 MPa. Thus they may not be suitable for handling the varying rock strengths and diggabilities associated with the Ft. McMurray oil sands formation (Durst and Vogt, 1988; Sagner, 1990). As well, due to the high abrasiveness of the oil sands materials, there is a high potential for increased wear on the bucket and teeth of the BWEs leading to increased downtimes, operating and maintenance costs. In addition, the stickiness of the oil sands materials (particularly in the summer), will present difficulties in discharging the excavated materials from the buckets of the BWE. This will require the bucket wheel to be run at lower speeds leading to lower production rates. Additionally, the large weights of the BWEs and bridge conveyors will require pit floors with high soil bearing characteristics. Besides BWEs have low mobilities due to their weight and size, and cannot be easily moved from one pit face to the other. This could lead to delays in production scheduling to ensure an even feed of uniform grade materials to the processing plant.

The main challenges with this system include the proper choice of the type of BWEs with the required cutting forces to excavate the in-situ oil sands material, as well as handling the abrasive and sticky oil sands materials efficiently all year round. Another challenge will be how to match the operational capacities of the train of hopper-crusher-slurrification units with that of the BWEs. The mobile train of bulk material transportation units will have to fit within the pit to accommodate the movements of the BWEs at the faces. This will require the prior excavation and completion of a large initial cut before the option can be employed. The ground bearing characteristics also have to be carefully studied and monitored to allow for the efficient movement and operation of BWEs, crushing and slurrification units.

3.1.6 Option 5: Dredge Pipeline Control System (DRE PCS)

Option 5 comprises a dredge mounted on a pontoon and connected to the main extraction plant by means of either flexible or rigid pipelines that will convey the slurried materials. The crusher-slurrification unit may not be mounted on the pontoon. The type of dredge will depend on the depth (thickness) and nature of the materials to be excavated but it will involve a bucket ladder dredge, a bucket wheel suction dredge or a suction ladder dredge. This option requires the presence of water or the creation of an artificial lake around the mining area. Fig. 3.10 is a schematic diagram of Option 5.

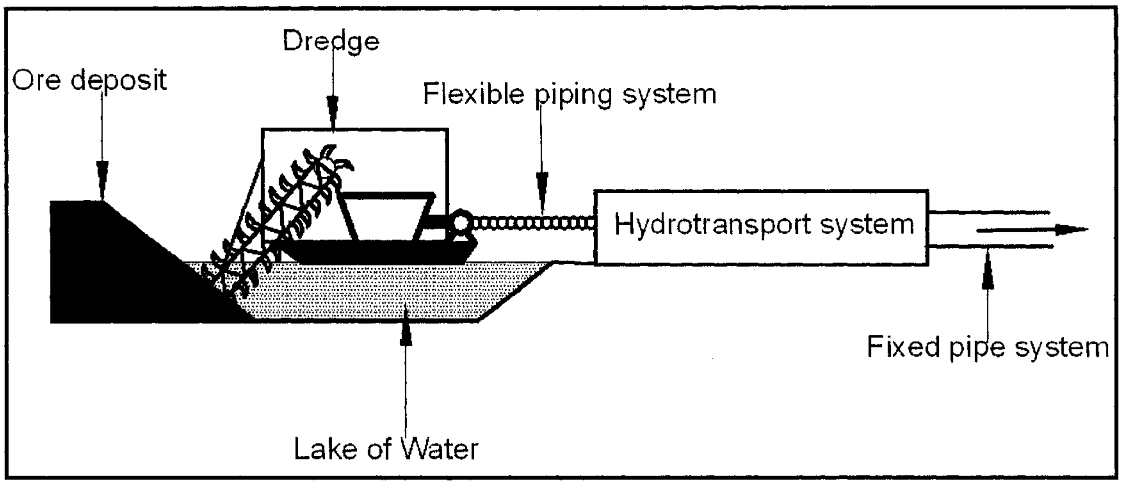


Fig. 3.10 Schematic Diagram of Dredge Pipeline Control System (DRE PCS)

The major features that will permit the use of dredges are that (i) the material in place can be easily disintegrated by either mechanical and hydraulic action; (ii) the presence of large amounts of water; (iii) adequate space available for waste or tailings disposal; (iv) low relief of the area to allow for hydraulic transport of slurried materials; and (v) the ability to meet environmental water quality standards and regulations.

The major challenges of this option are that (i) the dredges must be able to handle the sticky oil sands materials during excavation; (ii) the buckets must have enough cutting force to excavate the materials; (iii) there must be adequate and safe means to prevent the pools of water from freezing during cold winters that are common in the Ft. McMurray area; and (iv) meet the requirements of all environmental laws and regulations with regard to the possible spillages, pollution or flooding of nearby surface and underground water sources from the dredging operations.

3.1.7 Option 6: Hydraulic Mining System (HYDS)

This option involves the use of high pressure water jets from remotely controlled monitors directed at the mining face or bank to break up the oil sands material, convert it into slurry and pump it through rigid or flexible pipelines to the main processing plant for further treatment. As in Option 5, this option requires the presence of large amounts of water for the hydraulicking, slurrification and educing processes. Fig. 3.11 is a schematic diagram of the hydraulic mining system.

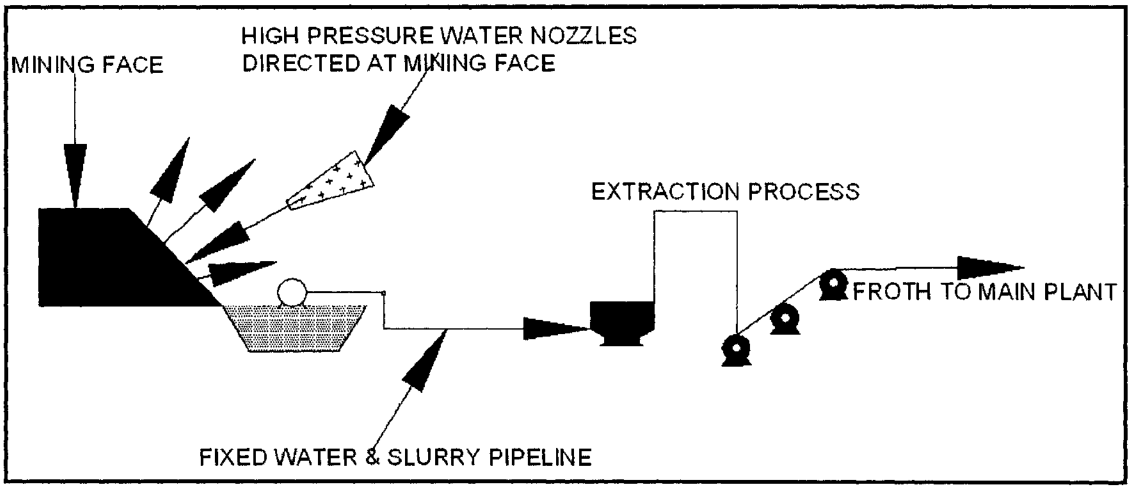


Fig. 3.11 Schematic Diagram of the Hydraulic Mining System (HYDS)

Option 6 requires unconsolidated materials of low dips (2 – 6°) which can easily be disintegrated by the high pressure water jets and the maximum bank height must not exceed 60 m. It also requires an elaborate layout to collect all the fragmented material from the face (which is in the form of a slurry) and pump it through pipelines to the main processing plant. Some of the advantages of this option include high productivity (70 to 230 m^3 per manshift), low mining cost, low capital cost due to the use of simple equipment, and the operations can be automated.

The challenges of this option are (i) ensuring a continuous supply of water all year round for the operations; (ii) meeting the requirements of all environmental laws and regulations with regard to the possible spillages, pollution or flooding of nearby surface and underground water sources from the runoffs from operations. Due to the sticky and oily nature of the tar sands materials, it will be necessary to use hot water jets to break up and dissolve the materials into slurry. This will involve a complicated and expensive setup of a water heating plant to provide the hot water requirements. It will also present problems in the safe handling and use of hot water at the face especially during the cold winter temperatures that are characteristic of the Ft. McMurray area.

3.1.8 Option 7: In Situ Recovery Techniques (INS REC)

In situ recovery techniques involve mainly cyclic steam stimulation (CSS) and steam assisted gravity drainage (SAGD) methods. Both systems require the production and injection of steam at very high temperatures of about 300 °C and pressures averaging 11,000 kPa. The high pressured hot water is injected into either horizontal or vertical boreholes to heat and crack the in situ material to allow paths for the melted fluid to flow into well bore to be pumped up for processing. In situ recovery techniques are applicable to parts of oil sands formation that are so deeply buried that they cannot be economically mined by surface mining techniques. The CSS method is known to have low recoveries between 20 to 25% while that of SAGD could be up to 60% depending on the layout of the horizontal injection and production boreholes (Anon., 2000b).

The main advantages of the in situ recovery techniques are that they are applicable to portions of the deposit that are so deeply buried that surface mining techniques cannot be employed to extract them. In addition, they cause less damage to the surface environment as there is no need for large scale waste stripping to allow for ore mining. The major disadvantages include the fact that the methods require large capital outlays in the form of equipment for drilling the vertical and horizontal production and recovery boreholes and shafts. Other disadvantages include high operating costs as large amount of energy is consumed to produce the steam at the required high temperatures and pressures to ensure that the methods work properly, and the low recoveries of the CSS and SAGD systems.

The challenges involved in the application of Option 7 are how to adapt these in situ mining methods, which are designed for the extraction of deep-seated deposits, to the extraction of near-surface deposits. Other challenges include getting access to reliable sources of cheap energy to produce the large amounts of steam required for the successful operation of the methods.

3.1.9 Capital and Installation Cost of AFS Options

The capital cost of the shovel is common to the CMS and Options 1 to 3. Option 2 will involve sophisticated technology which will result in larger capital investments and installation costs than belt conveyor wagons and PWA systems in Options 1 and 3 respectively. Apart from the shovel, the major capital and installation cost of components in Option 2 include the use of multiple untested dual ball joint systems together with short lengths metal-helix reinforced rubber connections. Other major cost sectors will involve (i) the instrumentation and control systems for ground articulated pipeline (GAP) reticulation and articulation and, (ii) hybridization of hopper, apron feeder, crusher, trommel, pumpbox/pump, swivel joint and tumbler units into a hybrid tumbler. Most of these items are not part of the conveyor belt wagons and PWA systems. On the other hand, major capital and installation cost of components in Option 1 will consist of a hybrid mobile crusher, conveyor belt wagons consisting of custom-made belt wagons, semi-mobile mixing tower, and semi-mobile pump systems whereas Option 3 will include a hybrid tumbler and PWA system. Studies conducted and reported by Changirwa et al. (2000) show that the combined capital and installation costs of mobile crusher, conveyor belt wagons, and semi-mobile pump systems in Option 1 will be equivalent to those of the hybrid tumbler and PWA systems in Option 3. However, there is likelihood of continuous material flow in Option 1 than in Options 2 and 3 because the outputs from several shovel-belt conveyor systems are fed to one slurrification plant. Besides, handling of any spillages due to a problem with any of the shovel-conveyor belt systems will be faster and more easily done with Option 1 (involving dry materials) than with Options 2 and 3 which involved slurrified material.

Options 4 to 7 will involve high initial capital investment in the major equipment like BWEs, dredges, mobile conveyor bridges and field conveyors. As well, expensive large diameter drilling machines for the injection and production boreholes together with shaft sinking and drift development equipment will be required for the successful operation of the CSS and SAGD methods. In addition, Options 4 to 7 will require a lot of engineering plans and strategies to mitigate their adverse effects on the environment and to meet the many stringent environmental laws and regulations. In general, the operating costs of Options 4 to 7 will be higher than those of the CMS and Options 1 to 3.

3.2 Recommended Mining Options

Options 2 and 3 will involve higher capital costs in equipment than Option 1 because each shovel in Options 2 and 3 will require its own slurrification unit. Both options will lead to discontinuous material flow in the system due to the cyclical nature of the single shovel feeding the slurrification facility. Spillages in Options 2 and 3 will be more difficult to handle than in Option 1 and could lead to pit floor flooding and long periods of production shutdowns. In addition, Options 2 and 3 are unlikely to meet the production requirements of Syncrude due to the expected high torsional and bending stresses that will be imposed on the flexible pipelines. Also it will be difficult to design flexible pipelines that can withstand the extreme operational field temperatures ranging from –40 °C to +35 °C as pertains at Ft. McMurray. The complicated design of the pipe joints that will carry the water and slurry in the flexible pipes will lead to many anticipated downtimes. When the BWE-belt conveyor system was employed at the Base and North mines at Syncrude, the extremely low winter temperatures led to frequent failures in the component parts of the BWE and field belt conveyors which resulted in frequent shutdowns and downtimes (Kershaw, 1987). It may be extremely difficult to convince management to revert to Option 4. Besides the high cohesiveness and abrasiveness, varying composition, hardness and diggabilities of the oil sands formations at Ft. McMurray may seriously affect the performance of BWEs and dredges as envisaged in Options 4 and 5. Also the use of high pressure water jets may not be able to break up the cohesive tar sand formations as envisaged in Option 6. The application of in situ recovery techniques (Option 7) to near surface oil sands deposits that are presently being mined at Ft. McMurray will make the option uncompetitive with other surface mining options. Based on the above discussions, Option 1 seems to be a more promising option for the AFS design. However, further study on all the options may be required to before being discounted from oil sands mining systems.

3.3 Summary

In this Chapter, the CMS and seven conceptual AFS options were outlined together with their prominent components. The advantages and disadvantages as well as the various challenges facing the CMS and each of the AFS options were identified. It is noted that the CMS and various AFS options involve the combination of cyclical and continuous loaders and materials handling equipment such as shovels, BWEs, dredges, rigid/flexible pipelines, belt conveyor wagons, hoppers, crushers, slurrification units and pumps. Option 1 is selected to form the basis for this research study because it is more promising and potentially more capable of meeting the production, efficiency and economic requirements of Syncrude.

CHAPTER 4

DYNAMIC EVOLUTION OF AFS LAYOUT AND MATERIAL FLOW

4.0 Introduction

The dynamic evolution of the AFS layouts and material flow on the multi-bench, multi-face pit configurations associated with the CMS and Option 1 are analyzed mathematically in this chapter. Integral calculus, solid geometry using 3D ellipsoid of expansion and parabolic partial differential equations are used to model and analyze the continuous flow of the materials by volume. The initial and boundary conditions in the material flow of the two mining methods are enumerated.

In order to analyze the various geometrical layouts and their associated bench material flow simulation models with the CMS and AFS option, the relative locations and movements of the excavators in the multi-bench, multi-face pits have to be studied. In addition, the geometrical changes of the pit faces, quantity and grade of materials excavated from the various faces, the production capacities of the excavators and materials handling systems within the pit in continuous time paradigm have to be incorporated into the mathematical models of the options. In the following sections, mathematical models of the geometrical changes in the pit volumes and dimensions are discussed.

4.1 Dynamic Geometric Models of Surface Mine Layouts

The dynamic geometrical models of the mine layouts may be obtained by using various mathematical techniques. These techniques involve integral calculus, solid geometry using the 3D ellipsoid and by parabolic partial differential equations to model and analyze the continuous flow of materials by volume. The volume of materials generated by incremental changes in the pit dimensions are calculated by a combination of the truncated cone or frustum methods and parabolic partial differential equations.

4.1.1 Volume of Materials Excavated from First Bench of Circular Pit

From Fig. 4.1, assuming a right circular conical cross-section, the volume of the frustum with radius a_0 and a_1 at the top and bottom respectively and a bench height, H, is given by (Etgen, 1995; Frimpong et al., 2001):

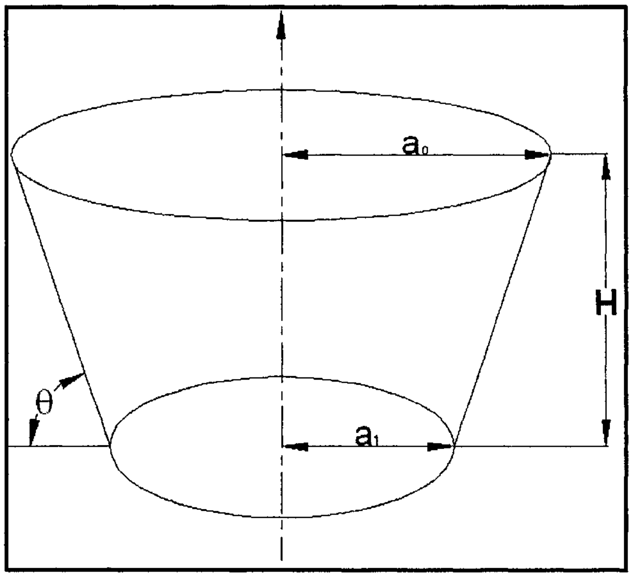


Fig. 4.1 Frustum on First Bench with Radii a_o and a_1

$$V_1 = \frac{\pi H}{3}\left[a_o^2 + a_o\, a_1 + a_1^2\right] \tag{4.1}$$ [§]

But $a_1 = a_o - H/\tan\theta = a_o - k$; where $k = H/\tan\theta$

Thus the volume of materials excavated from the frustum in Fig. 4.1 is:

$$V_1 = \frac{\pi H}{3}\left[3a_o(a_o - k) + k^2\right] \tag{4.2}$$

From Fig. 4.2, if an incremental pushback Δx is excavated in the x-direction and the geometric shape of the frustum is maintained, both a_o and a_1 will increase by Δx. Thus the new volume of the frustum, $V_1^{'}$, is given by:

$$\left(V_1^{'}\right)_{\Delta x} = \frac{\pi H}{3}\left[3(a_o + \Delta x)(a_o + \Delta x - k) + k^2\right] \tag{4.3}$$

Let the bench be denoted Bench #1. The change in volume of materials excavated from the incremental pushback on Bench #1 is the difference between $V_1^{'}$ and V_1 which is given by:

$$(\Delta V_1)_{\Delta x} = \left(V_1^{'}\right)_{\Delta x} - V_1 = \pi H \Delta x(2a_o + \Delta x - k) \tag{4.4}$$

[§] All terms are defined in the Nomenclature

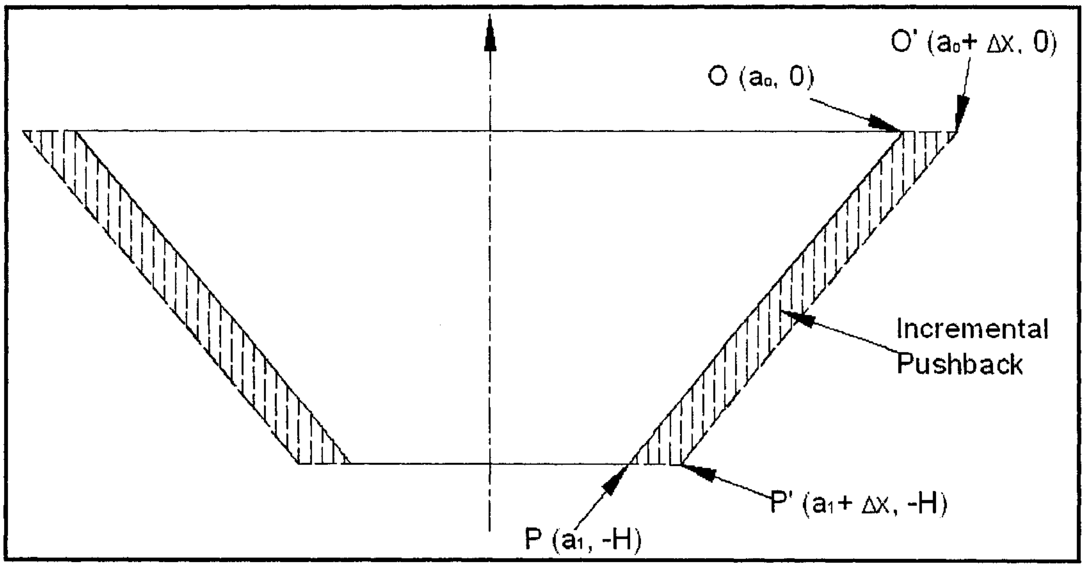


Fig. 4.2 Longitudinal Section of Bench #1 after One Incremental Pushback, Δx

When a second incremental pushback is taken on the same bench, the volume of materials extracted is given by:

$$\left(V_1^{'}\right) = \frac{\pi H}{3}\left[3(a_o + 2\Delta x)(a_o + 2\Delta x - k) + k^2\right] \tag{4.5}$$

Accordingly, the change in volume from the previous pushback, i.e. $\left(V_1^{'}\right)_{2\Delta x} - \left(V_1^{'}\right)_{\Delta x}$ is given by:

$$\Delta V_{12}^{'} = \pi H \Delta x(2a_o + 3\Delta x - k) \tag{4.6}$$

Likewise, the change in volume between the second and third incremental pushbacks on Bench #1 is given by:

$$\Delta V_{23}^{'} = \pi H \Delta x(2a_o + 5\Delta x - k) \tag{4.7}$$

The general equation for the volume of materials excavated from the n^{th} incremental pushback on Bench #1 is given by:

$$\Delta V_n^{'} = \pi H \Delta x[2a_o + (2n - 1)\Delta x - k] \tag{4.8}$$

4.1.2 Volume of Materials Excavated from Second Bench of Circular Pit

Mining on Bench #2 begins when pit radius on Bench #1 has advanced by a minimum distance of B + H/tanθ. From Fig. 4.3, the volume of the frustum formed using the coordinates of points R (a_2, -H) = (a_o – k – B, -H) and Q (a_3, -2H) = (a_o – 2k – B, -2H) on the second bench, Bench #2, is given by:

$$V_2 = \frac{\pi H}{3}\left[a_2^2 + a_2 a_3 + a_3^2\right] \tag{4.9}$$

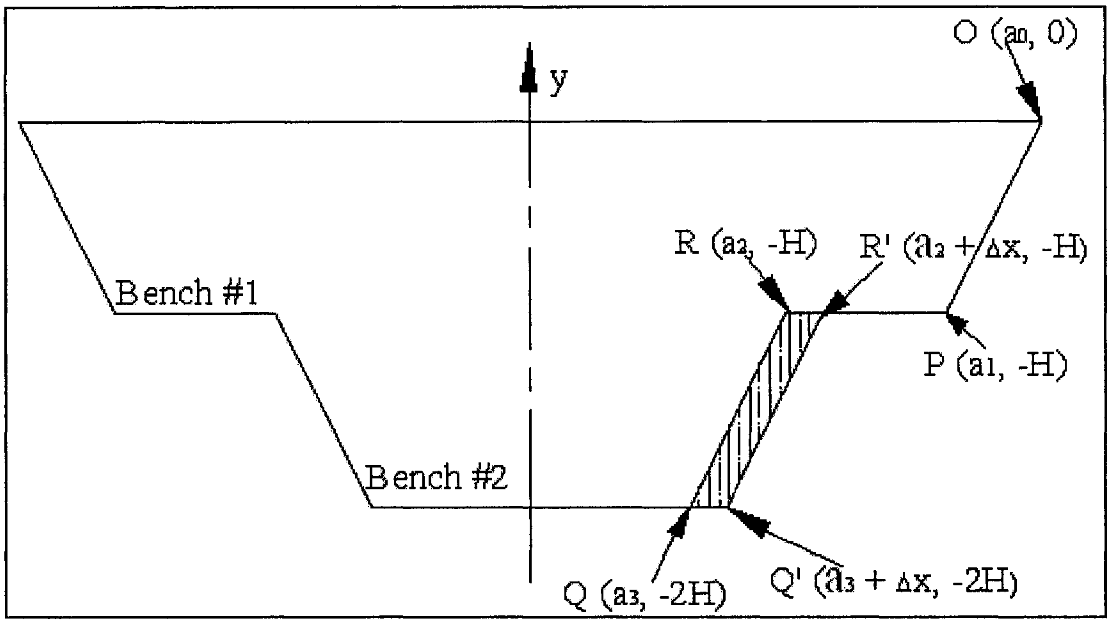


Fig. 4.3 Section of Bench #2 after Excavating One Incremental Pushback, Δx

But $a_3 = a_2 – H/\tan\theta = a_2 – k$. Substituting for a_2 and a_3 into equation (4.9) gives:

$$V_2 = \frac{\pi H}{3}\left[3a_2(a_2 - k) + k^2\right] \tag{4.10}$$

If an incremental pushback, Δx, is excavated on the second bench, the new coordinates of points R and Q (see Fig. 4.3) become: R′ (a_2 + Δx, -H) = R′ (a_o - k - B + Δx, -H) and Q′ (a_3 + Δx, -2H) = Q′ (a_o - 2k - B + Δx, -2H) respectively. Thus the volume of the new frustum generated is given by:

$$\left(V_2'\right)_{\Delta x} = \frac{\pi H}{3}\left[3(a_2 + \Delta x)(a_2 + \Delta x - k) + k^2\right] \tag{4.11}$$

The change in volume, ΔV_2 from V_2 to $(V_2')_{\Delta x}$ is given by:

$$(\Delta V_2)_{\Delta x} = \left(V_2'\right)_{\Delta x} - V_2 = \pi H \Delta x(2a_o + \Delta x - 3k - 2B) \tag{4.12}$$

Similarly, the changes in volume after taking two and three incremental pushbacks are given by the following equations:

$$\Delta V'_{12} = \pi H \Delta x (2a_o + 3\Delta x - 3k - 2B) \tag{4.13}$$

$$\Delta V'_{23} = \pi H \Delta x (2a_o + 5\Delta x - 3k - 2B) \tag{4.14}$$

The general expression for the change in volume between incremental pushbacks *i* and *j* on Bench #2 is given by:

$$(\Delta V'_2)_{ij} = \pi H \Delta x (2a_o + (i + j)\Delta x - 3k - 2B) \tag{4.15}$$

Likewise, the changes in volume of the frustums between subsequent incremental pushbacks of Δx on the third and fourth benches are given respectively by:

$$(\Delta V'_3)_{ij} = \pi H \Delta x (2a_o + (i + j)\Delta x - 5k - 4B) \tag{4.16}$$

$$(\Delta V'_4)_{ij} = \pi H \Delta x (2a_o + (i + j)\Delta x - 7k - 6B) \tag{4.17}$$

From the preceding equations it can be deduced that the change in volume of the frustum between the i^{th} and j^{th} incremental pushback of length Δx mined on the n^{th} bench is given by:

$$(\Delta V'_n)_{ij} = \pi H \Delta x [(2a_o + (i + j)\Delta x - k)\, 2(n - 1)(k + B)] \tag{4.18}$$

If V_{CUM} denotes the cumulative volume of materials excavated from the benches, then:

$$V_{CUM} = V_1 + V_2 + V_3 + \ldots + V_n = \sum_{i=1}^{n} V_i \tag{4.19}$$

V_1, V_2, V_3, V_n are the total volumes of the frustums excavated on the first, second, third to n^{th} benches respectively.

Disregarding the volumes of the initial cuts on the various benches, the cumulative volume of materials excavated from incremental pushbacks, Δx, on the various benches, V_{ICUM} is given by:

$$V_{ICUM} = \sum_{i=0}^{n} (\Delta V_n) = (\Delta V_1 + \Delta V_2 + \Delta V_3 + \cdots + \Delta V_n) \tag{4.20}$$

The cumulative incremental volume of the material flow on the first four benches is given by:

$$V_{ICUM} = \Delta V_1 + \Delta V_2 + \Delta V_3 + \Delta V_4 = 4\pi H\Delta x[(2a_o + \Delta x - k) - 3(k + B)] \quad (4.21)$$

A general equation can be derived for the cumulative volume of material flow from *n* benches as:

$$V_{ICUM} = n\pi H\Delta x[(2a_o + \Delta x - k) - (n - 1)(k + B] = n\Delta V_1 - n\pi H\Delta x(n - 1)(k + B) \quad (4.22)$$

The cumulative volume of material flow from the benches comprises ore, waste and stockpile materials. If $V_{CUM(O)}$, $V_{CUM(W)}$ and $V_{CUM(SP)}$ denote the cumulative volumes of ore, waste and stockpile respectively, then the cumulative volume is given by:

$$V_{CUM} = V_{CUM(O)} + V_{CUM(W)} + V_{CUM(SP)} \quad (4.23)$$

The cumulative tonnage, T_{CUM} of materials moved from the benches is given by:

$$T_{CUM} = \rho_o V_{CUM(O)} + \rho_W V_{CUM(W)} + \rho_{SP} V_{CUM(SP)} \quad (4.24)$$

4.1.3 Volume of Materials Excavated from an Elliptical Shaped Pit

Fig. 4.4 shows a pit with an elliptical cross-section. If an incremental push back, Δx, is taken, the change in volume of the ellipsoid can be obtained from solid geometry. If the dimensions of the major and minor axes at the top of frustum in Fig. 4.4 are $2a_o$ and $2b_o$ respectively, while those of the ellipse at a depth of one bench height, H, are $2a_1$ and $2b_1$ respectively, then the areas of the ellipses at the top and bottom of the frustum are given by equation (4.25).

$$\left.\begin{aligned} A_o &= \pi a_o b_o \\ A_1 &= \pi a_1 b_1 \end{aligned}\right\} \quad (4.25)$$

From Fig. 4.4, for a bench slope angle of θ, then $a_1 = a_o - H/\tan\theta$ and $b_1 = b_o - H/\tan\theta = a_o - k$. The volume of the elliptical frustum is given by:

$$V = \frac{H}{3}\left[A_o + A_1 + (A_o + A_1)^{0.5}\right] \quad (4.26)$$

Substituting for A_o and A_1 into equation (4.26), the volume of the frustum is given by:

$$V = \frac{\pi H}{3}\left[(a_o b_o + a_1 b_1) + (a_o b_o a_1 b_1)^{0.5}\right] \quad (4.27)$$

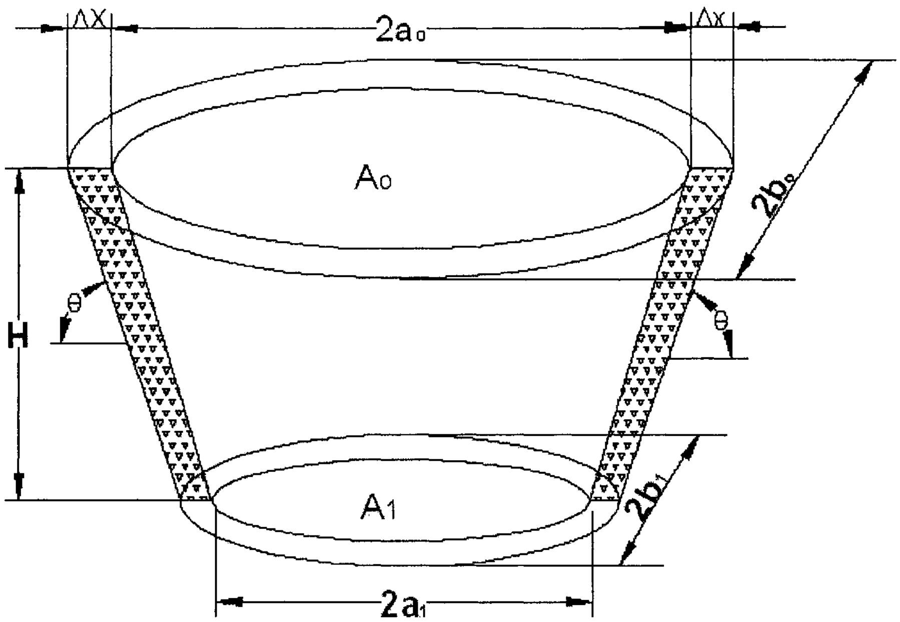


Fig. 4.4 Elliptical Pit Section with one Incremental Pushback, Δx

Substituting for a_1 and b_1 into equation (4.27) the volume of an elliptical frustum in terms of a_o and b_o is given by:

$$V = \frac{\pi H}{3}\left[2a_o b_o - k(a_o + b_o - k) + \sqrt{a_o b_o (a_o - k)(b_o - k)}\right] \tag{4.28}$$

If an incremental pushback, Δx, is taken along both the major and minor axes, then the radii of the top and bottom ellipses increase by the same amount as illustrated by:

$$\left.\begin{aligned} a_o^{'} &= a_o + \Delta x; \quad a_1^{'} = a_1 + \Delta x \\ b_o^{'} &= b_o + \Delta x; \quad b_1^{'} = b_1 + \Delta x \end{aligned}\right\} \tag{4.29}$$

The new areas of the ellipses at the top and bottom of the frustum are given by:

$$\left.\begin{aligned} A_o^{'} &= \pi a_o^{'} b_o^{'} = \pi(a_o + \Delta x)(b_o + \Delta x) \\ A_1^{'} &= \pi a_1^{'} b_1^{'} = \pi(a_1 + \Delta x)(b_1 + \Delta x) \end{aligned}\right\} \tag{4.30}$$

The new volume of the frustum is given by:

$$V_1^{'} = \frac{\pi H}{3}\left[\begin{aligned} &2(a_o + \Delta x)(b_o + \Delta x) \ - \ k(a_o + b_o + 2\Delta x - k) \\ &+ \sqrt{(a_o + \Delta x)(b_o + \Delta x)} \times \{(a_o + \Delta x - k)(b_o + \Delta x - k)\}^{0.5} \end{aligned}\right] \tag{4.31}$$

The change in volume of the elliptical frustum, ΔV, from V to V′ is given by the difference between equations (4.28) and (4.31).

For *n* incremental pushbacks, the new volume of the pit will be given by:

$$V_n' = \frac{\pi H}{3}\left[\begin{array}{l} 2(a_o + n\Delta x)(b_o + n\Delta x) - k(a_o + b_o + 2n\Delta x - k) \\ + \sqrt{(a_o + n\Delta x)(b_o + n\Delta x)} \times \{(a_o + n\Delta x - k)(b_o + n\Delta x - k)\}^{0.5} \end{array}\right] \quad (4.32)$$

The change in volume of the elliptical frustum, ΔV, from one incremental pushback to the next is given by:

$$\Delta V = V_n' - V_{n-1}' \quad (4.33)$$

4.2 Waste Mining Model

The volume of waste mined can be determined from the volume of ore mined. If the stripping ratio is SR and the volume of ore mined at any given time is V_O, then the volume, V_W and tonnage, T_W, of waste stripped are given by:

$$V_W = SR \times V_o \quad (4.34)$$

$$T_o = \rho_o \times V_o \quad (4.35)$$

$$T_W = \rho_W \times V_W = SR(\rho_o \times V_o) = SR \times T_o \quad (4.36)$$

4.3 Continuous Time Formulation of Bench Material Flow

In order to calculate the rate of change in the volume of the pit when the incremental pushbacks are being taken in only one direction, it is necessary to employ continuous time formulation to model the changes in pit volume. Using the chain rule, the rate of change in the volume of the pit with circular and elliptical cross-sections in any direction with time are determined by partial differentiation of the volume with respect to the dimensions of the pit (i.e. a, b, H and k) in the following sections.

4.3.1 Circular Shaped Frustum

For a circular shaped frustum, the volume is given by:

$$V(a, b, H) = \frac{\pi H}{3}\left(a^2 + ab + b^2\right) \quad (4.37)$$

But in a circular frustum, b = a – k = a – H/tanθ. Thus the total volume is given by:

$$V(a, k, H, k) = \frac{\pi H}{3}\left(3a^2 - 3ak + k^2\right) \quad (4.38)$$

From the chain rule:

$$\frac{dV}{dt} = \frac{\partial V}{\partial a} \times \frac{\partial a}{\partial t} + \frac{\partial V}{\partial H} \times \frac{\partial H}{\partial t} + \frac{\partial V}{\partial k} \times \frac{\partial k}{\partial t} \quad (4.39)$$

Applying the chain rule to equation (4.38) results in:

$$\frac{dV}{dt} = \pi H(2a - k) \times \frac{\partial a}{\partial t} + \frac{\pi}{3}\left(3a^2 - 3ak + k^2\right) \times \frac{\partial H}{\partial t} + \frac{\pi H}{3}(2k - 3a) \times \frac{\partial k}{\partial t} \quad (4.40)$$

Since the bench height, H, bench slope angle, θ, and the geotechnical parameters of the rock do not change with time, ∂H/∂t, ∂θ/∂t and ∂k/∂t thus equation (4.40) simplifies to equation (4.41).

$$\frac{dV}{dt} = \pi H(2a - k) \times \frac{\partial a}{\partial t} \quad (4.41)$$

4.3.2 Elliptical Shaped Frustum

A similar analysis for an elliptical frustum of volume given by equation (4.28) results in the following system of partial differential equations:

$$V(a, b, H, k) = \frac{\pi H}{3}\left[2ab - k(a + b -) + \sqrt{ab}\{(a - k)(b - k)\}^{0.5}\right] \quad (4.42)$$

$$\frac{dV}{dt} = \frac{\partial V}{\partial a} \times \frac{\partial a}{\partial t} + \frac{\partial V}{\partial b} \times \frac{\partial b}{\partial t} + \frac{\partial V}{\partial H} \times \frac{\partial H}{\partial t} + \frac{\partial V}{\partial k} \times \frac{\partial k}{\partial t} \quad (4.43)$$

$$\frac{dV}{da} = \frac{\pi H}{3}\left[2b - k + \frac{1}{2}\sqrt{\frac{b}{a}}\{(a - k)(b - k)\}^{0.5} + \frac{\sqrt{ab}}{2}\left\{\frac{b - k}{[(a - k)\quad(b - k)]^{0.5}}\right\}\right] \quad (4.44)$$

$$\frac{dV}{db} = \frac{\pi H}{3}\left[2a - k + \frac{1}{2}\sqrt{\frac{a}{b}}\{(a - k)(b - k)\}^{0.5} + \frac{\sqrt{ab}}{2}\left\{\frac{a - k}{[(a - k)(b - k)]^{0.5}}\right\}\right] \quad (4.45)$$

$$\frac{dV}{dH} = \frac{\pi}{3}\left[2ab - k(a + b - k) + \sqrt{ab}\{(a-k)(b-k)\}^{0.5}\right] \tag{4.46}$$

$$\frac{dV}{dk} = \frac{\pi H}{3}\left[2k(k+1)(a+b) + \frac{\sqrt{ab}}{2}\left\{\frac{2k - a - b}{[(a-k)(b-k)]^{0.5}}\right\}\right] \tag{4.47}$$

Substituting equations (4.44) to (4.47) into equation (4.43), the rate of change in volume the elliptical frustum is given by:

$$\frac{dV}{dt} = \begin{cases} \dfrac{\pi H}{3}\left[2b - k + \dfrac{1}{2}\sqrt{\dfrac{b}{a}}\{(a-k)(b-k)\}^{0.5} + \dfrac{\sqrt{ab}}{2}\left\{\dfrac{b-k}{[(a-k)\quad(b-k)]^{0.5}}\right\}\right] \times \dfrac{\partial a}{\partial t} \\[2ex] + \dfrac{\pi H}{3}\left[2a - k + \dfrac{1}{2}\sqrt{\dfrac{a}{b}}\{(a-k)(b-k)\}^{0.5} + \dfrac{\sqrt{ab}}{2}\left\{\dfrac{a-k}{[(a-k)(b-k)]^{0.5}}\right\}\right] \times \dfrac{\partial b}{\partial t} \\[2ex] + \dfrac{\pi}{3}\left[2ab - k(a+b-k) + \sqrt{ab}\{(a-k)(b-k)\}^{0.5}\right] \times \dfrac{\partial H}{\partial t} \\[2ex] + \dfrac{\pi H}{3}\left[2k(k+1)(a+b) + \dfrac{\sqrt{ab}}{2}\left\{\dfrac{2k-a-b}{[(a-k)(b-k)]^{0.5}}\right\}\right] \times \dfrac{\partial k}{\partial t} \end{cases} \tag{4.48}$$

But $\partial H/\partial t = 0$ and $\partial k/\partial t = 0$, so equation (4.48) simplifies to:

$$\frac{dV}{dt} = \begin{cases} \dfrac{\pi H}{3}\left[2b - k + \dfrac{1}{2}\sqrt{\dfrac{b}{a}}\{(a-k)(b-k)\}^{0.5} + \dfrac{\sqrt{ab}}{2}\left\{\dfrac{b-k}{[(a-k)\quad(b-k)]^{0.5}}\right\}\right] \times \dfrac{\partial a}{\partial t} \\[2ex] + \dfrac{\pi H}{3}\left[2a - k + \dfrac{1}{2}\sqrt{\dfrac{a}{b}}\{(a-k)(b-k)\}^{0.5} + \dfrac{\sqrt{ab}}{2}\left\{\dfrac{a-k}{[(a-k)(b-k)]^{0.5}}\right\}\right] \times \dfrac{\partial b}{\partial t} \end{cases} \tag{4.49}$$

Thus, given the values of a, b, H, k and θ, the rate of change in volume of either the circular frustum or the elliptical frustum with time, $\partial V/\partial t$, can be calculated using equations (4.41) and (4.49) respectively.

4.4 Bench Material Flow Dynamics

From Fig. 4.4, the volume of the pit layout resulting from lateral and longitudinal expansions due to incremental pushbacks, V', at any given time is a function of the changes in the radii of the ellipsoid and the geometry of the layout given by (Frimpong et al., 1998):

$$V' = \psi[\phi_1(a, NF_N, NE_N, A, U, t, L_N); \phi_2(b, NF_E, NE_E, A, U, t, L_E); \phi_3(\theta, H)] \qquad (4.50)$$

The magnitude of the change in volume in equation (4.50) depends on the production planning and scheduling requirements.

Let $\psi_1 = \phi(a, N_{FN}, NE_N, A, U, t, L_N)$; $\psi_2 = \phi(a, N_{FE}, NE_E, A, U, t, L_E)$ and $\psi_3 = \phi(\theta, H)$. Finding the partial differential of V' with respect to *t* results in (Frimpong et al., 2001):

$$\left.\begin{aligned} \frac{\partial V'}{\partial t} &= \frac{\partial V'}{\partial \psi_{1(t)}}\Delta\psi_1 + \frac{\partial V'}{\partial \psi_{2(t)}}\Delta\psi_2 + \frac{\partial V'}{\partial \psi_{3(t)}}\Delta\psi_3 + \frac{1}{2}\frac{\partial^2 V'}{\partial \psi_{1(t)}^2}(\Delta\psi_1)^2 + \frac{1}{2}\frac{\partial^2 V'}{\partial \psi_{2(t)}^2}(\Delta\psi_2)^2 \\ &+ \frac{1}{2}\frac{\partial^2 V'}{\partial \psi_{3(t)}^2}(\Delta\psi_3)^2 + \frac{1}{2}\frac{\partial^2 V'}{\partial \psi_1\,\partial \psi_2}(\Delta\psi_1)(\Delta\psi_2) + \frac{1}{2}\frac{\partial^2 V'}{\partial \psi_1\,\partial \psi_3}(\Delta\psi_1)(\Delta\psi_3) \\ &+ \frac{1}{2}\frac{\partial^2 V'}{\partial \psi_2\,\partial \psi_3}(\Delta\psi_2)(\Delta\psi_3) + O(h^{3+}) \end{aligned}\right\} \qquad (4.51)$$

$O(h^{3+})$ in equation (4.51) is the error term resulting from truncating the equation at order 2. In most open pit mining operations the material properties are often uniform and similar excavation methods are adopted, thus H and θ are usually kept constant. Hence, $\Delta\psi_3 = 0,\ \frac{\partial V'}{\partial \psi_3} = 0, \frac{\partial^2 V'}{\partial \psi_3^2} = 0$. In that case, neglecting the higher derivatives (i.e. considering $O(h^{3+}) = 0$), equation (4.51) simplifies to:

$$\frac{\partial V'}{\partial t} = \begin{cases} \frac{\partial V'}{\partial \psi_{1(t)}}\Delta\psi_1 + \frac{\partial V'}{\partial \psi_{2(t)}}\Delta\psi_2 + \frac{1}{2}\frac{\partial^2 V'}{\partial \psi_{1(t)}^2}(\Delta\psi_1)^2 + \frac{1}{2}\frac{\partial^2 V'}{\partial \psi_{2(t)}^2}(\Delta\psi_2)^2 \\ + \frac{1}{2}\frac{\partial^2 V'}{\partial \psi_1 \psi_2}(\psi_1\psi_2) \end{cases} \qquad (4.52)$$

4.5 Boundary Conditions of Equation (4.52)

The boundary conditions are governed by the initial and final volumes of the excavation, the aerial extent of the orebody, stripping ratio, the ultimate pit depth and bench slope angles, economic conditions (price of mineral on market, initial capital requirements, and operating costs). The initial conditions are governed by (i) the length of the train of conveyor belt wagons of the AFS system; (ii) the radius of the production envelope of the AFS train (R_{AFS}); (iii) the bench height (H); (iv) the bench slope angle (θ); (v) the overall pit slope angle (β); (vi) the volume of the initial cut (V_{01}), and (vii) the lengths of the initial cut along the major and minor axes (a_o, b_o). Economic factors that affect the initial conditions include (i) the commodity prices on the market (P_m); (ii) the price of the mineral used in the economic evaluations of the mine (P_t); (iii) initial capital investments required (ICI); (iv) the operating costs (OC_t) and revenues (REV_t); (v) discount rate (R); (vi) internal rate of return (IRR); (vii) net present value (NPV) of mine at any given time as well as the cut off grade (g_o) and run of mine ore grade (g_t). The initial dimensions of the pit, bench height and slope angles are given by equations (4.53) to (4.56) while the volume of the initial cut for either a circular or elliptical frustum, V_0, is given in equation (4.57). The initial conditions on the economic constraints are given in by:

$$a_o \geq b_o \geq L_{AFS} \tag{4.53}$$

$$a_o \geq b_o \geq R_{AFS} \tag{4.54}$$

$$H > 0 \tag{4.55}$$

$$\theta > \beta > 0 \tag{4.56}$$

$$V_o = \begin{cases} \dfrac{\pi H}{3}\left[3a_o(a_o - k) + k^2\right] & \forall \text{ circular frustum} \\ \\ \dfrac{\pi H}{3}\left[2a_o b_o - k(a_o + b_o - k) + \sqrt{a_o b_o}\{(a_o - k)(b_o - k)\}^{0.5}\right] & \forall \text{ elliptical frustum} \end{cases} \tag{4.57}$$

$$REV_t \geq \begin{cases} OC_t & \forall\, t = 0 \\ \\ CI_t & \forall\, t = 0 \end{cases} \tag{4.58}$$

$$g_t \geq g_o \tag{4.59}$$

$$Q_m > Q_p \tag{4.60}$$

$$R \leq IRR \tag{4.61}$$

$$NPV \geq 0 \tag{4.62}$$

The boundary conditions as given in equations (4.63) to (4.77) are functions of (i) the property boundaries (final dimensions of pit along the major and minor axes in the east-west and north-south directions (L_{PBE} and L_{PBN}); (ii) the maximum lengths and radius of production envelope of the AFS train ($L_{M\text{-}AFS}$ and R_{AFS}); (iii) volume of pit at ultimate pit depth (H_{ult}); (iv) total mineable reserves (R_T); (v) dimensions of AFS equipment; (vi) the stripping ratio (SR) at any time; and (vii) the total capital investment up to time t (CI_t). Equations (4.63) to (4.68) show the boundary conditions relating to the dimensions of the pit and equipment while equation (4.69) shows the boundary conditions of the stripping ratio. Equations (4.70) to (4.77) show the boundary conditions for operating cost (OC_t), revenue (REV_t), volume of final pit (V_F) and total ore reserves within the final pit (R_T), other constraints pertaining to the final pit and equipment limitations.

$$2a_f \leq L_{PBN} \tag{4.63}$$

$$2b_f \leq L_{PBE} \tag{4.64}$$

$$H_{ult} \leq H_d \tag{4.65}$$

$$H_s \geq H \tag{4.66}$$

$$b_f \leq a_f \leq R_{AFS} \tag{4.67}$$

$$b_f \leq a_f \leq L_{M\text{-}AFS} \tag{4.68}$$

$$SR = \begin{cases} f(B, H, z, SF, \theta, \beta, \alpha, g_o, g_t, \rho_O, \rho_W, \rho_{SP}) & \forall \text{ pit technical constraints} \\ f\left(\dfrac{\partial V_i'}{\partial t}\right), REV_t, OC_t, CI_t \ \forall \text{ pit economic constraints} \end{cases} \tag{4.69}$$

$$OC_t = f(CI_t, H, \theta, \beta, SF, z, T, V_W, UC_O, UC_W, UC_{SP}, V_O, V_{SP}, R, Q_m, Q_p) \tag{4.70}$$

$$REV_t = f(g_o, g_t, T, V_O, V_{SP}, P_t, P_m, R, Q_m, Q_p) \tag{4.71}$$

$$R_T = \rho_o V_F = \rho_o \sum_{i=1}^{n} [V_{ij}] \tag{4.73}$$

$$P_m \geq P_t \tag{4.74}$$

$$g_t \geq g_o \tag{4.75}$$

$$\beta \leq \theta \tag{4.76}$$

$$SF \geq 1.0 \tag{4.77}$$

4.6 Kinematics and Dynamics of CyExc CBCS Option

In this section, the kinematics and dynamics of the CycEx CBCS are analyzed using the Automatic Dynamic Analysis of Mechanical Systems (ADAMS) software (Anon., 2003g). The main purpose is to determine the motion, displacement, force, torque and stress-strain on the components of the CycEx CBCS option by simulating the real-time mechanical behaviour. In order to realize physical simulation, the kinematics and dynamic models of the CycEx CBCS option must be built in theory.

The theory of mechanism and machine can be used to do kinematic analysis and synthesis of any mechanical system (Hartenberg and Denavit, 1980; Wilson and Sadler, 1991). The general motion equations of Newton-Euler can be used to describe the dynamic behavior of the mechanical system for various reasons and purposes (Fu et al., 1987). This formulation when applied to a wagon unit of the CycEx CBCS option results in a set of forward and backward recursive equations (Frimpong et al., 2003). The most significant aspect of this formulation is that the computation time of the applied torques can be reduced to an allowable real-time control. In the CycEx CBCS option, the belt conveyor wagons are hooked together to form a train of wagons (see Fig. 3.4). The CycEx CBCS mechanism can be regarded as an assemblage of resistant bodies, connected by movable joints, to form a closed kinematic chain with one link fixed that transforms motions. Therefore, the CycEx CBCS system can be simplified to the kinematic diagram shown in Fig. 4.5 for the purpose of analysis. The mechanism is a six-bar linkage, including a frame (link 0), four belt-wagon units (links1-4), one mobile crusher (link 5) and six kinematic joints (joints 1-6). All of the relative motions of CycEx CBCS mechanism shown in Fig. 4.5 are in parallel planes, so it can be considered as a planar mechanism. The Kutzbach criterion expression can be used to determine the available degrees of freedom of planar mechanism (Paul, 1979).

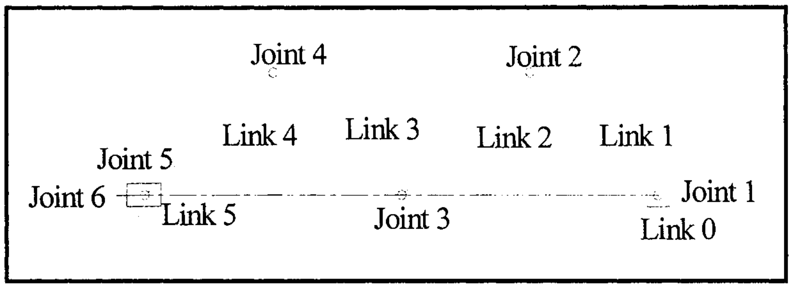


Fig. 4.5 Kinematic Diagram of CycExc CBCS Option
(Source: Frimpong et al., 2003)

4.6.1 Equations of Motion

Fig. 4.6 shows the coordinate system chosen for the CycExc CBCS mechanism according to Denavit and Hartenberg (1980). An orthonormal cartesian coordinate system (x_i, y_i, z_i) is established for each link i at its joint point i where i = 1,2,...,5 (Frimpong et al., 2003). The base coordinate is defined as the 0th coordinate frame (x_0, y_0, z_0), which is also the inertial coordinate frame of the CycEx CBCS system.

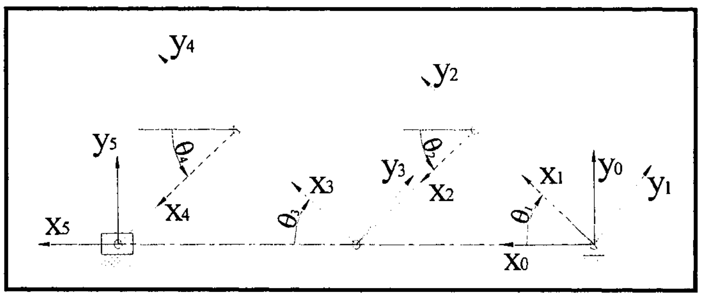


Fig. 4.6 Link Co-ordinate System of the CycEx CBCS System
(Source: Frimpong et al., 2003).

Let ($\vec{x}$, $\vec{y}$, $\vec{z}$) be its unit vector along the principal axes. Thus, for this system, there are six coordinate frames, namely (x_0, y_0, z_0), (x_1, y_1, z_1), ... , (x_5, y_5, z_5). Each coordinate frame is determined and established on the basis of three rules: (i) the z_{i-1} axis lies along the axis of motion of the i^{th} joint; (ii) x_i is normal to the z_{i-1} axis, and pointing away from it; (iii) the y_i axis completes the right-handed coordinate system as required. $\vec{r}_i$ is the position of joint i from the origin of the coordinate system (x_i, y_i, z_i). l_i and θ_i are assumed as the length of link i and the angle value between the x_0 axis and the x_i axis, respectively. θ_i has a positive sign when clockwise and it has a negative sign when counter-clockwise.

Position $\vec{R}_j$ of joint j = 1, 2,..., 5 relative to an inertial origin defined in the local (x_0, y_0, z_0) coordinate system is given by:

$$\vec{R}_j = \sum_{i=1}^{j} \vec{r}_i = \sum_{i=1}^{j} l_i \cos\theta_i \vec{x} + \sum_{i=1}^{j} l_i \sin\theta_i \vec{y} \tag{4.78}$$

Velocity $\vec{v}_j$ and acceleration $\vec{a}_j$ of joint j can be obtained by sequential differentiation of equation (4.78) with respect to time resulting in:

$$\vec{v}_j = \sum_{i=1}^{j} \dot{\vec{r}}_i = -\sum_{i=1}^{j} l_i \dot{\theta}_i \sin\theta_i \vec{x} + \sum_{i=1}^{j} l_i \dot{\theta}_i \cos\theta_i \vec{y} \tag{4.79}$$

$$\vec{a}_j = \sum_{i=1}^{j} \ddot{\vec{r}}_i = -\sum_{i=1}^{j} \left(l_i \ddot{\theta}_i \sin\theta_i + l_i \dot{\theta}^2{}_i \cos\theta_i \right)\vec{x} + \sum_{i=1}^{j} \left(l_i \ddot{\theta}_i \cos\theta_i - l_i \dot{\theta}^2{}_i \sin\theta_i \right)\vec{y} \tag{4.80}$$

In order to compute the forces and torques acting on the joints, it is necessary to calculate the link angular velocity and acceleration, and linear velocity and acceleration iteratively from link 1 out to link 5. Hence, the kinematic equations of all links are (Fu et al., 1979; Frimpong et al., 2003):

For i =1 to 4:

$$\omega_i = \omega_{i-1} + z_{i-1}\dot{q}_i \tag{4.81}$$

$$\dot{\omega}_i = \dot{\omega}_{i-1} + z_{i-1}\ddot{q}_i + \omega_{i-1} \times \left(z_{i-1}\dot{q}_i \right) \tag{4.82}$$

$$\dot{v}_i = \dot{\omega} \times p_i + \omega_i \times \left(\omega_i \times p_i \right) + \dot{v}_{i-1} \tag{4.83}$$

For i = 5:

$$\omega_5 = \omega_4 \tag{4.84}$$

$$\dot{\omega}_5 = \dot{\omega}_4 \tag{4.85}$$

$$v_5 = z_4\ddot{q}_5 + \dot{\omega}_5 \times p_5 + 2\omega_5 \times \left(z_4\dot{q}_5 \right) + \omega_5 \times \left(\omega_5 \times p_5 \right) + \dot{v}_4 \tag{4.86}$$

For i =1 to 5

$$\vec{a}_i = \dot{\omega}_i \times \vec{s}_i + \omega_i \times \left(\omega_i \times \vec{s}_i \right) + \dot{v}_i \tag{4.87}$$

If the velocity and acceleration of the links are known, the forces and torques of interaction and joint actuator torques can be calculated recursively from link 5 back to link 1. The equations are derived by applying Newton-Euler equations to the system resulting in (Fu et al., 1987; Frimpong et al., 2003).

$$F_i = m_i \vec{a}_i \tag{4.88}$$

$$N_i = I_i \dot{\omega}_i + \omega_i \times (I_i \omega_i) \tag{4.89}$$

$$f_i = F_i + f_{i+1} \tag{4.90}$$

$$n_i = n_{i+1} + p_i \times f_{i+1} + (p_i + \vec{s}_i) \times F_i + N_i \tag{4.91}$$

$$\tau_i = \begin{cases} n_i^T z_{i-1} + b_i \dot{q}_i + c_i \sin(\dot{q}_i) \\ f_i^T z_{i-1} + b_i \dot{q}_i + c_i \sin(\dot{q}_i) \end{cases} \tag{4.92}$$

The detailed results of the modeling and simulated displacement, torque and projection angles of the CycEx CBCS option with time are given in Chapters 5 and 6.

4.7 Pit Base Expansion Algorithm

To keep track of the coordinates of the various points on the pit as it expands, it is often necessary to use a central point at the base of the lowest bench as the point of reference (benchmark). The following section details how the coordinates of the points within the pit may be obtained using a benchmark located at the pit base. Open pit geometric modeling is a function of mainly the pit base and bench dimensions and their expansions through time and space. The pit base can be represented by a set of X, Y, Z coordinates. Each node point is a base for the calculation of toe and crest coordinates on each bench with pit expansion (Erarslan, 2002).

The determination of the pit base depends on the geological structure of the orebody, grade distribution, mining costs and prices of minerals/metals on the market. The pit base configuration usually encloses the orebody as the pit expands both laterally and longitudinally. Mathematically, it can be stated that the pit base comprises I data pairs each of which is a function of x (east), y (north) and z (elevation) coordinates as given by:

$$P_i = f(x,y,z) \tag{4.93}$$

The east, north and elevation coordinates of each node on the polygon can be represented by x(i), y(i) and z(i) respectively. The evolution of the polygon is directly proportional with the number of node points i(i = 1, 2, ..., I) on the polygon. The center point of the polygon C (x, y, z) = {C_x, C_y, C_z} can be used to calculate the coordinates of toe and crest and also determine the direction of expansion. These coordinates are given by Erarslan (2002):

$$C_x = \sum\left(\frac{x_i}{I}\right) \tag{4.94}$$

$$C_y = \sum\left(\frac{y_i}{I}\right) \tag{4.95}$$

$$C_y = \sum\left(\frac{z_i}{I}\right) \tag{4.96}$$

The angle between each node *i* and center *C* is illustrated in Fig. 4.7.and given by:

$$a_i = \arctan\left[\frac{|y_i\ C_y|}{|x_i - C_x|}\right] \tag{4.97}$$

The ratio of the absolute difference in vertical axis to the absolute difference in horizontal axis, a_i, determines the angle between the center and the node. This ratio can be used to calculate the toe and crest coordinates. The bench slope width after projecting the length of slope to plan view (horizontal plane) as shown in Fig. 4.8 is given by:

$$k = H/\tan\theta \tag{4.98}$$

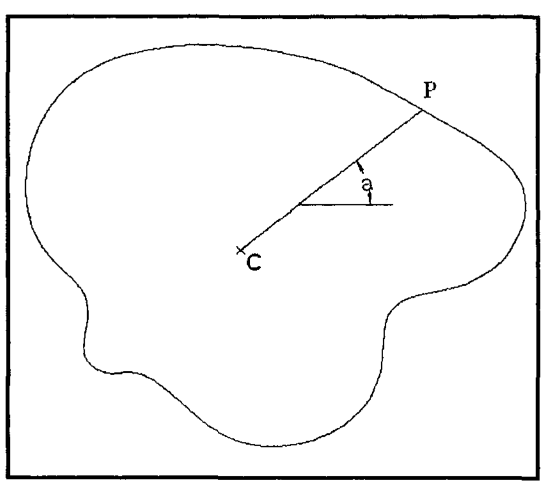


Fig. 4.7 Angle between Center of Pit and Node P

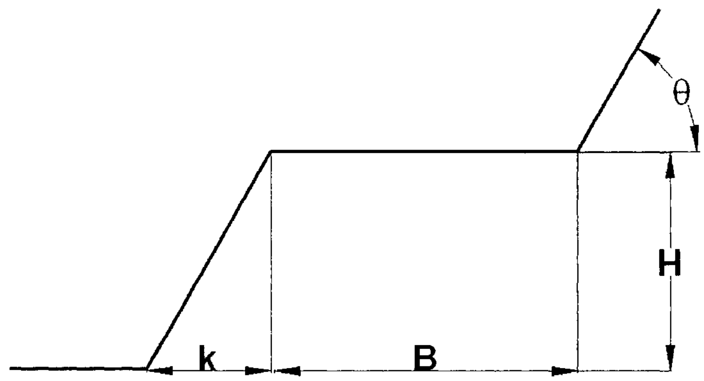


Fig. 4.8 Bench Slope and Width

Bench width B is also required to expand the pit. A regular expansion can be provided such that each P_i node is the toe of first bench b_n, n = 1, 2, ..., N, where n is bench number and N is the maximum number of benches. Toe and crest coordinates of each node P_i at bench level b_n can be represented with $\varphi_{i,n}$ and $\delta_{i,n}$ respectively.

$$\varphi_{i,n}(x, y, z, a_i) = \{\varphi_{i,n}^x, \varphi_{i,n}^y, \varphi_{i,n}^z\} \tag{4.99}$$

$$\delta_{i,n}(x, y, z, a_i) = \{\delta_{i,n}^x, \delta_{i,n}^y, \delta_{i,n}^z\} \tag{4.100}$$

Since the first (x_1, y_1, z_1) pairs are the toe of the first bench, initial $\varphi_{i,n}$ values and $\delta_{i,n}$ values can be calculated from:

$$\varphi_{1,1}^x = x_1 \tag{4.101}$$

$$\varphi_{1,1}^y = y_1 \tag{4.102}$$

$$\varphi_{1,1}^z = z_1 \tag{4.103}$$

$$\delta_{1,1}^x = x_1 + k \times \cos(a_i) \tag{4.104}$$

$$\delta_{1,1}^y = y_1 + k \times \sin(a_1) \tag{4.105}$$

$$\delta_{1,1}^z = z_1 + H \tag{4.106}$$

The terms $\varphi_{i,n}$ and $\delta_{i,n}$ are functions of (x, y, z) data set, bench slope angle θ, bench height H, bench width B, and center of polygon. Taking the east (x-axis) direction as the reference direction and moving in the counterclockwise direction, the coordinates of the direction of expansion D_i for the i^{th} node in the four quadrants are (Erarslan, 2002):

$$D_i = I \text{ for } y_i \quad C_y > 0 \qquad \text{and } x_i \quad C_x > 0 \tag{4.107}$$

$D_i = 2$ for y_i $C_y > 0$ and x_i $C_x < 0$ (4.108)

$D_i = 3$ for y_i $C_y < 0$ and x_i $C_x < 0$ (4.109)

$D_i = 4$ for y_i $C_y < 0$ and x_i $C_x > 0$ (4.110)

The corresponding toe ($\varphi_{i,n}$) and crest ($\delta_{i,n}$) coordinates of each base node *i*, after expanding through benches *j*, can geometrically be calculated using the algorithms illustrated by:

for $D_i = 1$:

$$\delta_{i,n}^x = \delta_{i,n-l}^x + k \times \cos(a_i) \tag{4.111}$$

$$\delta_{i,n}^y = \delta_{i,n-l}^y + k \times \sin(a_i) \tag{4.112}$$

$$\varphi_{i,n}^x = \varphi_{i,n-1}^x + (k + B) \times \cos(a_i) \tag{4.113}$$

$$\varphi_{i,n}^y = \varphi_{i,n-1}^y + (k + B) \times \sin(a_i) \tag{4.114}$$

for $D_i = 2$:

$$\delta_{i,n}^x = \delta_{i,n-1}^x - k \times \cos(a_i) \tag{4.115}$$

$$\varphi_{i,n}^x = \varphi_{i,n-l}^x - (k + B) \times \cos(a_i) \tag{4.116}$$

$\delta_{i,n}^y$ and $\varphi_{i,n}^y$ are calculated using Equations (4.115) and (4.117) respectively.

For $D_i = 3$:

$$\delta_{i,n}^y = \delta_{i,n-l}^y - k \times \sin(a_i) \tag{4.117}$$

$$\varphi_{i,n}^y = \varphi_{i,n-1}^y - (k + B) \times \sin(a_i) \tag{4.118}$$

$\delta_{i,n}^x$ is calculated using Equation (4.112) while $\varphi_{i,n}^x$ is obtained from Equation (4.117). For $D_i = 4$, $\delta_{i,n}^x$ is calculated using Equation (4.112) and $\delta_{i,n}^y$ is obtained from Equation (4.99). To calculate $\varphi_{i,n}^x$ and $\varphi_{i,n}^y$, Equations (4.114) and (4.119) can be used. Bench elevations (z coordinates) for toe and crest and for all directions are identical and these can be estimated by:

$$\delta_{i,n}^z = \varphi_{i,n}^z = \varphi_{i,n-1}^z + H \tag{4.119}$$

4.8 Parameterization of AFS Machinery and Layout Configuration

In this section, the ground space requirements for housing the CMS and CycEx CBCS options are outlined and calculated. This is necessary to determine the initial conditions before each mining option can be economically employed to mine the oil sands.

4.8.1 Option 1: Current Mining System (CMS)

The current mining system employs shovel-truck system in mining the ore and waste from the pit. The volume of material from the initial cut, V_{01}, is given in equation (4.2). From Section 4.1.2, the cumulative volume of materials excavated from a frustum with circular cross-section and *n* benches is given by equation (4.18). Also, the cumulative incremental volume of materials excavated from *n* benches is given by equation (4.22).

The daily production capacity of the CMS option depends on the fleet size of shovels and trucks, the carrying capacities of the trucks, the fleet capacity, cycle times of trucks and loaders, the nature and grades of haul roads, loose densities of materials to be excavated, loaded and hauled to the various destinations, the availabilities and utilizations of the equipment. The production capacity of the system can be modeled as in:

$$\phi(V_T, F_S, F_T, F_C, C_T, A, U, \gamma_{ML}) \qquad (4.120)$$

The volume of the pit layout resulting from the lateral and longitudinal expansions due to incremental pushbacks, V', at any given time is a function of the changes in the radii of the ellipsoid and the geometry of the layout and is given by equation (4.50). Finding the partial differential of V' with respect to t for the CMS option, a relation of the form expressed by equation (4.51) was obtained for the lateral and longitudinal changes in the pit geometrical layout with time. The partial differential equations for $\partial V'/\partial \psi_{1(t)}$ and $\partial V'/\partial \psi_{2(t)}$ express the sensitivity of the rate of change of volume of the pit with respect to marginal changes (pushbacks) in concurrent active faces in the north-south and east-west directions. The second partial derivative expressions $\frac{\partial^2 V'}{\partial \psi_{1(t)}^2}$, $\frac{\partial^2 V'}{\partial \psi_{2(t)}^2}$ and $\frac{\partial^2 V'}{\partial \psi_{1(t)} \partial \psi_{2(t)}}$ show the variances of the functions (contribution of the marginal changes in the variables in the active concurrent faces to the overall volume of the pit in the north-south and east-west directions).

The initial conditions of the CMS option are functions of the bench width, bench height, dimensions of the major and minor axes of the initial cut, and bench slope angle, overall pit slope angle and the reach of the shovel in this system. They also depend on the initial capital investment, operating costs and revenues, cut off grade, run of mine ore grade, the installed mine and processing plant capacities, the expected rate of return on investment, and the NPV. Equations (4.122) to (4.127) express the initial conditions imposed by the dimensions of the initial cut, bench height, slope angles, shovel reach and dumping radius on the model, volume of the initial cut and the economic conditions imposed by the CMS option respectively.

$$a_o \geq B \tag{4.121}$$

$$b_o \geq RLF \tag{4.122}$$

$$H_s \geq H \geq 0 \tag{4.123}$$

$$\theta \geq \beta \geq 0 \tag{4.124}$$

$$SF \geq 1.5 \tag{4.125}$$

$$B = b_{rw} + TW + 2 \times DR + RLF + CL \tag{4.126}$$

The initial conditions for the initial cut, economic constraints (operating costs, revenues, initial capital investment, installed mine and processing plant capacities, cut off grade and run of mine ore grade, discount rate and IRR) for the CMS option are the same as those in equations (4.53) to (4.62) in Section 4.4.

The boundary conditions of the CMS are governed principally by the final dimensions of the pit which in turn are affected by the aerial extent and depth of deposit below surface, bench widths and heights, dimensions of the shovel and dump trucks used in pit. Other factors are stripping ratios at any given time, economic conditions such as unit operating costs in ore mining and waste stripping, revenues, capital investment, rate of return on investment of the mine's operations, commodity prices and grades of the ore or quality of minerals. Equations (4.63) to (4.77) express some of the boundary conditions of the CMS option. To ensure that there is maximum safety to both men and equipment working in the pits, the safety factor (SF) is set equal to or greater than 1.5 (i.e. SF $\geq$ 1.5) in the boundary conditions of the CMS option. The maximum pit dimensions are also affected by the property boundaries in the north-south and east-west directions.

4.8.1.1 Numerical Example at Syncrude – CMS Option

In the CMS option, the dimensions of the initial cut are functions of the sizes of the trucks and shovels used in the pit and the initial cut must accommodate the equipment dimensions. Using the dimensions of mining equipment in Appendix A, the bench width required for loading trucks in a double back-up system as employed at Syncrude is given by equation (4.128). Table 4.1 summarizes the various parameters of the pit employing the CMS option using equations (4.53) to (4.62) and valued from Appendix A.

$$B = b_{cw} + TW + 2 \times DR + RLF + CL \tag{4.127}$$

Table 4.1 Summary of Initial Conditions of the Pit using the CMS Option

Parameter	Length
Safety bem width (b_{rw})	3.0 m
Width of largest truck (TW)	10.0 m
Radius of level floor (RLF)	25.35 m
Clearance between trucks and berm (CL)	5.0 m
Bench height, H	13 m
k (H/tan50^o)	10.91 m
B	65 m
$a_o \geq B$	90.91 m
$a_1 = a_o - k$	80 m
H_s	20.1 m
$b_1 = b_o - k$	19.09 m
$b_o \geq RLF$	$\geq$ 30 m
$A_o = \pi a_o b_o = \pi \times 90.91 \times 40.91$	11,683.99 m^2
$A_1 = \pi a_1 b_1 = \pi \times 80 \times 30$	7,539.82 m^2
V_{01} (elliptical pit)	123,975.36 m^3
V_{01} (circular pit)	298,646.67 m^3
Run-of mine ore grade, g_t	$\geq$ 10 – 18%
Cut-off grade, g_o	$\geq$ 10%
Safety Factor	$\geq$ 1.5

4.8.1.2 Boundary Conditions for CMS Option

Similarly, Table 4.2 summarizes the calculated values of the boundary conditions using the dimensions of the North Mine of 6 km × 10 km × 76 m deep (thickness of oil sands deposit $\approx$ 40 m).

Table 4.2 Summary of Boundary Conditions of Pit using the CMS Option

Parameter	Value
L_{PBN}	10 km
$a_f \le 0.5 \times L_{PBN}$	5,000 m
L_{PBE}	6 km
$b_f \le 0.5 \times L_{PBE}$	3,000 m
Safety Factor, SF	≥ 1.5

Table 4.3 summarizes the volume of materials excavated from Benches 1 to 3 using the CMS option for both circular and elliptical pits configurations. When any of the mining options is employed, the pit limits will be reached when the limits of the pit along the minor axis direction are reached (i.e. when b_f = 3000 m). For the circular pit configuration, the pit limits are reached when the limits of the pit along the minor axis direction are reached (hence $a_f = b_f$ = 3000 m).

Table 4.3 Volume of Materials Excavated from Pit on Benches 1 to 3 with CMS

Pit Shape	**Bench #1**		**Bench #2**		**Bench #3**	
	Circular	Elliptical	Circular	Elliptical	Circular	Elliptical
a_0, m	90.91	90.91	90.91	90.91	90.91	90.91
b_0, m	0	40.91	0	40.91	0	40.91
a_f, m	3000	3050	2909.09	2959.09	2818.18	2868.18
b_f, m	3000	3000	2909.09	2909.09	2818.18	2818.18
Initial Volume ($\times 10^5 m^3$)	2.99	1.24	2.99	1.24	2.99	1.24
Final Volume ($\times 10^8 m^3$)	3.66	3.72	3.44	3.50	3.23	3.29
Volume Excavated ($\times 10^8 m^3$)	3.66	3.72	3.44	3.50	3.23	3.29
Time to Excavate (days)	2933.05	2983.46	2757.52	2806.45	2587.41	2634.85
Years to Excavate	8.04	8.17	7.55	7.69	7.09	7.22

In the same way, the limits of pits with elliptical configuration are reached when the minor axis of the ellipse equals 3000 m. It is noted that generally, smaller volumes of materials are excavated from the circular shaped pits than from the elliptical ones because the boundary conditions along the minor axis of the pit are reached before those on the major axis. More materials can be extracted from the elliptical pit along the major axis after the pit limits along the minor axis have been reached than with circular pit sections. At the scheduled production rate of 262,000 tonnes/day, the ore within Benches 1 to 3 will be completely mined out between 7.09 and 8.04 years with circular pit configurations. With elliptical pit configurations, the ore in the three benches will be excavated between 7.22 and 8.17 years.

4.8.2 Option 1: Cyclic Excavator Conveyor Belt Control System (CycEx CBCS)

In this section the dimensions of the pit that will accommodate the equipment under the CycEx CBCS system are determined. Due to the fact that Option 1 contains a series of belt conveyor wagons, the equipment layout is originally set to be parallel to the major axis of the pit in the case of the elliptical pit configuration.

4.8.2.1 Material Flow Model of CycEx CBCS

The material flow dynamics of the Cyclic Excavator Conveyor Belt Control System (CycEx CBCS) option closely resembles that of the CMS option except that the bench layout is influenced more by the dimensions of the AFS train being used in this option. As in the CMS option, shovels are used for the extraction of the ore at the faces in this option. Thus the change in volume of material flow between the i^{th} and j^{th} pushbacks is as given by equation (4.18) for a circular frustum and by equation (4.32) for an elliptical shaped frustum.

The daily production capacity of the CycEx CBCS option depends on the fleet size of shovels cumulative production capacities of shovels engaged in the multi-bench, multi-face operations, C_s, the capacities of the conveyor belt wagons and the hydrotransport plant, the rheological properties of the slurry, the availabilities and utilizations of the shovels and slurrification facility. The production capacity can be modeled stochastically as:

$$P_{Cyc} = \phi(C_s, C_{cov}, C_{hyd}, A_s, Us, A_{cf}, U_{cf}, \gamma_s) \qquad (4.128)$$

The initial conditions for the CycEx CBCS option are determined mainly by the dimensions of the length, horizontal and vertical movements of the AFS train. To house the shovel, the train of conveyor belt wagons and slurrification units in the pit will require a much larger initial cut than pertained in the CMS option. The volume of initial cut and constraints on costs and revenues are the same as those in equations (4.57) and (4.62) respectively. Equations (4.130) to (4.135) show the initial conditions for the initial pit and equipment dimensions in the CycEx CBCS option.

$$a_o \geq b_o \geq L_{AFS} + L_{sh} \qquad (4.129)$$

$$b_o \geq R_{AFS} \qquad (4.130)$$

$$H_s \geq H \geq 0 \qquad (4.131)$$

$$\theta \geq \beta \geq 0 \quad (4.132)$$

$$SF \geq 0 \quad (4.133)$$

$$L_{AFS} = L_{cw} + L_{hc} + L_{cf} \quad (4.134)$$

The boundary conditions of the CycEx CBCS are functions of the property boundaries, stripping ratio, operating costs (unit costs of ore mining, waste stripping and treatment of ore) at any given time, cut off grade and run of mine ore grade, economic constraints on the operations (commodity prices, rate of return and internal rate of return on investment, capital investments), installed mine and processing plant capacities, characteristics of the orebody (ore grade, aerial extent and depth below surface). Equations (4.136) to (4.143) show the boundary conditions imposed on the CycEx CBCS option by the property boundaries, orebody characteristics and equipment limitations within the final pit. The boundary conditions for stripping ratio, operating cost, revenue, volume and total reserves are the same as in equations (4.69) to (4.77).

$$2a_f \leq L_{PBN} \quad (4.135)$$

$$2b_f \leq L_{PBE} \quad (4.136)$$

$$a_f \leq R_{AFS} \quad (4.137)$$

$$b_f \leq R_{AFS} \quad (4.138)$$

$$a_f \geq L_{M\text{-}AFS} \quad (4.139)$$

$$b_f \geq L_{M\text{-}AFS} \quad (4.140)$$

$$H_{ult} \leq H_d \quad (4.141)$$

$$SF \geq 1.2 \quad (4.142)$$

The main differences in the geometrical layouts of the CMS and the CycEx CBCS options are that the movements between the loader and the AFS train at the bench faces during production operations are required to be synchronized with the CycEx CBCS. Due to the high flexibility and mobility of the dump truck units in the CMS option, the relative lateral and vertical movements of the loader have very little impact on the productivity of the fleet of trucks. The final pit dimensions affect the maximum lengths of the AFS trains. As a result of the fact that it is only the loaders that are located at the face together with portions of the AFS train, the safety requirements of the pit slopes are not as rigid as in the CMS option. Hence the minimum safety factor may be set to a lower value of 1.2 in the CycEx CBCS option than in the CMS option where SF ≥ 1.5.

4.8.2.2 Numerical Example at Syncrude – CycEx CBCS Option

Table 4.4 summarizes the initial conditions for the CycEx CBCS option as defined in equations (4.130) to (4.135) together with data from Appendices A and B.

Table 4.4 Summary of Initial Conditions of Pit using the CycEx CBCS option

Parameter	Value
Radius of pit along the major axis, a_o	164 m
Radius of pit along the major axis, b_o	104 m
Length of slurrification unit, L_{cf}	54 m
Length of belt conveyor wagon, L_{cb}	20 m
Length of hopper-crusher unit, L_{hc}	54 m
Length of shovel, L_{sh}	30 m
Safety Factor, SF	≥ 1.2
Bench height, H	13 m
Bench slope angle, θ	50°

Figs. 4.9 and 4.10 show the initial layouts of the AFS mining equipment at the start of the CycEx CBCS option in a circular and elliptical pits respectively. Fig. 4.10 shows that in the elliptical pit, the equipment are laid out along the major axis of the elliptical pit.

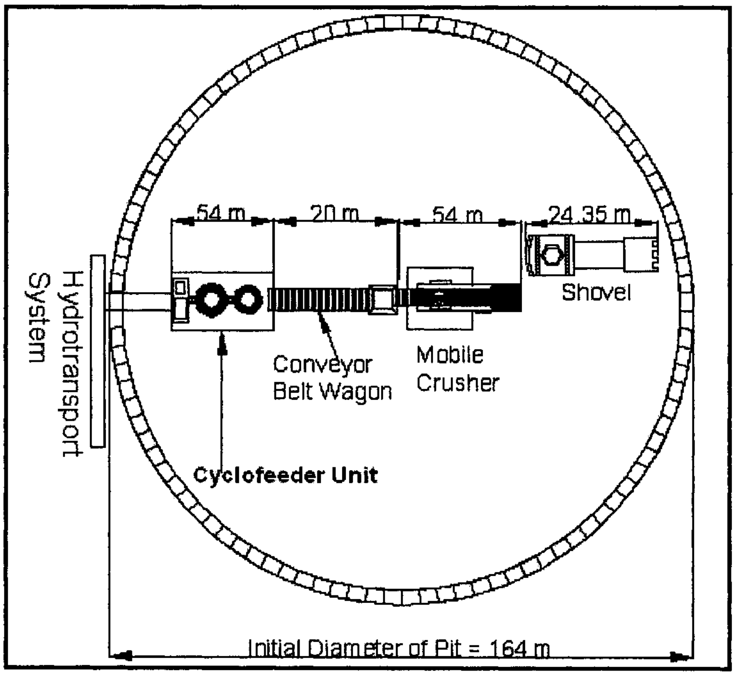


Fig. 4.9 Initial AFS Equipment Layout in a Circular Pit

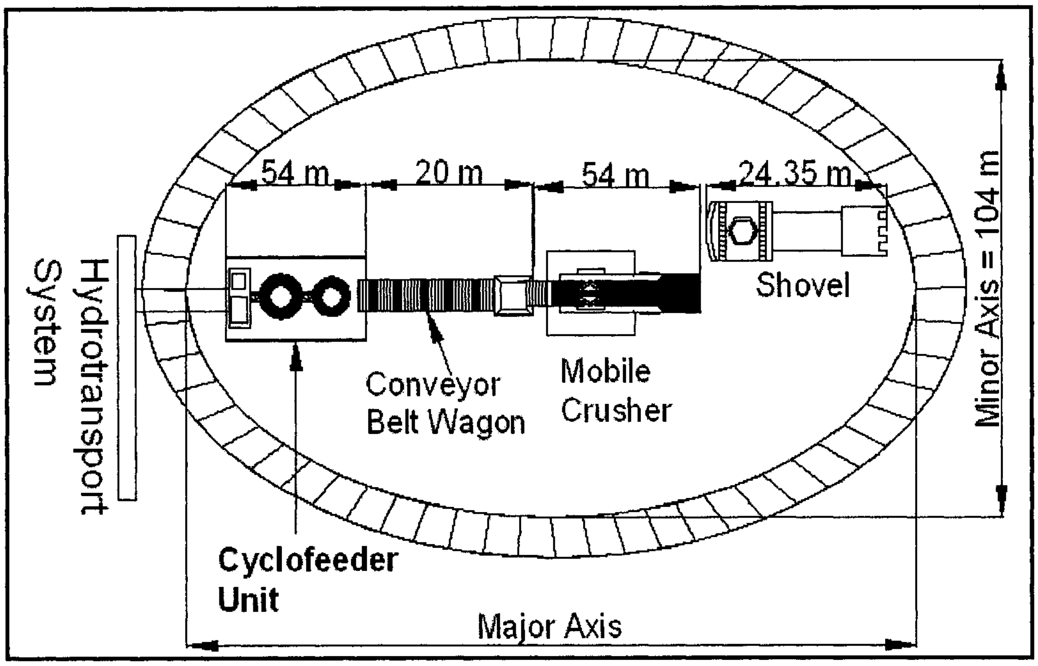


Fig. 4.10 Initial AFS Equipment Layout in an Elliptical Pit

4.8.2.3 Boundary Conditions for CycEx CBCS Option

Tables 4.5 and 4.6 respectively summarize the boundary condition and volume of materials excavated from Benches 1 to 3 using the CycEx CBCS option from circular and elliptical pits for the North Mine of Syncrude.

It is noted that larger initial pit volumes are required to be excavated before the CycEx CBCS option can be employed than for the CMS option due to the lengths of the belt conveyor wagons, mobile crusher and slurrification units. At the scheduled production rate of 262,000 tonnes/day, the ore within Benches 1 to 3 will be completely mined out between 6.25 and 8.02 years with circular pit configurations. With elliptical pit configurations, the ore in the three benches will be excavated between 6.40 and 8.19 years.

Table 4.5 Summary of Boundary Conditions of Pit using the CycEx CBCS Option

Parameter	Value
$L_{AFS} = L_{hc} + L_{cf} + L_{cw}$	134 m
Length of shovel, L_{sh}	30 m
a_f	5,000 m
b_f	3,000 m
H_{ult}	39
Safety Factor, SF	≥ 1.2

Table 4.6 Volume of Materials Excavated from Pit on Benches 1 to 3 with CycEx CBCS

Pit Shape	Bench #1		Bench #2		Bench #3	
	Circular	Elliptical	Circular	Elliptical	Circular	Elliptical
a_0, m	174.91	174.91	174.91	174.91	174.91	174.91
b_0, m	0	114.91	0	114.91	0	114.91
a_f, m	3000	3060	2825.09	2885.09	2650.18	2710.18
b_f, m	3000	3000	2825.09	2825.09	2650.18	2650.18
Initial Volume ($\times 10^6 m^3$)	1.17	0.76	1.17	076	1.17	076
Final Volume ($\times 10^8 m^3$)	3.66	3.74	3.25	3.32	2.86	2.92
Volume Excavated ($\times 10^8 m^3$)	3.65	3.73	3.24	3.31	2.84	2.91
Time to Excavate (days)	2926.04	2988.18	2593.14	2651.85	2280.27	2335.54
Years to Excavate	8.02	8.19	7.10	7.27	6.25	6.40

4.9 Summary

In this Chapter, the mathematical models of the geometrical changes in the volumes and dimensions of pits with circular and elliptical frustums were discussed. Various mathematical models and equations have been derived for modeling and calculating the changes in the volumes and dimensions of pits using ordinary mathematical and partial differential equations. The kinematics and dynamics, pit base expansion algorithms as well as the parameterization of the CMS and CycEx CBCS were also outlined.

It is noted from the sample calculations that the volume of materials excavated from pits with elliptical configurations are generally larger than those of circular configurations irrespective of whether the CMS or the CycEx CBCS option is being employed. Accordingly, it takes slightly longer times to mine the materials in the elliptical pits than those with circular pit configurations. Also with the CycEx CBCS option, larger initial pits need to be excavated before the option can be employed than with the CMS option. This may lead to delays in the application of the CycEx CBCS option. These delays could adversely impact the economics of the mine.

In the next chapter, the details on the simulation models of the CMS and conceptual AFS option using software like Visual SLAM with AweSim, Simphony, Automatic Dynamic Analysis of Mechanical Systems (ADAMS) and Matlab are presented. The models, network diagrams, computer algorithms within the software will be applied to Syncrude's North Mine which measures 6 km × 10 km × 76 m deep.

CHAPTER 5

COMPUTER MODELING AND VERIFICATION OF MINING OPTIONS

5.1 Introduction

This chapter presents the details on the computer modeling and verification of the existing mining method at Syncrude and one of the conceptual AFS methods chosen for further analysis. The detailed network diagrams, algorithms of the computer programs used to model, verify and run the simulations in this work are given. The models in this chapter are applied to Syncrude's North Mine with three 13 m high operating benches and the final dimensions on bench #1 being 10000 m $\times$ 6000 m $\times$ 13 m. Bench slopes are 50° while the daily ore production target is 262,000 tonnes. The bank density of the oil sands averages 2.1 t/m^3 while the bulk density is 1.8 t/m^3. The results of sample calculations are also given to illustrate the procedure for applying the models in practice.

5.2 Computer Modeling and Simulation Techniques

Computer modeling and simulation techniques are increasingly being used to model, experiment and test physical systems on a computer without necessarily interfering with the actual physical system. Computer simulation, in its broadest sense, is the process of designing a mathematical logical model of a real system and experimenting with this model on computer (Pritsker et al., 1997). Simulation can be defined as a method in which real world data essential to the study of a complex problem is processed through a model that represents the operating environment. Simulation therefore comprises model building process, as well as, the design and implementation of an appropriate experiment involving that model. This allows inferences to be made about systems (i) without building them if they are only proposed systems; (ii) without disturbing them if they are operating systems that are costly or unsafe to experiment with; and (iii) without destroying them if the object of an experiment is to determine their limits of stress (Pritsker et al., 1997).

As a result of these, simulation models can be used for design, procedural analysis and performance assessment. The simulation approach is applicable to systems where classical analytical techniques are not feasible. There are no precise guidelines as to how to simulate a system. Therefore, the attributes of general applicability and the lack of guidelines have not only given simulation great flexibility and wide use, but a possibility for misapplication and abuse. Consequently, there is a need for model builders to ensure that the simulation model is a factual system and a valid representation of reality. The widespread use of simulation as an analytical tool has led to the development of a number of languages specifically designed for simulation. These languages provide specific concepts and statements for representing the state of a system at a point in time and moving the system from state to state (Pritsker et al., 1997; Baka et al., 1999).

The main objective of this work is the development of a new mining method associated with the AFS technology. The work aims at studying the evolution of a pit geometry that will allow for the timely extraction, transportation and slurrification of oil sands by slurrying units located within the pit. It is also aimed at simulating the production schedules and sequence in continuous time paradigm. It also makes comparative analysis of the production-economic functions of the AFS technology versus the conventional shovel-truck mining system. To achieve the objectives of the study, there is a need to calculate the volumes of materials that are excavated at various stages in the expansion of the pit geometry. As well, it is necessary to determine how the excavated materials are handled and processed with time in multi-bench and multi-face operations. The success of both mining systems depend on the interaction of the various equipment units during operations. Accordingly, the torsional stresses, torques and bending moments that the equipment components have to withstand within the pits have to be assessed. As the CycEx CBCS option involves the use of hoppers-crushers, trains of belt conveyor wagons or flexible pipelines together with slurrification units, it is necessary to evaluate the torsional stresses, torques and bending moments on the equipment.

Fig. 5.1 is a flow chart of the different types of software packages and areas where they were used in this study. The computer modeling and simulation techniques are broadly classified into two groups in this work – dynamics of excavated materials, and simulation of mining systems and equipment. The dynamics of excavated materials comprise geometric mine design and bench characteristics. This involves the calculation of the geometric shapes of the pit at various stages and the volumes of materials excavated from the different pit shapes, and the differential changes in pit layout and volume using partial differential equations. The simulation of mining systems and equipment, economic and risk characterization involved stochastic variables. Accordingly, the Matlab software was used in the calculation of the geometric volumes of materials and dynamics of the pit shape with time, and the pit layout and shape using partial differential equations. Matlab was found suitable for these calculations due to the ease of learning and use in programming various mathematical equations, its ability to handle the recursive nature of calculations using M.files and partial differential equations components/functions in the software.

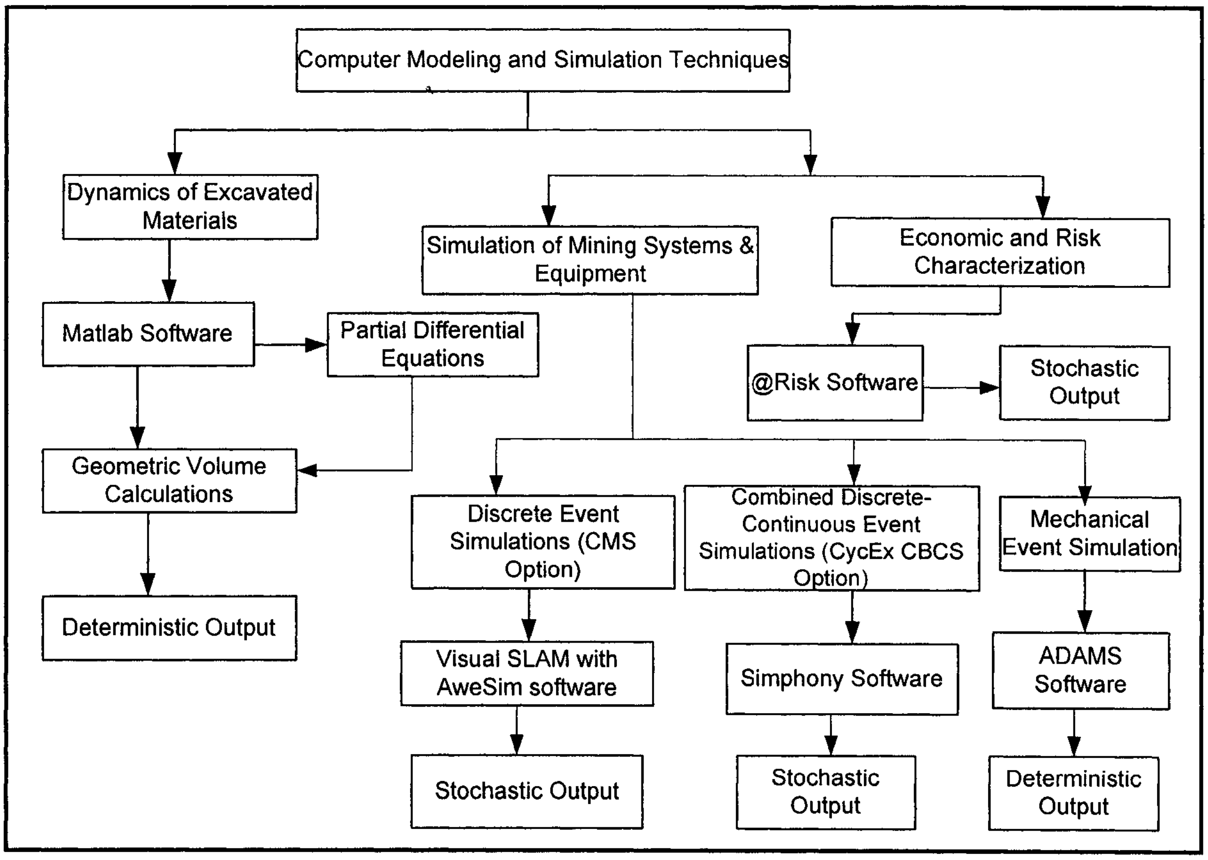


Fig. 5.1 Flow Chart of the different types of software used in study

Mining operations involve discrete, continuous, as well as, combined events whose input values are variable (stochastic) in nature. The current mining system that is being used at Syncrude involves the shovel-truck system with short hauls. The operations of the system were modeled and simulated by Visual SLAM with AweSim software (Pritsker et al., 1997) because it was a familiar software and due to its versatility in modeling and simulation discrete events.

The new AFS mining option (i.e. the CycEx CBCS option) involves a combination of cyclic excavator continuous materials handling systems. Cyclic shovels feed materials into hoppers, apron feeders feed the materials into double roll crushers which in turn feed the crushed materials on to a train of belt conveyor wagons. The material from the belt conveyor wagons is fed to a slurrification facility where it is slurried before being pumped into the main hydrotransport system. The CycEx CBCS option therefore combines both discrete and continuous materials handling systems which are best modeled with the Simphony software (Anon., 2000a). Simphony software was found to be extremely suitable for modeling and simulating the continuous operations in the CycEx CBCS option because the user inserts required are easy and simple to program in Visual Basic. The mechanical event simulation aspect of this work is handled using the ADAMS software (Anon., 2003g). This enables the modeling and simulation of the interactions between the various equipment units during operations, and the torsional stresses, torques and bending moments that the equipment components have to withstand within the pits. ADAMS software was used here because it was the only software available for modeling and simulation of the interactions between the mechanical components in the CycEx CBCS option. @Risk software was used in modeling and simulating the economic and risk characterization of the CycEx CBCS option because it is simple and easy to use, and also because it is one of the best software available on the market for economic and risk analysis.

5.3 Dynamics of Excavated Materials

In this section, the dynamic modeling of the pit design and the volumes of materials excavated from pits with different shapes is done using the Matlab software. In addition, the progressive change in the volumes of circular and elliptical pits are modeled using parabolic partial differential equations. Sample verification results are given in each situation to illustrate the points raised in the ensuing sections.

5.3.1 Geometric Volume of Materials Excavated from a Circular Pit

In open pit mining, it is necessary to know the volume of materials excavated at any given time from any bench face in order to schedule the equipment and the sequence of extraction of the blocks to meet the daily production target and for grade control purposes. Fig. 5.2 is the Matlab M-file for calculating the volume extracted with each incremental pusback in a pit having a circular frustum based on equation (4.8). The length of the incremental pushback, Δx, is variable. The Matlab algorithm in Fig. 5.2 allows the user to input the required values of the bench number, initial and final bench widths, length of incremental pushback (Δx), bench slope angle and bench height.

5.3.1.1 Verification of Algorithm for Circular Pit

The Matlab algorithm for the circular pit in Fig. 5.2 was verified by conducting sample calculations on a pit with initial and final bench widths of 80 m and 200 m respectively. The length of each incremental pushback, Δx, is taken as 10 m. The following input data was provided at the prompting of the program:

Enter bench number: 1
Enter Initial bench width (m): 80
Enter final bench width (m): 200
Enter bench height (m): 13
Enter incremental pushback (Δx, m): 10
Enter working bench width (m): 80
Enter bench slope (degrees): 50

Table 5.1 summarizes the results of the sample calculations. The column labeled "Expansion" in Table 5.1 refers to the total increase in the dimension of the base (bottom) of the frustum after a number of incremental pushbacks are taken.

Table 5.1 Summary of Simulated Output from the Algorithm on a Circular Pit

Increment No.	Expansion (m)	Volume Extracted (m^3)	Time to Extract (days)
1	10	64974.17	0.52
2	20	73142.31	0.59
3	30	81310.45	0.65
4	40	89478.60	0.72
5	50	97646.74	0.78
6	60	105814.88	0.85
7	70	113983.02	0.91
8	80	122151.16	0.98
9	90	130319.30	1.04
10	100	138487.44	1.11
11	110	146655.58	1.18
12	120	154823.72	1.24

```
% Matlab algorithm to calculate circular pit volume by specifying
% initial and final bench numbers, bench height and slope angle
%  and the size of incremental pushback
%
Bench_No = input('Enter bench number:');
Init_bench = input('Enter Initial bench width (m):');
Final_bench = input('Enter final bench width (m):');
Height = input('Enter Bench height (m):');
Incr_pushback = input('Enter incremental pushback (m):');
Bench_width = input('Enter minimum working bench width (m):');
theta = input('Enter bench slope (degrees):');
c = tan(theta*pi/180);
k = Height/c;
C = 262000/2.1;
Be = 0:Incr_pushback:(Final_bench - Init_bench + Incr_pushback);
N = round((Final_bench - Init_bench)/Incr_pushback);
%
% Specify the number of incremental pushbacks to be taken
% Calculate the volume excavated and tabulate the increment number
% Distance from initial toe of bench and Volume of materials excavated
% and time to extract volume in days. List values of n, Ex, V and T
% alongside each other in columns
%
n = 1:N;
Ex = n*Incr_pushback;
s = 2*Init_bench + (2*n -1)*Incr_pushback - k;
V = pi*Height*Incr_pushback*s;
T = V/C;
Values = [n' Ex' V' T'];
fprintf('No.  Expansion  Volume  Time')
[Values]
```

Fig. 5.2 Matlab M-File for Volume of Materials within a Circular Pit

5.3.2 Geometric Volume of Materials Excavated from an Elliptical Pit

Fig. 5.3 is the Matlab algorithm for calculating the volume of materials excavated from each bench in an elliptical pit using equation (4.32).

5.3.2.1 Verification of Algorithm for Elliptical Pit

The Matlab algorithm in Fig. 5.3 for the elliptical pit was verified by conducting sample calculations on a pit with initial pit dimensions of 160 m $\times$ 60 m and final pit dimensions of 400 m $\times$ 300 m. The following input data was provided at the prompting of the program:

```
% Matlab algorithm to calculate elliptical pit volume by specifying initial and final
% bench numbers, bench height and slope angle, Incremental pushback value;
% C is daily production in cubic meters
%
Bench_No = input('Enter bench number:');
Init_bench = input('Enter Initial bench width along major axis, ao, (m):');
Init_bench2 = input('Enter Initial bench width along minor axis, bo, (m):');
Final_bench = input('Enter final bench width along major axis, af,(m):');
Final_bench2 = input('Enter final bench width along minor axis, bf,(m):');
Height = input('Enter Bench height (m):');
Incr_pushback = input('Enter length of incremental pushback (m):');
Bench_width = input('Enter minimum working bench width (m):');
theta = input('Enter bench slope angle (degrees):');
c = tan(theta*pi/180);
k = Height/c;
C = 262000/2.1;
Ae = 0:Incr_pushback:(Final_bench-Init_bench + Incr_pushback);
Be = 0:Incr_pushback:(Final_bench2-Init_bench2 + Incr_pushback);
Na = round((Final_bench-Init_bench)/Incr_pushback);
Nb = round((Final_bench2 - Init_bench2)/Incr_pushback);
%
% Specify the number of incremental pushback to be taken
% Calculate the volume excavated and tabulate the increment number
% Distance from initial toe of bench and Volume of materials excavated
% alongside each other in columns; "Time" is in days
%
n1 = 1:Nb;
n2 = n1-1;
Exca = n1*Incr_pushback;
Excb = n1*Incr_pushback;
G1 = 2.*(Init_bench + n2.*Incr_pushback).*(Init_bench2 + n2.*Incr_pushback)...
   - k*(Init_bench + Init_bench2 + 2.*n2.*Incr_pushback - k)...
   + sqrt((Init_bench + n2.*Incr_pushback).*(Init_bench2 + n2.*Incr_pushback))...
   .*sqrt((Init_bench + n2.*Incr_pushback - k).*(Init_bench2 + n2.*Incr_pushback -
k));

G2 = 2.*(Init_bench + n1.*Incr_pushback).*(Init_bench2 + n1.*Incr_pushback)...
   - k*(Init_bench + Init_bench2 + 2.*n1.*Incr_pushback - k)...
   + sqrt((Init_bench + n1.*Incr_pushback).*(Init_bench2 + n1.*Incr_pushback))...
   .*sqrt((Init_bench + n1.*Incr_pushback - k).*(Init_bench2 + n1.*Incr_pushback -
k));

Vn = (pi*Height/3)*G1;
Vn2 = (pi*Height/3)*G2;
V_incr = Vn2 - Vn;
Time = V_incr/C;
Major_axis = 80 + Exca;
Minor_axis = 30 + Excb;
Values = [n1'  Exca'  Major_axis' Minor_axis' V_incr' Time'];
fprintf('No_a  Major_axis Minor_axis  Volume Time')
[Values]
```

Fig. 5.3 Matlab M-File for Volume of Materials within an Elliptical Pit

Enter bench number: 1

Enter Initial bench width along major axis, a_o, (m): 80

Enter Initial bench width along minor axis, b_o, (m): 30

Enter final bench width along major axis, a_f, (m): 200

Enter final bench width along minor axis, b_f, (m): 150

Enter bench height (m): 13

Enter incremental pushback (Δx, m): 10

Enter working bench width (m): 80

Enter bench slope (degrees): 50

Table 5.2 summarizes the sample calculations of the volume of materials excavated using the Matlab algorithm with 10 m incremental pushbacks for the CMS option. For the CMS option the initial cut measures 1600 m × 60 m × 13 m while the final pit dimensions are 400 m × 300 m × 13 m, with a bench slope angle of 50°. For the CycEx CBCS Option, the initial pit dimensions are 328 m × 208 m × 13 m while the final dimensions are 3060 m × 3000 m × 13 m.

Table 5.2 Results of Sample Calculation from Matlab Algorithm for Elliptical Pit

Increment Number	Expansion (m)	Radius of Major Axis (m)	Radius of Minor Axis (m)	Volume Extracted (m^3)	Time to Extract (days)
1	10.00	90.00	40.00	44,661.18	0.36
2	20.00	100.00	50.00	52,776.28	0.42
3	30.00	110.00	60.00	60,921.99	0.49
4	40.00	120.00	70.00	69,078.76	0.55
5	50.00	130.00	80.00	77,240.44	0.62
6	60.00	140.00	90.00	85,404.61	0.68
7	70.00	150.00	100.00	93,570.17	0.75
8	80.00	160.00	110.00	101,736.54	0.82
9	90.00	170.00	120.00	109,903.44	0.88
10	100.00	180.00	130.00	118,070.67	0.95
11	110.00	190.00	140.00	126,238.14	1.01
12	120.00	200.00	150.00	134,405.76	1.08

5.3.3 Dynamics of Circular Pit Volume

In order to calculate the rate of increase in the pit dimensions in any direction, assuming the other dimensions remain constant, it is necessary to employ parabolic partial differential equation (PDE) models in Section 4.3 of Chapter 4. Figs. 5.4 and 5.5 are the Matlab M-Files for calculating the rate of change in volume of materials from a circular pit on any bench using PDE models respectively based on the dimensions of the North Mine pit.

```
% Matlab M.File for calculating the volume of a circular pit
%
function f = circ(t,y)
%
% State bench height, H; Bench slope width after projecting the length of
slope % to plan view, k; Set rate of change with time as DaDt = 1.92 m/hr
%
H = 13;
k = H/(tan(5*pi/18));
a = y(1); b = y(1)-k; DaDt = 1.92;
f(1) = DaDt;
f(2) = (pi*H)*(2*a-k)*DaDt;
```

Fig. 5.4 Matlab M.File for Volume of Materials within a Circular Pit (PDE)

```
% M.File in Matlab for calculating rate of expansion of a circular pit volume with time
% using partial differential equations. If initial pit radius is not specified, set it to a0 = 80 m.
%
function [t,y] = circ_driver(tf,a0)

if (nargin<2); a0 = 80;end
%
% Calculate initial pit volume, vo, from bench height, H, bench slope angle and plot
% graphs of time versus pit volume, Pit Volume versus Pit major axis, and Pit
% Dimensions versus time
%
H = 13; k = H/(tan(5*pi/18));
v0 = (pi*H/3)*(3*a0^2 - 3*a0*k + k^2);
[t,y]=ode45(@circ,[0 tf],[a0 v0]);

plot(t,y(:,2)); grid
xlabel('Time, hrs');
ylabel('Volume of Pit, m^3');
title('Rate of change in Volume of Circular Pit');

figure
plot(y(:,1),y(:,2), 'k:'); grid
title('Pit Volume vrs. Pit Dimensions');
xlabel('Pit Major Axis, m');
ylabel('Volume of Pit, m^3');

figure
plot(t,y(:,1),'k:'); grid
title('Pit Dimensions vrs. Time');
xlabel('Time, hr');
ylabel('Pit Major Axis, m');
```

Fig. 5.5 Matlab M-File for Rate of Volume Increase of a Circular Pit using PDEs

For the CMS option, the initial pit measures 160 m × 60 m × 13 m while the final dimensions of bench #1 are 10000 m × 6000 m × 13 m. During simulation involving the algorithm in Fig. 5.5, Matlab automatically links with and initializes the variables using the initial data in Fig. 5.4. An allowance of about 70% in the change in volume is made for a job efficiency factor[§] to accommodate all delays in the pit operations. The radii of the initial and final pit are set to 80 m and 3000 m respectively during calculations.

5.3.4 Dynamics of Elliptical Pit Volume

Figs. 5.6 and 5.7 are the Matlab M-Files for calculating the volume and the rate of change in volume of an elliptical frustum on any bench using partial differential equations respectively using the North Mine pit for modeling. During simulation, the algorithm in Fig. 5.7, Matlab automatically links with and initializes the variables with initial data from Fig. 5.6.

```
% Function to calculate the rate of change of volume of % an elliptical pit using partial
differential equations
%
function f=ellip(t,y)
%
% H = bench height; k is horizontal component of the inclined bench face, V is the
% volume of materials excavated (cubic meters)
H=13;
k=H/(tan(5*pi/18));
a=y(1); b=y(2); V=y(3); DaDt=1.92; DbDt=1.92;
%
% f(1) and f(2) are the rate of change of pit face per hour along the major and minor
% axis directions of the elliptical pit respectively.
%
f(1) = DaDt;
f(2) = DbDt;
G1 = 2*b - k;
G2 = 2*a - k;
G3 = 0.5*sqrt(a*b);
M1 = sqrt((a - k)*(b - k));
M2 = 0.5*sqrt(b/a);
M3 = 0.5*sqrt(a/b);
c1 = (pi*H/3)*(G1 + M2*M1 + G3*((b-k)/M1));
c2 = (pi*H/3)*(G2 + M3*M1 + G3*((a-k)/M1));

f(3) = c1*DaDt + c2*DbDt;
f = f';
```

Fig. 5.6 Matlab M.File for Calculating the Volume of an Elliptical Pit

§ Job efficiency factor = availability × utilization (0.875 × 0.8)

```
% M.File called ellip_driver in Matlab for calculating rate of change in elliptical pit
% volume using partial differential equations
% Specify initial pit major axis length = 80 m, initial pit minor axis length = 30 m
% and final time, tf (hr); Default lengths of major and minor axes set to 80 m
% and 30 m respectively if they are not specified in function equation
%
function [t,y]=ellip_driver(tf,a0,b0)

if (nargin<3); b0=30;end
if (nargin<2); a0=80;end
%
% Calculate the initial volume of pit using the bench height, initial pit dimensions
% along the major and minor axes
%
H = 13; k=H/(tan(5*pi/18));
v0 = (pi*H/3)*(2*a0*b0-k*(a0+b0-k)+ sqrt((a0*b0)*(a0-k)*(b0-k)));

[t,y] = ode45(@ellip,[0 tf],[a0 b0 v0]);

plot(t, y(:,3)); grid;
xlabel('Time, hrs');
ylabel('Volume of Pit, m^3');
title('Rate of change in Elliptical Pit Volume');

figure
plot(y(:,2),y(:,3), 'k:'); grid;
xlabel('Length minor axis, m');
ylabel('Volume Excavated, m^3');
title('Volume of Elliptical Pit vr. Minor Axis Length');
```

Fig. 5.7 Matlab M-File for Rate of Change in Volume of an Elliptical Pit using PDEs

5.4 Simulation of Mining Methods

There are many possible combinations of equipment (shovels, trucks, conveyor belts) which can be used to achieve the targeted production requirements of a mine. Simulation techniques have been found to offer excellent but less costly means of testing and experimenting with the materials handling systems on the computer to determine the possible combinations and to select the optimum ones. In this section, the mining operations of the CMS and CycEx CBCS options are modeled using the Visual SLAM with AweSim and Simphony software respectively. These software have been found to be the most suitable ones available for simulating mining and materials handling systems that involve both discrete and continuous mining equipment. They allow for the insertion of user inserts in either C++ or Visual Basic for the modeling of continuous operations.

5.4.1 Simulation of Discrete Mining System (CMS Model)

The CMS option at Syncrude consists of hydraulic and cable shovels loading various truck sizes in the materials handling operation. In this work, O&K RH400 shovels working with combinations of the most common truck sizes (i.e. the 240-tonne, 320-tonne and 360-tonne trucks) were considered for simulation under three scenarios. Scenario 1 has all the trucks dumping at one location at the crusher location while Scenarios 2 and 3 allow for two or three trucks to dump their materials simultaneously at the crusher location. Visual SLAM with AweSim software was used to model and simulate the various options under the different scenarios.

In a typical open pit mine, both the waste and ore must be mined at the required rates simultaneously. The waste stripping operation has to precede the ore mining and must continue at a good rate to ensure that sufficient ore material is exposed to meet the required ore production targets. In the North Mine, for every tonne of ore mined about 1.1 tonnes of waste have to be removed to maintain a stripping ratio of 1.1:1. This means that the amount of ore to be mined is almost the same as that of the waste stripping requirements. Thus the production planning and equipment scheduling for both ore and waste in this mine are almost the same. Thus simulation may safely be carried out to determine the fleet requirements for the ore mining operation which could also be used (with slight modifications) in meeting the required production targets for waste stripping operations. Accordingly, about 35 different combinations (Options) of truck fleets were considered and simulated under each scenario. The 35 options were arrived at by combining various numbers of 240-t, 320-t and 360-t trucks to each shovel. Fig. 5.8 is the Visual SLAM network for the ore mining operations. In this model, the entities are the trucks that cycle through the various stages in the network. The trucks are scheduled to return to specific shovels after dumping their materials at the crusher location. Loaded trucks waiting in the queue at the crusher location cannot balk or skip the queue to dump their materials elsewhere even if the queue length is excessive.

Testing of the collected data from the time and motion studies using the stabilised probability plot method and BestFit software show that the data closely fit various statistical distributions (Suglo and Szymanski, 1995; Anon., 1997). Tables 5.3 and 5.4 give the component times in ore mining and waste stripping operations of the rear dump trucks respectively.

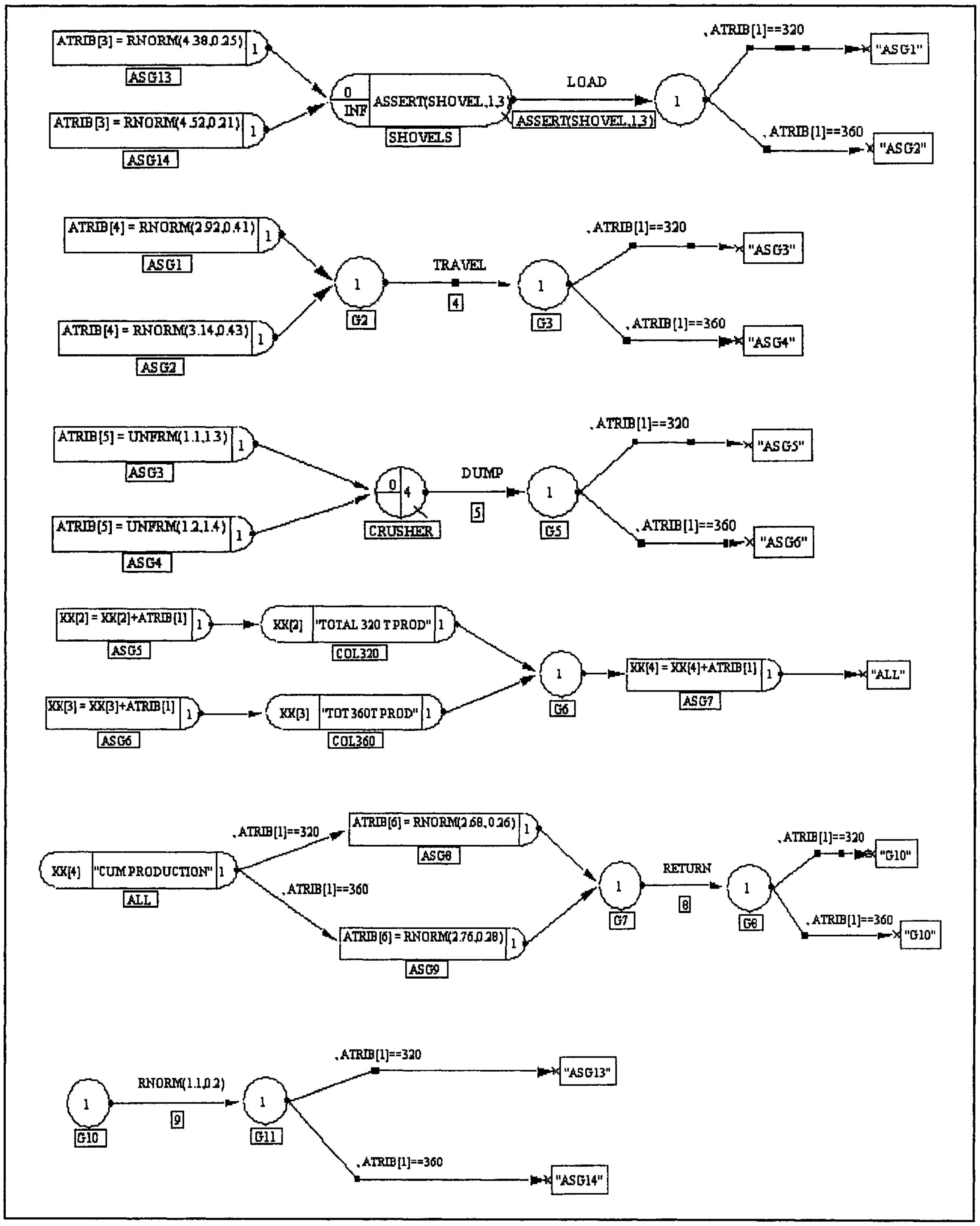


Fig. 5.8 Visual SLAM Network for the Ore Mining using 320 and 360 tonne trucks

Table 5.3 Trucks Component Times in Ore Mining

Type of Operation	Time (min.) (240-t truck)	Time (min.) (320-t truck)	Time (min.) (360-t truck)	Statistical Distribution
Loading time	3.45 ± 0.20	4.38 ± 0.25	4.52 ± 0.21	Normal
Travel time (loaded)	2.80 ± 0.40	2.92 ± 0.41	3.14 ± 0.43	Normal
Return time (empty)	2.45 ± 0.25	2.68 ± 0.26	2.76 ± 0.28	Normal
Dumping time	1 – 1.2	1.1 – 1.3	1.2 – 1.4	Uniform
Spotting time	1.1 ± 0.2	1.1 ± 0.2	1.1 ± 0.2	Normal

Table 5.4 Trucks Component Times in Waste Stripping

Type of Operation	Time (min.) (240-t truck)	Time (min.) (320-t truck)	Time (min.) (360-t truck)	Statistical Distribution
Loading time	3.44 ± 0.21	4.35 ± 0.24	4.48 ± 0.20	Normal
Travel time (loaded)	4.58 ± 0.42	4.62 ± 0.45	4.72 ± 0.43	Normal
Return time (empty)	3.43 ± 0.23	3.63 ± 0.24	3.73 ± 0.29	Normal
Dumping time	1 – 1.2	1.1 – 1.3	1.2 – 1.4	Uniform
Spotting time	1.1 ± 0.2	1.1 ± 0.2	1.1 ± 0.2	Normal

Table 5.5 summarizes the initial costs, maintenance and fuel costs, operator costs (including other incentives, vacation, overtime, health and unemployment insurances) and life spans of the dump trucks, shovels and operator costs. Syncrude employs the double declining balance method of depreciation in calculating the depreciation of the major capital equipment. Up to 25% of the total cost is allowed for miscellaneous costs on all items (e.g. supervision costs).

Table 5.5 Costs of Major Mining Equipment

Item/Equipment	Initial Cost ($ million)	Maintenance & Operating Cost ($ million/yr)	Life Span (yr)
O&K RH Shovel	10.00	3.80	20
Trucks (240 tons)	2.50	0.50	6
Trucks (320 tons)	2.75	0.55	6
Trucks (360 tons)	3.00	0.60	6
Graders (large)	1.30	0.50	6
Dozers (large)	1.60	0.90	6
Crusher	22.00	4.00	20
Shovel operator wages ($/min.)		2.20	
Truck operator wages ($/min.)		1.60	
Scheduled Operating time per year (hr.)		525,600	
Cycle time of shovel (s)		61.60	

Source: Coward (2003)

5.4.2 Simulation of Continuous Mining System (CycEx CBCS Model)

The CycEx CBCS option comprises a shovel, a mobile crusher and a series of conveyor belt wagons that convey the materials excavated by the shovel at the face to a slurrification unit for slurrification before being pumped to join the main HTS of the mine. Simphony software was used to model the CycEx CBCS option in this work because it allows for the insertion of user inserts written in Visual Basic to be used in modeling the discrete loader-continuous materials handling system involving belt conveyors in the option (Anon., 2000a).

Fig. 5.9 is the Simphony network for simulating the continuous CycEx CBCS model. It models three shovels as resources which can be captured during the loading cycle. After dumping the load after each cycle, the shovel is released for another excavation cycle. The apron feeder, crusher and conveyor belt capacities are modeled as functions of each dipper load that is released into the hopper taking into account the fact that their capacities must be matched to avoid ore spillages or clogging of the system. In this model, the entities are the excavated oil sand materials which pass through the various stages in the network.

Each of the three shovels is modeled with a separate split in the network with the hopper-crusher-belt conveyor wagons and surge bins that are attached to them. The model was simulated over periods ranging from 450 minutes to 1440 minutes under five different scenarios. Four simulation runs were conducted on each occasion. The first scenario involved the present operating conditions and equipment capacities (the standard case) with the input parameters summarized in Table 5.6. In the second and third scenarios, the carrying capacity of the belt conveyor wagons was increased and decreased by 20% respectively. In the fourth and fifth scenarios, the digging capacity of the shovel was reduced by 20% and 40% respectively to assess the impact of such changes on total production and system efficiencies. Due to surges and variations during operation, the cycle time of the various operations of the equipment have been estimated using various statistical distributions. Statistics on total production, system productivity, shovel utilization are collected on the system at the end of each simulation run for analysis.

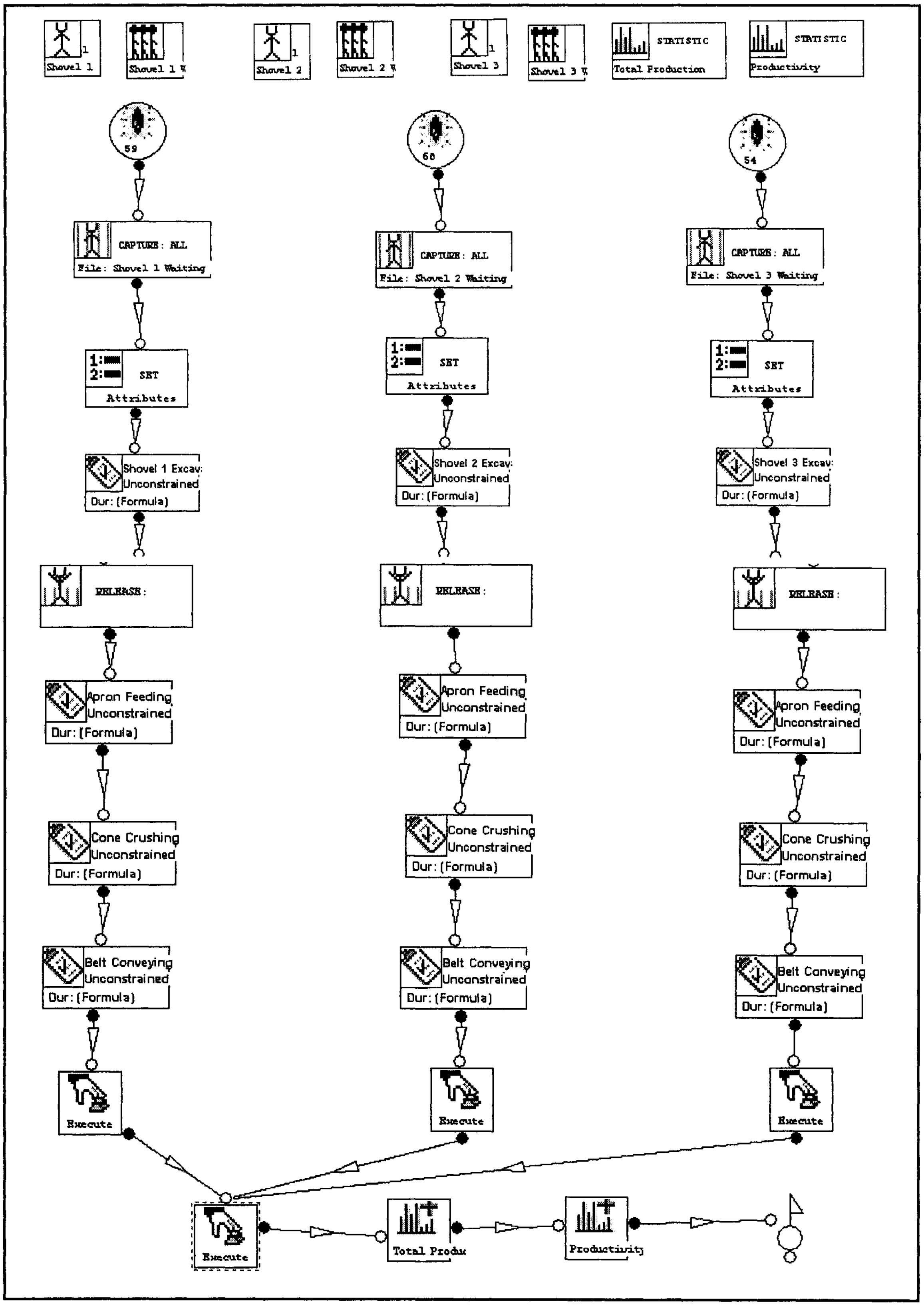


Fig. 5.9 Simphony Network for CycEx CBCS Option for Ore Mining

Table 5.6 Input Parameters of the CycEx CBCS Option

Parameter/Equipment	Capacity	Statistical Distribution
Nominal shovel dipper capacity (m^3)	43.5	Constant
Shovel cycle time (min)	0.88 – 1.25	Uniform
Shovel digging capacity (tonne/min)	61.83 – 85.02	Uniform
Apron feeder capacity (tonne/min)	80 – 91.67	Uniform
Belt conveyor capacity (tonne/min)	101.67 ± 8.33	Normal
Double cone crusher capacity (tonne/min)	80 – 91.67	Uniform
Belt conveyor velocity (m/s)	5.2 ± 0.22	Normal
Surge tank/bin capacity (tonnes)	18,000	Constant
Hopper Capacity (tonnes)	6,000	Constant

5.5 Mechanical Event Simulation of CycEx CBCS Model

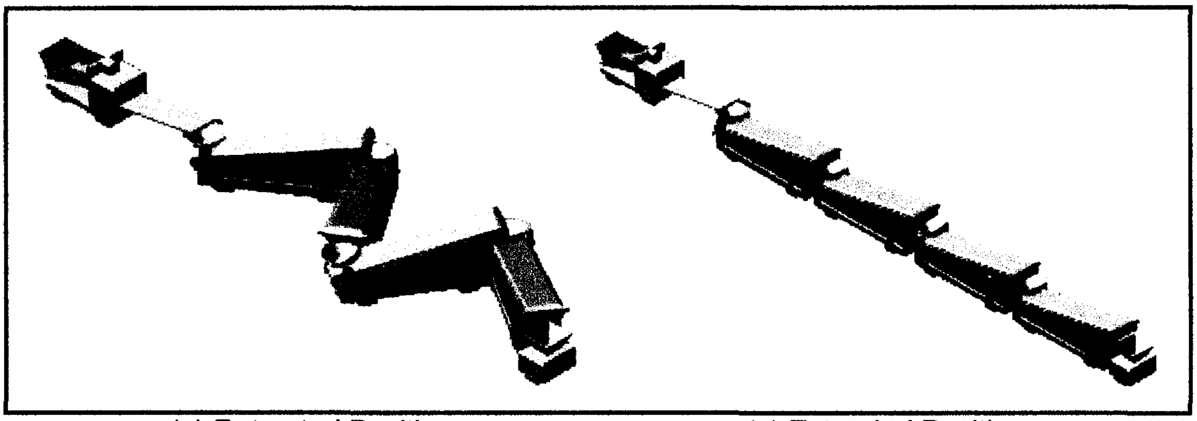
Some of the challenges of the AFS option include the need to ensure that the movements of the shovel and train of mobile or semi-mobile belt conveyor wagons are well-matched and monitored to avoid imposing undue bending stresses and torques on the component parts of the system that could lead to sudden failure of the parts and serious disruptions to production. In Section 4.6, the dynamics and kinematics of the CycEx CBCS option were analyzed and the equations for of motion were derived. This section contains the results of the mechanical modeling of the CycEx CBCS using the ADAMS software. The characteristics of the oil sands, the targeted displacement per week and the required capacity of the conveyor belt trains that will ensure efficient operation of the CycEx CBCS option are given in Table 5.7. These parameters are required as input data in the modeling and mechanical event simulation of the CycEx CBCS option.

Table 5.7 Design Parameters CycEx CBCS Option

Design Parameter	CycEx CBCS
Unit length (m)	20 - 60
Angles, θ_3 or α	0 - 60
Diameter or width (m)	3
Advance (m/week)	400
Material density (kg/m^3)	2,100
Capacity (t/ph)	6,100

5.5.1 Modeling of CycEx CBCS Option

Fig. 5.10 shows 3D solid image models of CycEx CBCS option using the ADAMS software. Fig. 5.10a shows CycEx CBCS model in a retracted position while Fig. 5.10 shows the model in an extended position. The major components of CycEx CBCS system include a mobile crusher, a system of conveyor belt wagons and semi-mobile mixing tower. In the CycEx CBCS option, oil sands will be loaded into the mixing tower via a hopper attached to it. The conveyor belt wagon units are connected together by revolute joints. The conveyor belt wagon rotates dependently on a revolving frame that rotates on the lower crawler-mounted support structure. The top left hand side of Fig. 5.10a shows a mobile crusher unit that feeds the oil sand materials on to a series of belt conveyor wagons that feed the materials to the main slurrification unit for slurrification and subsequent pumping to the main treatment plant through the HTS. It shows that the conveyor belt wagons are hooked together to form a train.



(a) Retracted Position (a) Extended Position

Fig. 5.10 Mechanical Simulation Model of CycEx CBCS Option
(Frimpong et al., 2003)

The frequent need to relocate the AFS train in line with the changing position of the shovel excavator within the pit subjects the component parts of the equipment units to varying degrees of torsional stresses, torques and bending moments. It is therefore necessary to model and conduct a mechanical simulation analysis on the CycEx CBCS option. This is to ascertain whether it can withstand the mechanical forces imposed on it during the operation of the AFS option to meet the production requirements of the mine.

The components of the CycEx CBCS model which include the mobile hopper-crusher attached wagons, conveyor belt, crawlers, control equipment and semi-mobile mixing tower are modeled as multibody systems (Stribersky et al., 2002; Frimpong et al., 2003). Using ADAMS software, a differential-algebraic equation system can be derived and integrated in time to simulate the dynamics of the CycEx CBCS model. The dynamic movements of the model depend on the interaction of the elements like weightless joints, which constrain the relative movements of the hopper-attached conveyor belt wagons, and on weightless force elements like springs and dampers (Frimpong et al., 2003).

5.6 Summary

In this chapter the volume of materials excavated from circular and elliptical pits as well as the changes in the pit geometry were modeled using Matlab. PDEs were used to model the differential changes in the pit volume and dimensions with the lateral expansion of the pit dimensions. The algorithms were verified and sample calculations were done. The operations of the two mining options were modeled with the appropriate networks in Visual SLAM and Simphony. The mechanical event modeling of the CycEx CBCS option was done using ADAMS software.

CHAPTER 6

AFS PERFORMANCE SIMULATION MODELING AND ANALYSIS

6.0 Introduction

Different combinations of equipment can be used to achieve the desired production targets of a mine. However, some combinations of equipment and their operating times result in lower unit operating costs and higher system efficiencies than others. As the objective function is to maximize the overall profit of the mine, it is necessary to simulate the production schedules and sequence of operations of the two mining options in continuous time so as to make comparative analysis of the production-economic functions of the AFS technology versus the conventional shovel-truck mining system. This chapter deals with the solution of the layout evolution based on ellipsoid and spherical processes, performance simulation, economic analysis and risk characterization of the CMS and AFS options.

6.1 Geometric Volume of Pits

Surface mine production scheduling is complicated by the fact that most open pit mines work with multiple benches and often involve the simultaneous excavation of both ore and waste from a large number of working faces. These production schedules and plans are (i) used to maintain and maximize the expected profit; (ii) determine the future investment in mining; (iii) optimize the return on investment; (iv) evaluate alternative investment options; and (v) conserve and develop the mine's resources. To determine the geometrical pit volumes, the pit expansion rates and the volume of materials to be handled at any given time within the pits, Matlab software was employed to calculate the geometric volumes of materials excavated on the different benches in a multi-bench, multi-face open pit mine subject to constraints imposed by the initial and boundary conditions.

6.1.1 Geometric Volume of Materials Excavated from a Circular Pit

The Matlab algorithm for calculating the geometric volume of materials excavated from a circular pit as given in Fig. 5.2 was applied to the operations at Syncrude. Appendix B is a sample of the results of the volume of oil sands excavated from bench #1 at the North Mine using incremental pushbacks of 1.92 m wide from the initial cut to the final pit limit. The column labeled "Expansion" in Appendix B refers to the total increase in the dimension of the base (bottom) of the frustum after a number of incremental pushbacks are taken.

The pit limits are reached when the pit dimensions along the minor axis reach the boundary conditions along the minor axis of the pit. In this case the calculations are terminated when b_o is 3000 m. The results show that, at the projected rate of production of 262,000 tonnes/day from the North Mine, the ore reserves within the circular pit on bench #1 would be mined out in 2,934.25 days (8.04 years). Detailed calculations show that it will take between 0.11 to 3.76 days to mine the ore materials within each incremental pushback on bench #1.

Fig. 6.1 shows the volumes of oil sands excavated from each of the three benches together with the cumulative volume extracted from the pit using the CMS option. It is noted that the volumes excavated from benches 1 to 3 were almost equal except that the extraction of ore from benches 2 and 3 were delayed until enough space was available on the previous bench. Mining on bench #2 starts about 174 days after bench #1. As well, mining on bench #3 begins 172 days after mining has started on bench #2 Appendix C summarizes the volume of materials excavated and times to completely mine out those volumes using the CMS option on bench nos. 1 to 3.

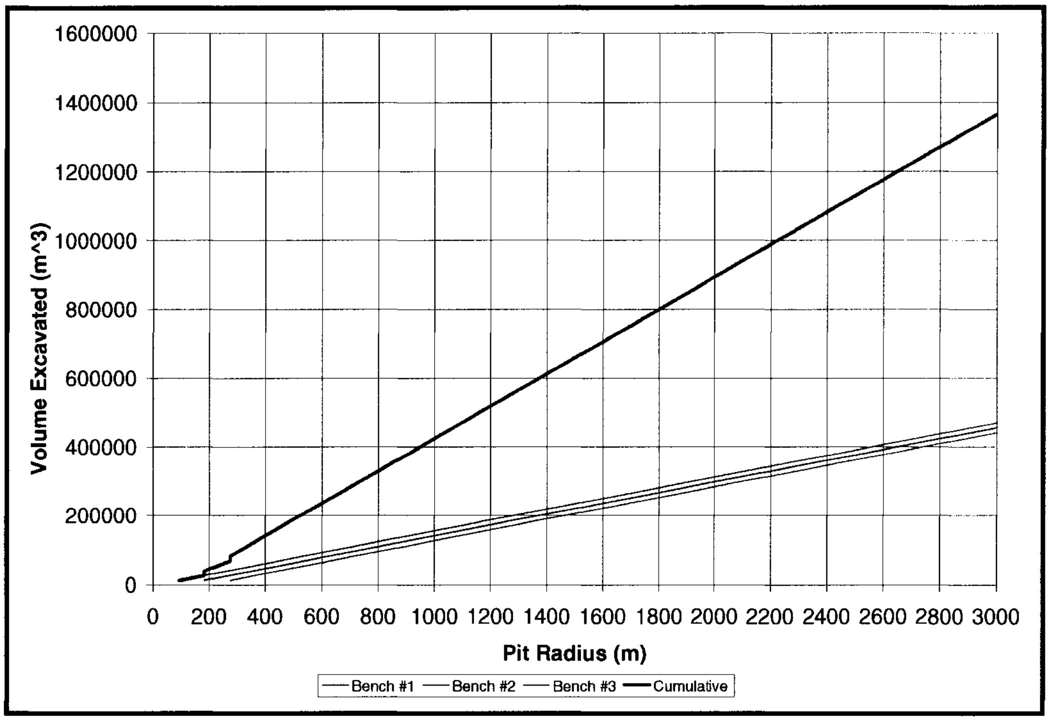


Fig. 6.1 Volume of Materials Excavated from Benches 1 to 3 using CMS Option

When the CycEx CBCS option is applied to the pit with a circular frustum, the volumes excavated and the times to completely extract the volumes from the various benches are also given in Appendix C. It is assumed here that the pit limits are reached along the minor axis first. In this case the calculations are terminated when b_o is 3000 m. The remaining reserves along the major axis can then be mined later and the volumes calculated using PDEs. It is noted that it will take from 6.25 to 8.01 years to completely mine all the ore on benches 1 to 3. Mining on bench #2 starts about 332 days after bench #1. As well, mining on bench #3 begins 313 days after mining has started on bench #2.

Fig. 6.2 is a 3D diagram of the an open pit with three benches while Fig. 6.3 is a solid 3D diagram of the faces of a circular pit after taking ten incremental pushbacks of 10 m each. The initial pit radius in Fig. 6.3 is 80 m while the final pit diameter at the top is 381.82 m after taking the 10 pushbacks. The spaces in between the colored circular frustums show the voids created after taking each pushback.

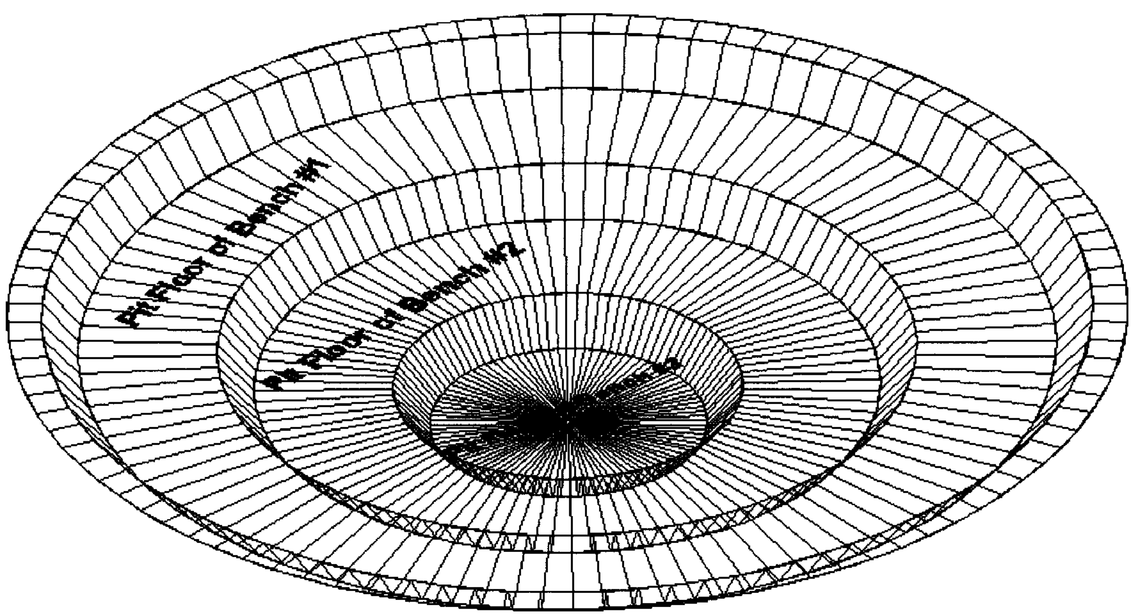


Fig. 6.2 3D View of a Circular Pit with three Benches

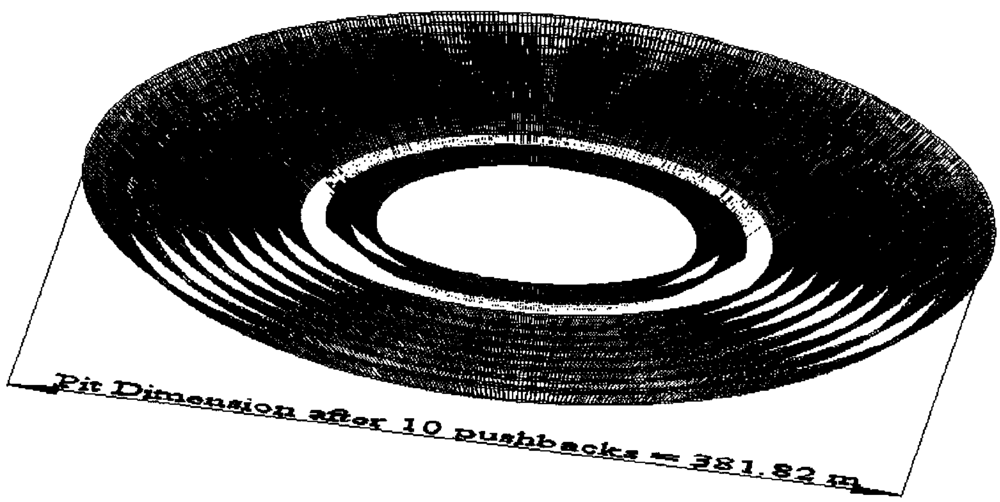


Fig. 6.3 3D View of Circular Pit Faces after taking incremental pushbacks

6.1.2 Geometric Volume of Materials Excavated from an Elliptical Pit

The Matlab algorithm for calculating the geometric volume of materials excavated from an elliptical pit was applied to the operations at Syncrude (see Fig. 5.3). Appendix C gives the detailed results of the calculated volumes of materials excavated from an elliptical pit using the algorithm in Fig. 5.3. The pit limits are reached when the pit dimensions along the minor axis reach the boundary conditions. In this case the calculations are terminated when b_o is 3000 m. Appendix C summarizes the results of the volumes excavated from benches 1 to 3 and the times taken to mine all the ore at the projected rate of production of 262,000 tonnes/day from the North Mine with the elliptical pit configuration when the CMS option is used. From Appendix C it will take between 7.21 years and 8.17 years to mine the materials from benches 1 to 3 using the CMS option.

When the CycEx CBCS option is applied to an elliptical pit, the volumes excavated and time taken to completely extract the volumes from the various benches are given in Appendix C. It is assumed here that the pit limits are reached along the minor axis first. In this case the calculations are terminated when b_o is 3000 m. The results show that it will take from 6.39 to 8.19 years to completely mine all the ore on benches 1 to 3 using the CycEx CBCS method. The remaining reserves along the major axis on the various benches can then be mined later and the volumes calculated using PDEs. Except for rounding errors, the values given in Appendix C closely agree with those summarized in Tables 4.3 and 4.6 using the mathematical models developed in Chapter 4. This shows that the Matlab algorithms developed in this work are adequate for calculating the volumes of materials in pits of different shapes.

Fig. 6.4 is a 3D diagram of an elliptical pit with three benches at the start of the AFS method on Bench #3 while Fig. 6.5 is a solid 3D diagram of the faces of the elliptical pit after taking ten incremental pushbacks of 10 m each. The initial pit dimensions are 328 m x 208 m while the final pit diameter at the top is 549.82 m x 431.82 m after taking 10 pushbacks.

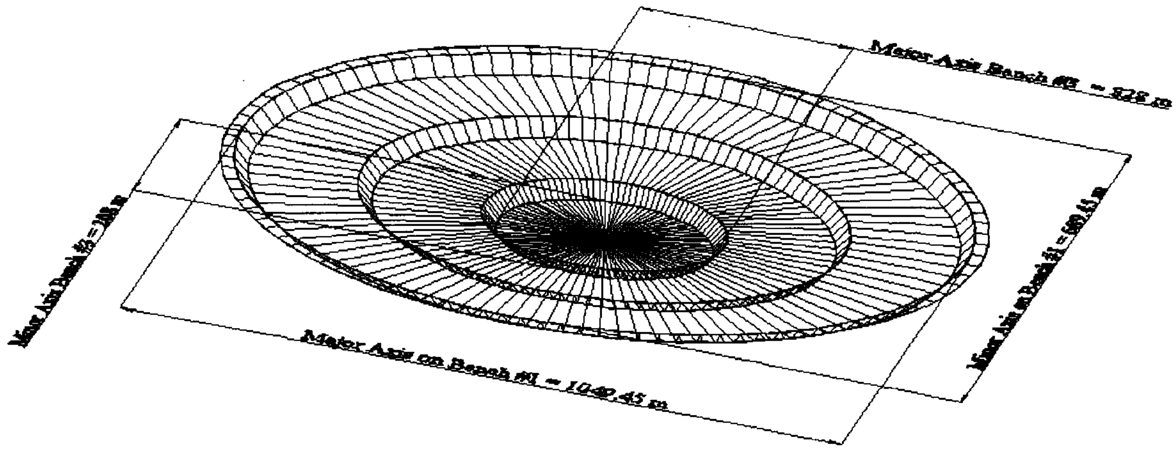


Fig. 6.4 3D View of an Elliptical Pit with three Benches

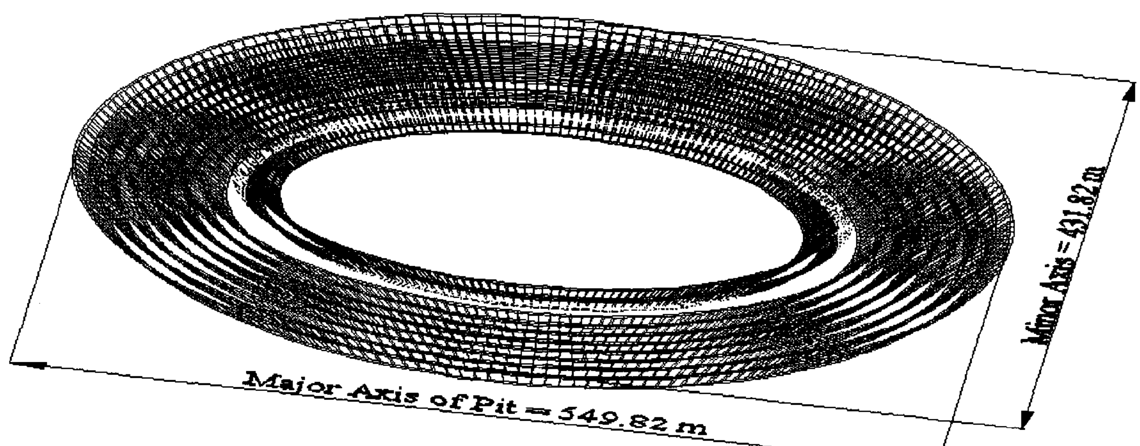


Fig. 6.5 3D View of Elliptical Pit Faces after taking incremental pushbacks

6.2 Dynamics of Excavated Volumes

As the pit expansion and materials handling operations within the pit involve continuous processes, most of the available software and simulators for mine planning, design and scheduling cannot be used to accurately capture the continuous changes in the pit dimensions and layout. Parabolic partial differential equations (PDE) with their associated boundary conditions are used to model changes in pit volume and configuration when the pit expansion is in only one direction. This work represents the first successful attempt at using PDE in the calculation of the geometric volumes of materials excavated from open pits. Thus the PDE function in Matlab was used in the geometric volume calculations and in the determination of the rate of expansion in the pit volume and dimensions.

The resulting graphs of the pit dimensions versus time and the volume of the pit at any given time on bench #1 are shown in Figs. 6.6 and 6.7 respectively. Fig. 6.6 shows that the rate of increase in the pit dimensions (gradient of the line) was about 0.048 m/hr. Using the CMS option, the volume of ore that can be mined from benches 1 to 3 (obtained from geometric calculations earlier on) are 3.66×10^8 m^3, 3.44×10^8 m^3 and 3.23×10^8 m^3 respectively. Fig. 6.7 shows that the reserves on bench #1 will be completely extracted after about 69,800 hr (2,908.33 days). As well, the 3.44×10^8 m^3 of reserves contained on bench #2 will be excavated after about 67,500 hr (2,812.50 days) while the 3.23×10^8 m^3 of ore contained in bench #3 will be completely excavated after about 65,000 hr (2,708.33 days). From Figs. 6.7 and 6.8, if the differential volume expansion of the pit is known, then the time it will take to excavate it can be obtained from the x-axis or vice versa.

Fig. 6.8 shows the volume of the circular pit at any given time when the CycEx CBCS option is employed using PDEs. In this system, the volume of ore that can be mined on bench #1 is 3.65×10^8 m^3 while that on benches 2 and 3 are 3.23×10^8 m^3 and 2.84×10^8 m^3 respectively. From Fig. 6.8, the volume of ore contained on bench #1 will be extracted within 68,500 hr (2,854.17 days). The ore on bench # 2 will be completely mined out after about 62,400 hours (2,600 days) while the ore contained on bench #3 will be completely extracted after 61,000 hr (2,541.67 days). Figs. 6.9 to 6.11 show the respective rates of change of the major axis of the elliptical pit, the volume of the elliptical pit at any given time when the CMS option is used (assuming constant extraction of the ore) and the volume of the elliptical pit at any given time measured along the major axis for the CycEx CBCS option.

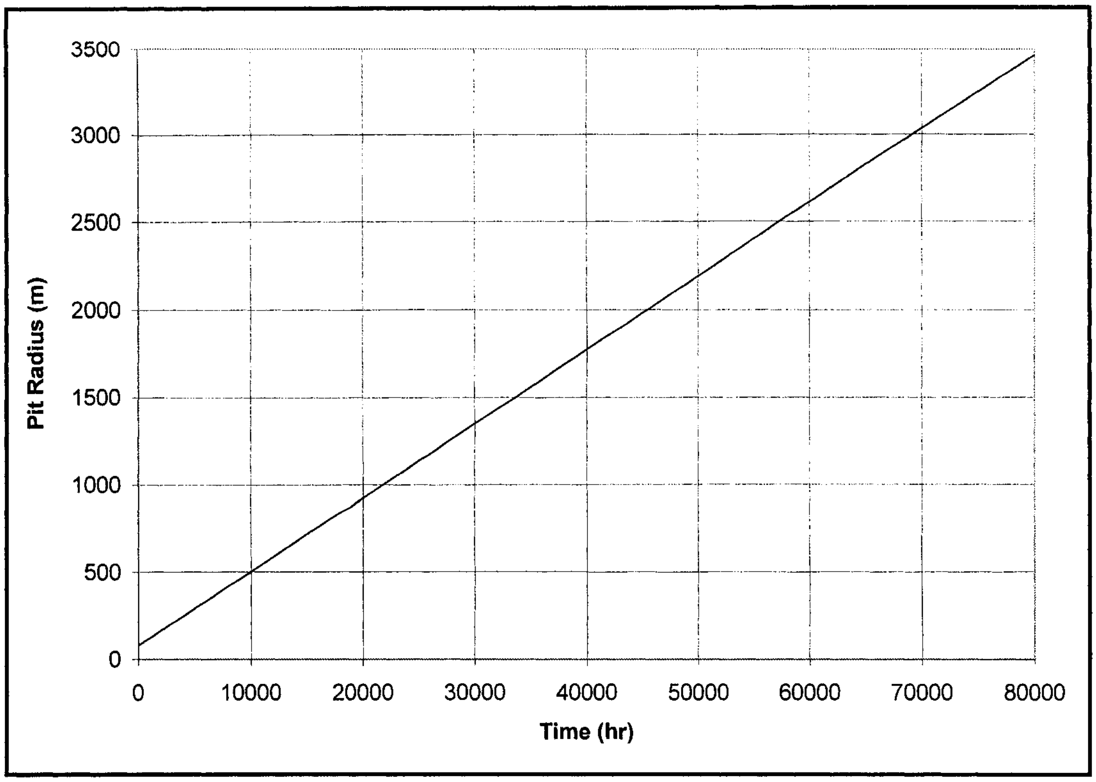


Fig. 6.6 Rate of Change in Pit Dimensions for a Circular Pit using PDEs

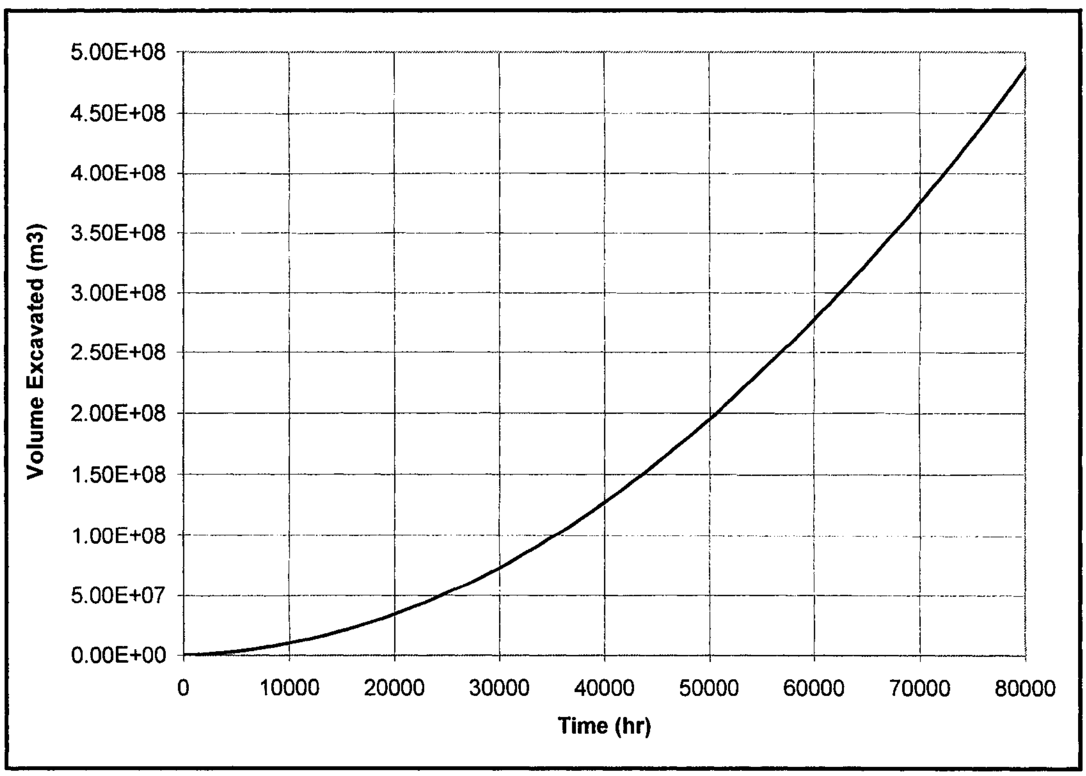


Fig. 6.7 Volume of Circular Pit vs. Time using CMS Option

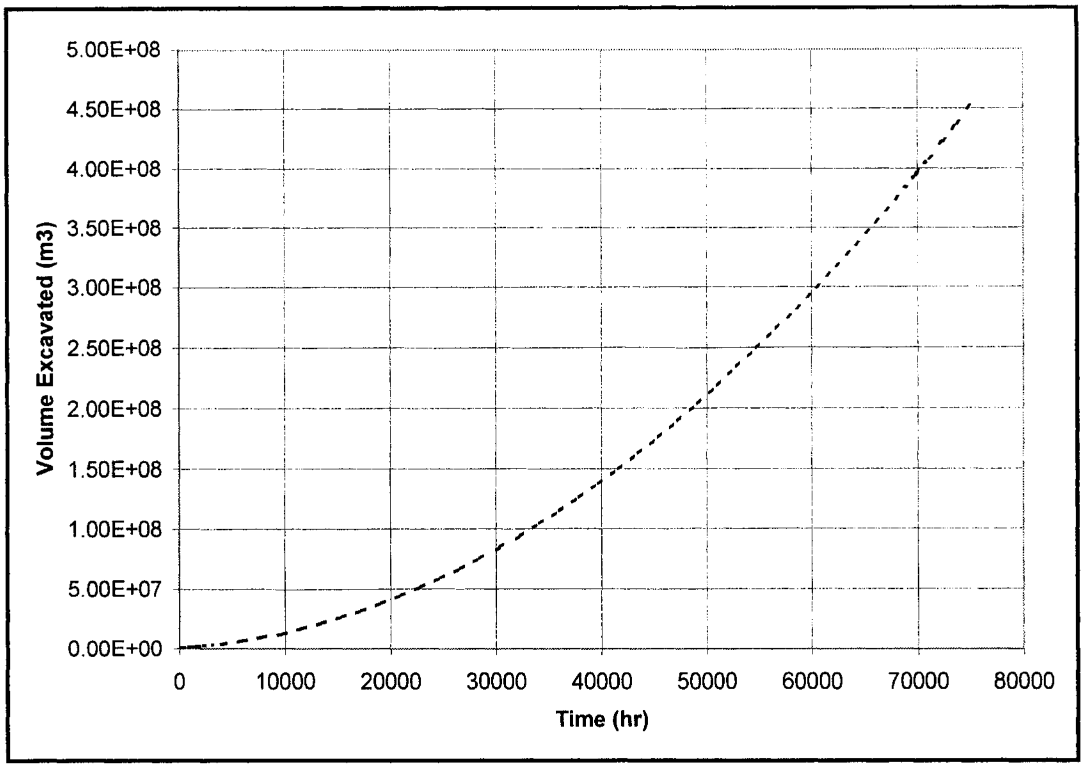


Fig. 6.8 Volume of the Circular Pit vs. Time using CycEx CBCS Option

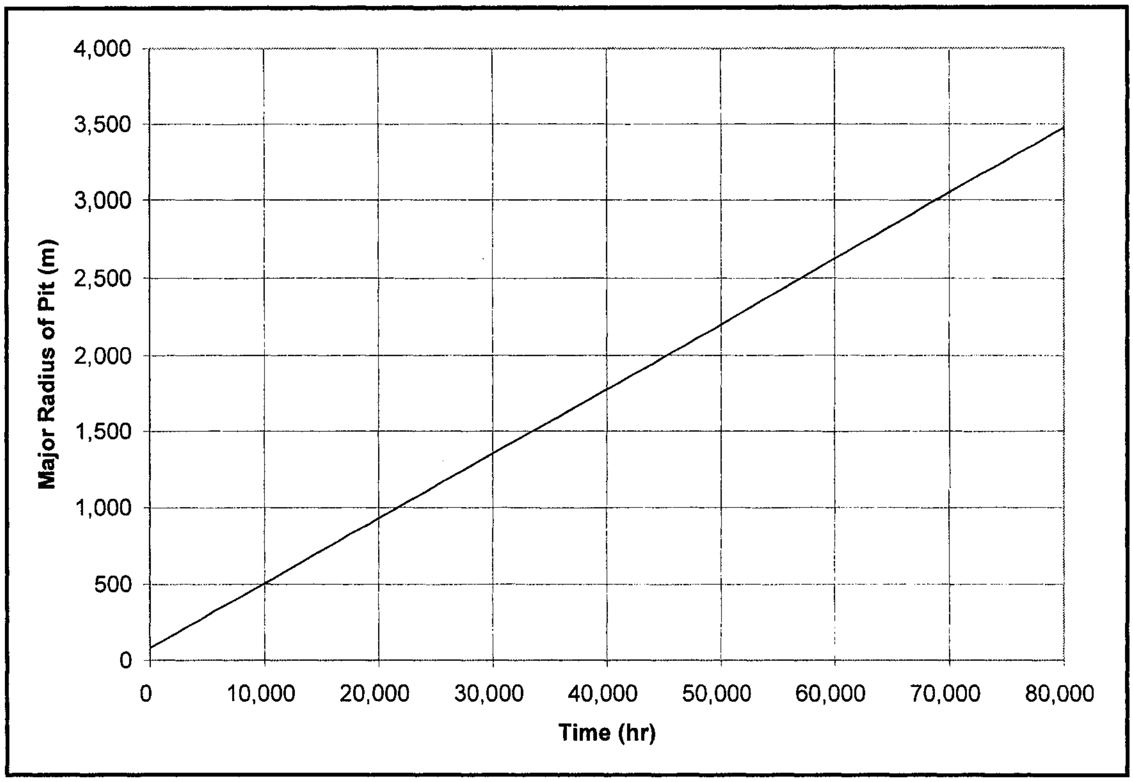


Fig. 6.9 Rate of Change in Major Axis of Elliptical Pit on Bench #1 with CMS Option

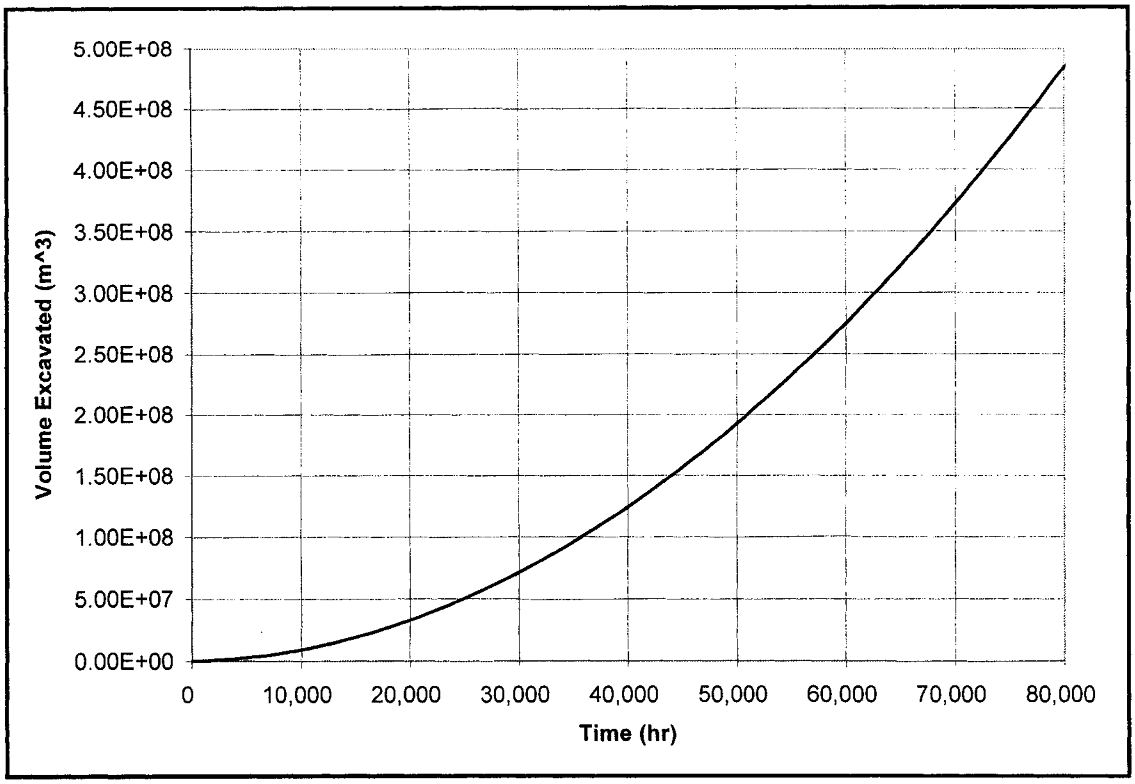


Fig. 6.10 Volume of Elliptical Pit vs. Time using PDEs on Bench #1 for CMS Option

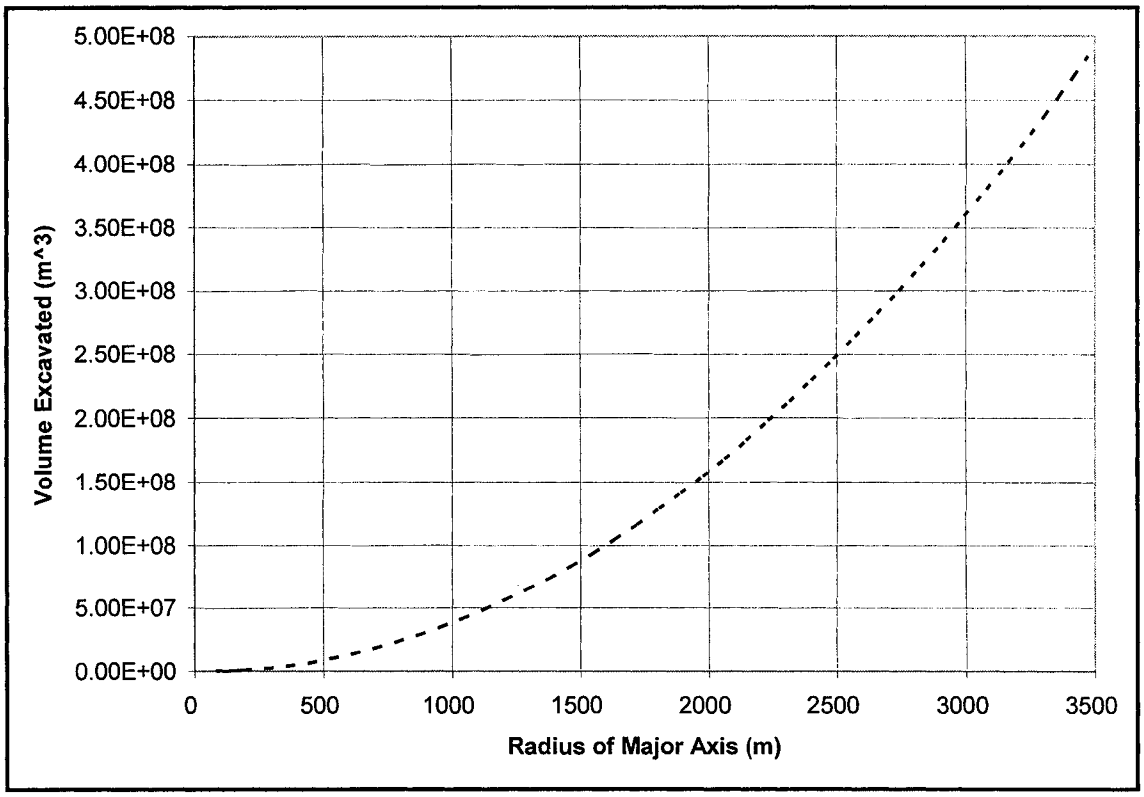


Fig. 6.11 Volume of Elliptical Pit vs Radius of Major Axis of Pit using CMS Option

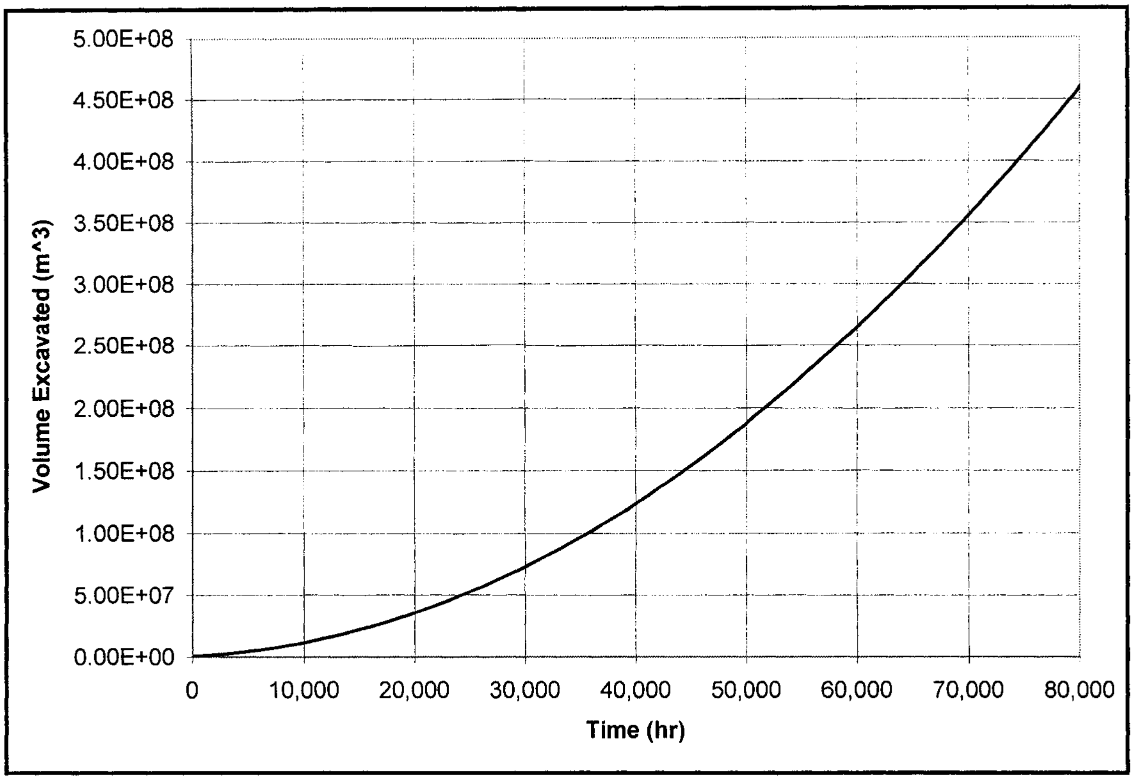


Fig. 6.12 Volume of Elliptical Pit vs. Time using PDEs for CycEx CBCS Option

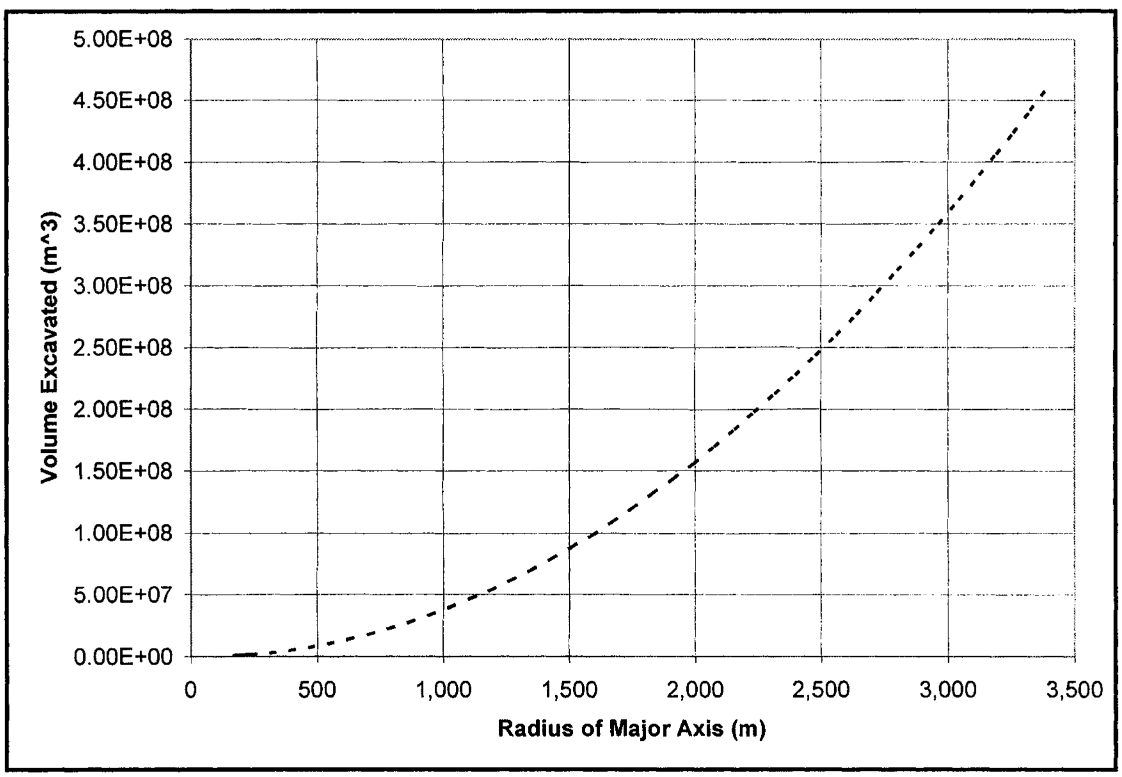


Fig. 6.13 Volume of Elliptical Pit vs. Radius of Major Axis of Pit using CBCS Option

The geometric volumes of ore that can be extracted from the elliptical frustum on benches 1 to 3 using the CMS option are 3.72×10^8 m^3, 3.50×10^8 m^3 and 3.28×10^8 m^3 respectively. From Fig. 6.9, the dimensions of the elliptical frustum on bench #1 will expand at the rate of 0.046 m/hr. From Fig. 6.10, the ore contained on bench #1 will be extracted in 69,600 hr (2,900 days) while the ore in benches 2 and 3 will be extracted in 68,200 hr (2841.67 days) and 65,100 hr (2,710 days) respectively. Fig. 6.11 shows that the pit will expand to about 3,060 m along the major axis before reaching the boundary conditions along the minor axis.

Figs. 6.12 and 6.13 show the volume of the elliptical pit at any given time and the volume of the pit at any given time measured along the major axis of the elliptical pit for the CycEx CBCS option. The geometric volumes of ore that can be extracted from benches 1 to 3 of the elliptical pit using the CycEx CBCS option are 3.73×10^8 m^3, 3.31×10^8 m^3 and 2.91×10^8 m^3 respectively. From Fig. 6.12, the ore in benches 1 to 3 would be completely extracted in 71,800 hr (2,991.67 days), 66,700 hr (2,780.50 days) and 60,000 hr (2,497.53 days) respectively. As well, Fig. 6.13 shows that the pit will only expand to about 3,060 m along the major axis, before reaching the boundary conditions along the minor axis of 3,000 m. Figs. 6.14 and 6.15 are the plan and isometric views of five incremental pushbacks taken in an elliptical pit on any bench at the beginning of the CycEx CBCS option with partial expansion in the major axis direction.

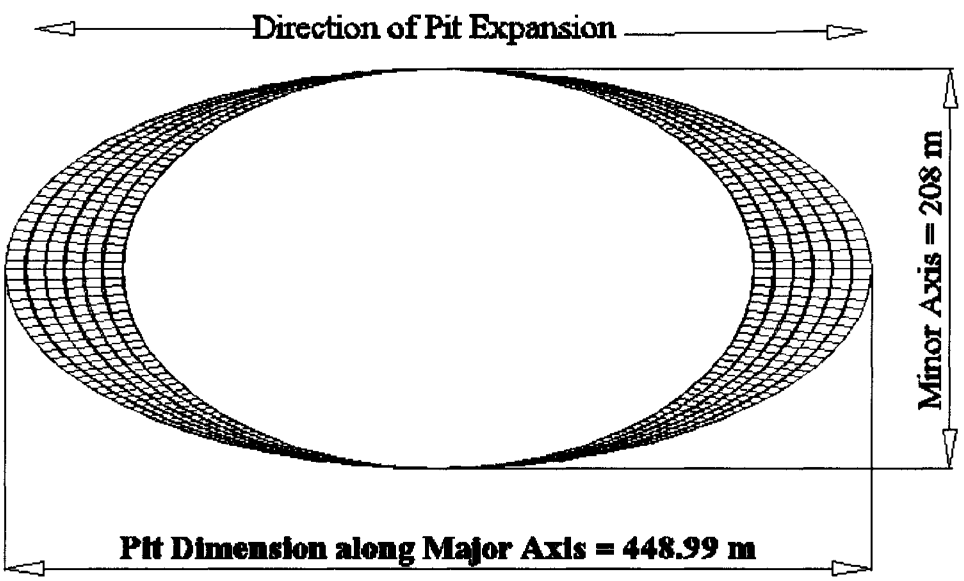


Fig. 6.14 Plan View of Incremental Pushbacks using PDEs in Elliptical Pit

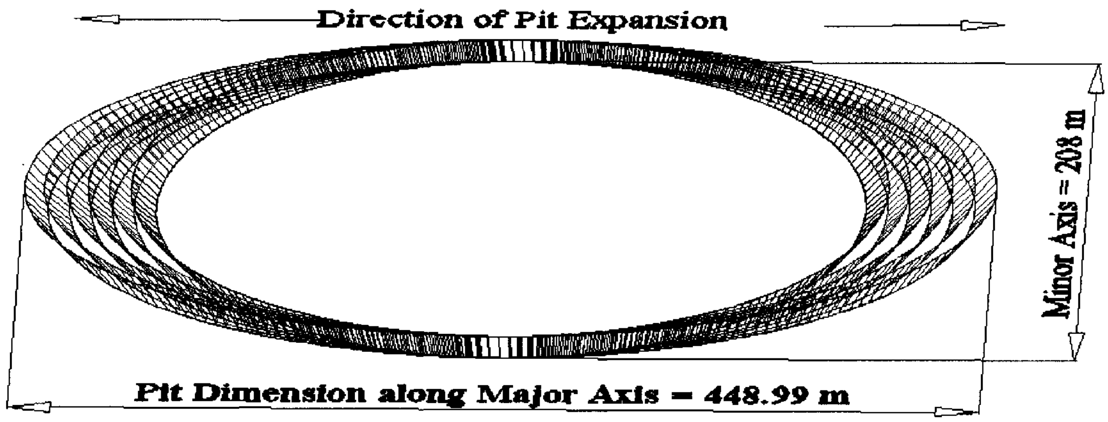


Fig. 6.15 Isometric View of Incremental Pushbacks using PDEs in Elliptical Pit

Tables 6.1 and 6.2 summarize the results of the times required to mine all the ore on benches 1 to 3 using geometric calculations and PDEs for circular and elliptical pit configurations respectively. The results show that the calculated values from geometric calculations are almost the same as those obtained from PDEs for different pit configurations. Thus it has been shown that PDEs may be successfully used in volume calculations to arrive at the same values as obtained from geometric calculations. However, calculations using PDEs for pit expansion in all directions are usually terminated when the boundary conditions in any direction are attained. This may leave some reserves in the pit in the other direction whose volume can be similarly determined using PDEs if excavation is assumed to be taking place in only one direction. The results from both tables show that generally, the PDE calculations tend to overestimate the volumes excavated as the pit deepens (i.e. at the lower benches) relative to geometric calculations.

Table 6.1 Time to Excavate Ore Reserves from Circular Pit Configuration

		Geometric Calculations (days)	PDE Calculations (days)	Ratio
Bench #1	CMS	2934.25	2908.33	1.01
	CBCS	2,926.90	2854.17	1.03
Bench #2	CMS	2778.26	2812.50	0.99
	CBCS	2615.68	2600.00	1.01
Bench #3	CMS	2630.10	2708.33	0.97
	CBCS	2635.24	2541.67	1.04
Average				1.01

Table 6.2 Time to Excavate Ore Reserves from Elliptical Pit Configuration

		Geometric Calculations (days)	PDE Calculations (days)	Ratio
Bench #1	CMS	2,984.33	2900.00	1.03
	CBCS	2,987.87	2991.67	1.00
Bench #2	CMS	2827.03	2841.67	0.99
	CBCS	2,673.50	2780.50	0.96
Bench #3	CMS	2677.58	2710.00	0.99
	CBCS	2,376.56	2497.53	0.95
Average				0.99

6.3 Discrete and Continuous Event Simulations

Visual SLAM with AweSim and Simphony software packages were used in this section for modeling, verifying and simulating the CMS and CycEx CBCS mining options which involve discrete events, continuous events as well as combined discrete-continuous event operations.

6.3.1 Simulation of CMS Option

Figs. 6.16 to 6.18 are graphs of the production of the top 6 truck fleets of the 35 options simulated using Visual SLAM with AweSim under each scenario. Appendix D summarizes the results of the best 10 options under each scenario using the total cost per tonne as the determinant. The details of all simulation runs for all options in the CMS option are contained in Appendix E. Preliminary calculations and simulation results showed that the mine can achieve the targeted daily production requirements of 262,000 tonnes with 2.42 shovels. Hence, all the simulations were done with three shovels working in combination with the trucks in each option. The times for each option indicate the optimum shift time over which the truck fleet must operate to meet the daily ore production target of 262,000 tonnes. The optimum truck fleets in the three scenarios were Option A9 (operating over 1094 min.), Option B34 (operating over 1094.85 min.) and C9 (operating over 1092.49 min.). These options involved the allocation of 320-tonne and 360-tonne trucks to each of the three shovels.

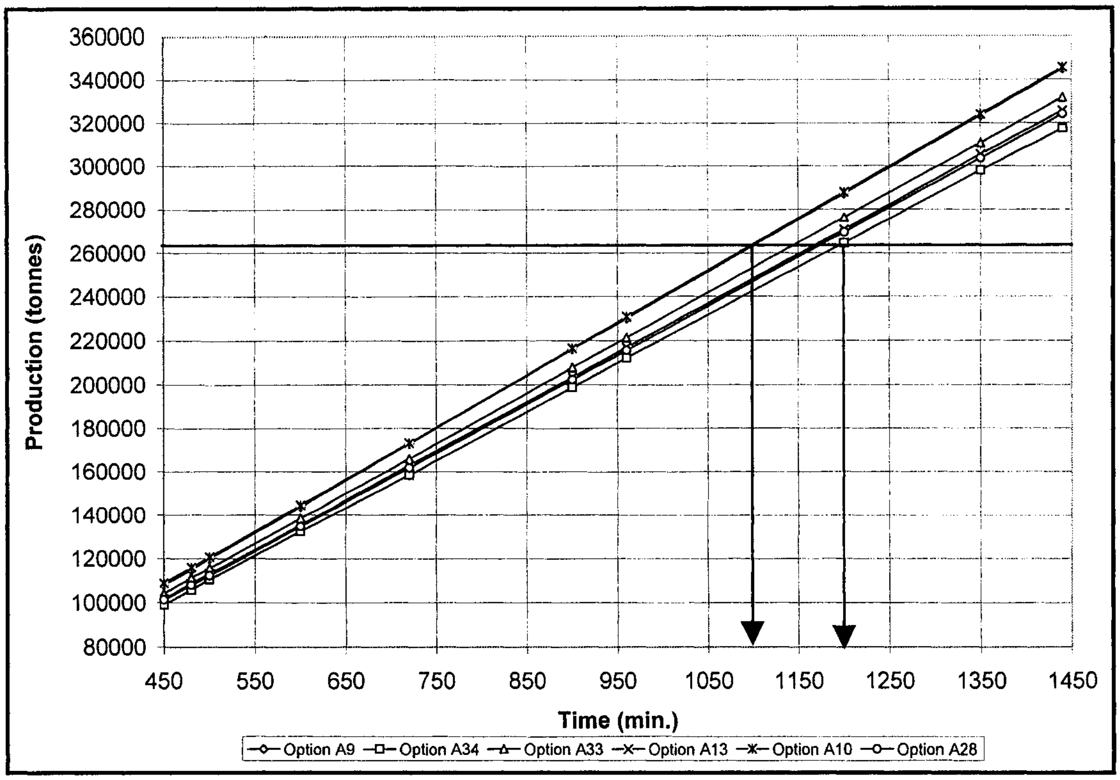


Fig. 6.16 Production of Best Fleet Options with One Server at Crusher

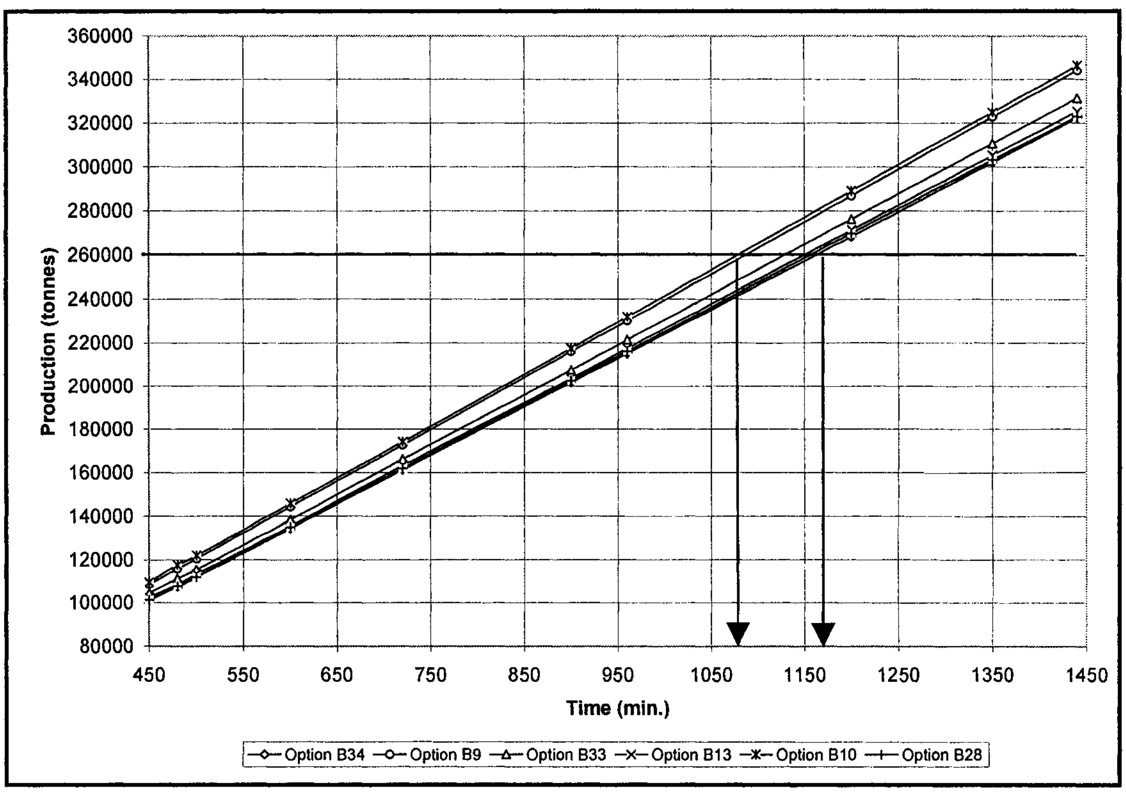


Fig. 6.17 Production of Best Fleet Options with Two Servers at Crusher

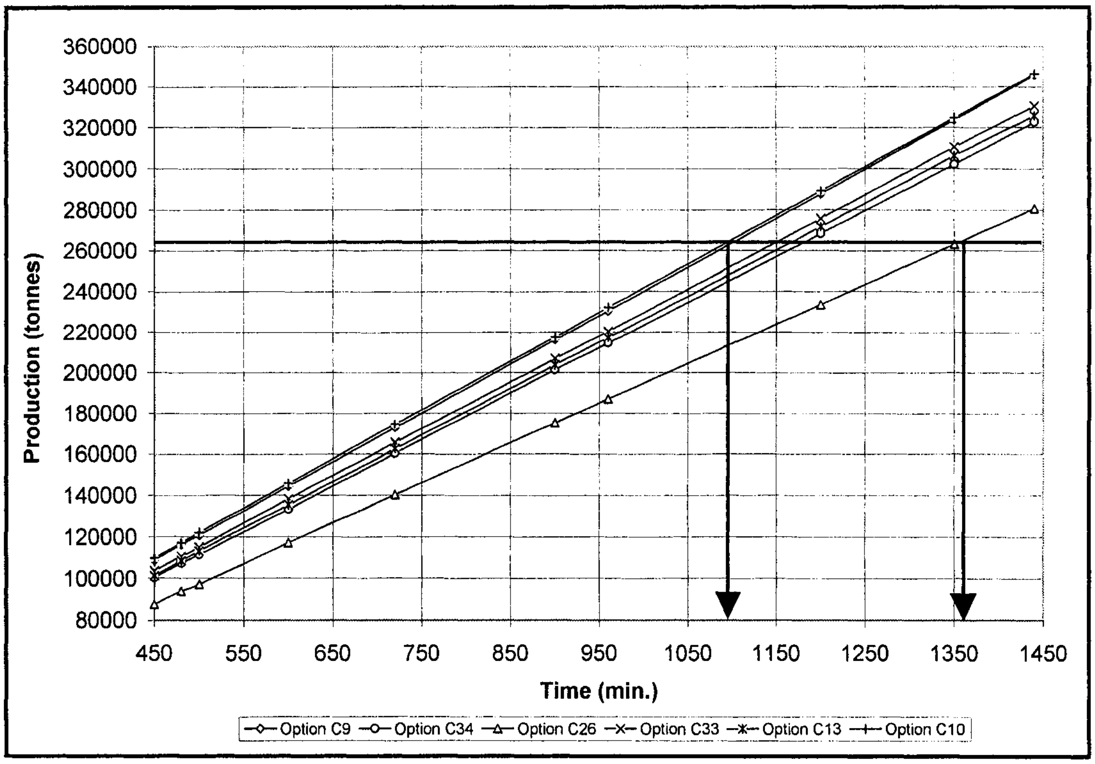


Fig. 6.18 Production of Best Fleet Options with Three Servers at Crusher

Allowing for combined truck mechanical availability and utilization, the mine must maintain fleet size of about twenty-four 360-tonne trucks to meet its production requirements. The results in Appendix D and Fig. 6.19 show that there was some level of unit costs reduction when the number of servers (dumping points) at the crusher was increased from 1 to 3. The unit production costs of the various truck fleets were generally lower when the number of servers at the crusher was increased to 3. This shows that in general it is more economic for the mine to provide facilities for three trucks to dump simultaneously at the crusher location to achieve the least costs per tonne compared with the use of one or two servers. As expected, the results in Appendix D and Table 6.3 confirm the fact that due to the economies of scale, it is often better to use larger capacity carrying trucks over smaller capacity trucks in all three scenarios (Sullivan, 1990). The average unit cost for the best 18 options (six in each scenario) is \$1.386/tonne (\$2.774/barrel).

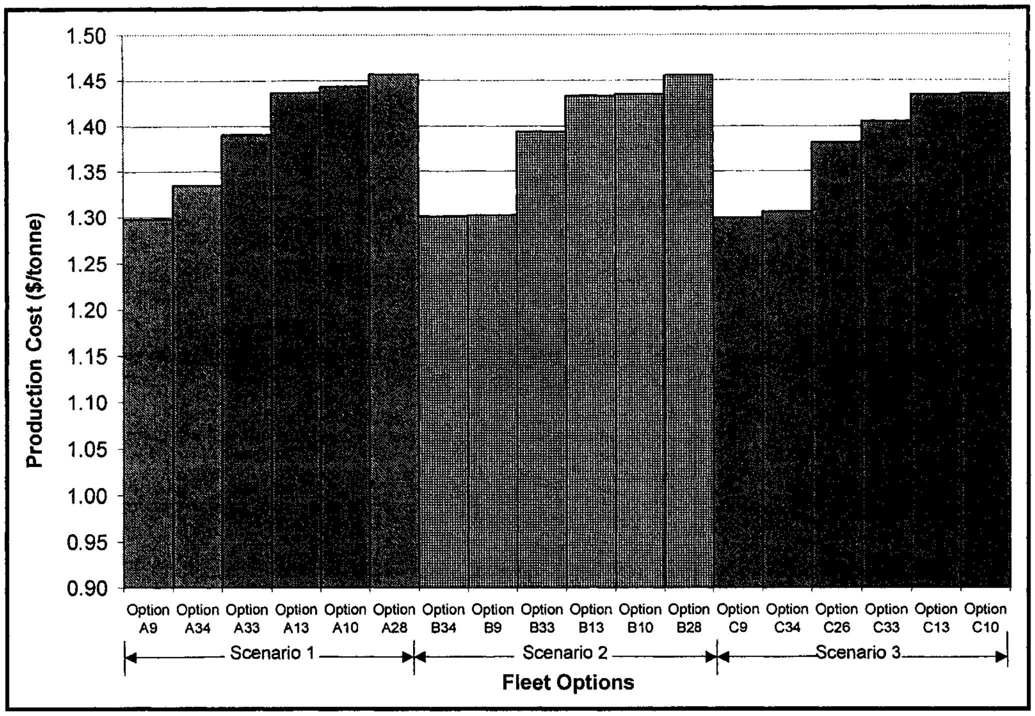


Fig. 6.19 Unit Production Costs of CMS with one to three servers at crusher

Table 6.3 Best CMS Options Based on Unit Cost ($/t)

Rank	Option	Combination of Trucks			Unit Cost ($/tonne)
		240 tonnes	320 tonnes	360 tonnes	
1	Option A9	0	0	12	1.297
2	Option C9	0	0	12	1.300
3	Option B34	0	6	3	1.301
4	Option B9	0	0	12	1.302
5	Option C34	0	6	3	1.306
6	Option A34	0	6	3	1.335
7	Option C26	6	3	0	1.382
8	Option A33	0	6	6	1.390
9	Option B33	0	6	6	1.394
10	Option C33	0	6	6	1.405
11	Option B13	6	0	6	1.433
12	Option C13	6	0	6	1.434
13	Option B10	0	0	15	1.435
14	Option C10	0	0	15	1.436
15	Option A13	6	0	6	1.436
16	Option A10	0	0	15	1.444
17	Option B28	0	9	3	1.456
18	Option A28	0	9	3	1.457
Average					1.386

6.3.2 Simulation of CycEx CBCS Option

Appendix F summarizes the sample results of the simulation of the CycEx CBCS option using Simphony software time periods ranging from 450 to 480 minutes for scenarios 1 to 5. Fig. 6.20 shows the productivity of the CycEx CBCS option after the first simulation run in a shift of 450 minutes. It shows that productivity increases to and stabilizes after reaching a maximum of about 200 t/hr. The maximum productivity is reached after about 100 minutes of the start of the shift. Due to setup time incorporated in the software, the simulation run is actually less than the specified 450 minutes.

Fig. 6.21 shows the production of the CycEx CBCS with time for periods ranging from 450 minutes to 1440 minutes allowing for a job operating efficiency of 70%. It is noted here that the standard case involves the use of O&K RH400 shovels with bucket capacities of 43.5 m^3 (77.29 tonnes) per cycle in combination with 1.8 to 2.2 m wide belt conveyors having average speeds of 5.2 m/s. In Scenario 2, the belt conveyor capacity has been increased by about 20% to 7,320 t/hr while in Scenario 3, the belt conveyor capacity has been decreased by about 20% to 4,880 t/hr. In Scenarios 4 and 5, the shovel digging capacity has been reduced by 20% and 40% respectively from the standard case.

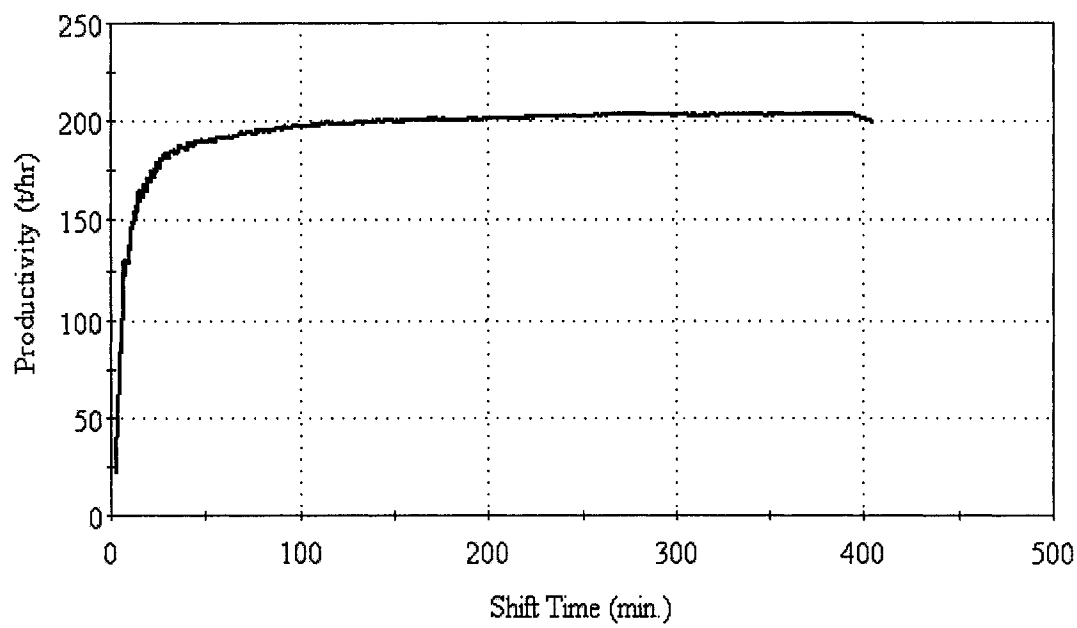


Fig. 6.20 Productivity for Run #1 in a shift of 450 minutes

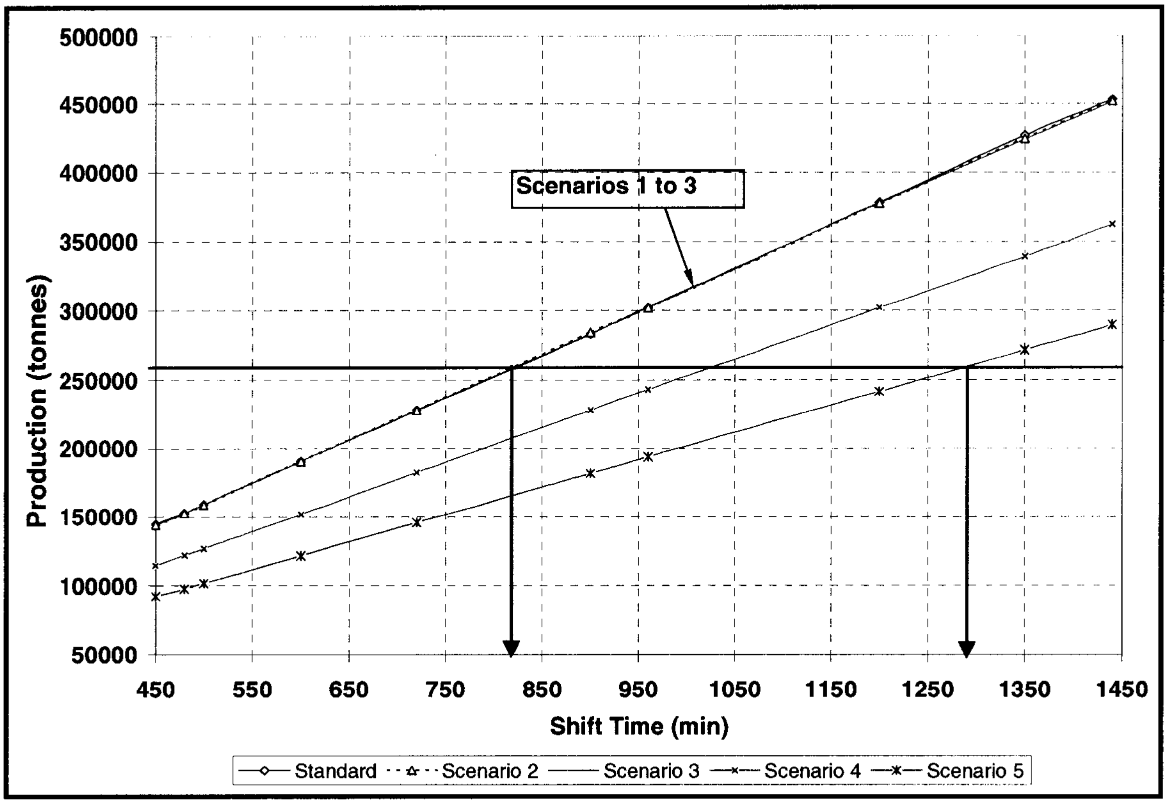


Fig. 6.21 Production of CycEx CBCS Option with Time

Fig. 6.21 shows that the targeted production of 262,000 t/day can be achieved with the CycEx CBCS option operating shift periods ranging from 820 min. to 1300 min. The curves for the standard case and those of scenarios 2 and 3 overlap implying that their overall productions are about the same. This indicates that varying the belt conveyor capacity by ±20% while maintaining the present O&K RH400 shovel dipper capacity of 43.5 m^3 does not affect the output of the CycEx CBCS option significantly. Thus conveyor belt carrying capacity will need to be increased significantly to cope with the digging capacity of the in the O&K RH400 in the CycEx CBCS option. However, changes in the shovel digging capacity (i.e. varying the dipper capacity) strongly affects the production of the CycEx CBCS option as shown in the curves for Scenarios 4 and 5.

Appendix G summarizes the pertinent data from the simulation of the CycEx CBCS option while Figs. 6.22 to 6.24 are the queue lengths, waiting times and system utilization of the CycEx CBCS option[§].

[§] Queue lengths and waiting times refer to times when there was choking of the crusher, clogging at apron feeder or on the belt conveyor wagons such that the shovel could not discharge its next load of oil sands.

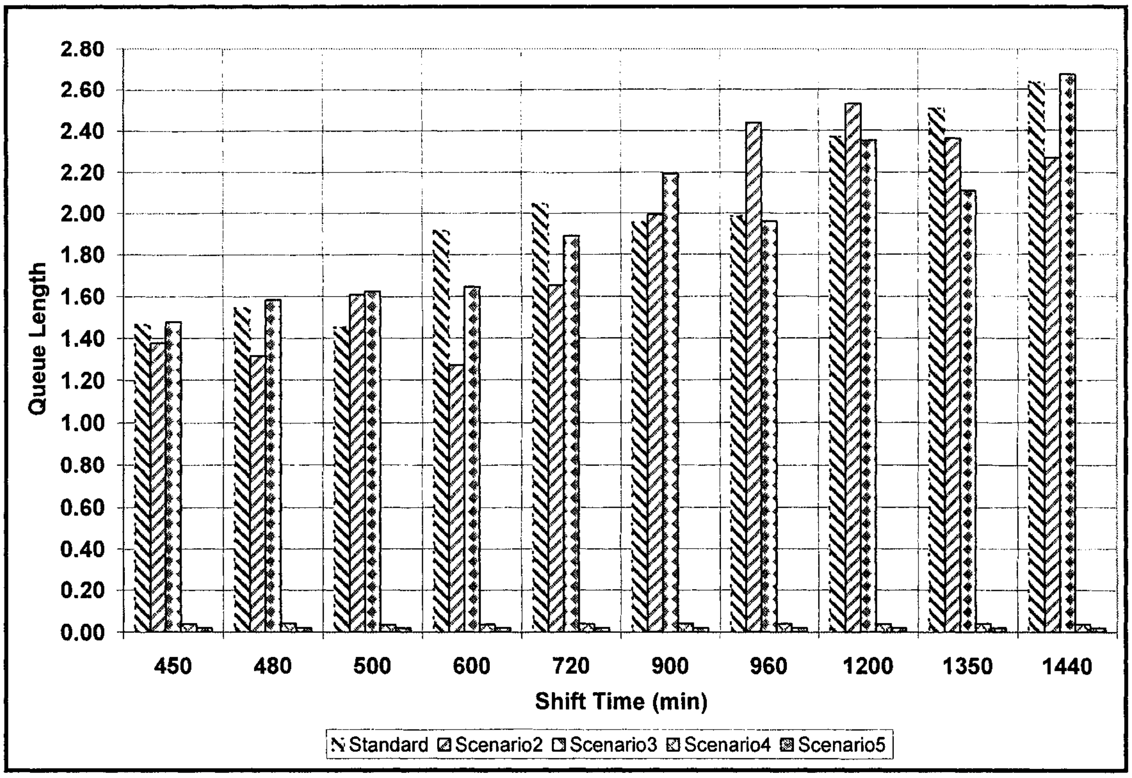


Fig. 6.22 Queue Lengths in the CycEx CBCS Option

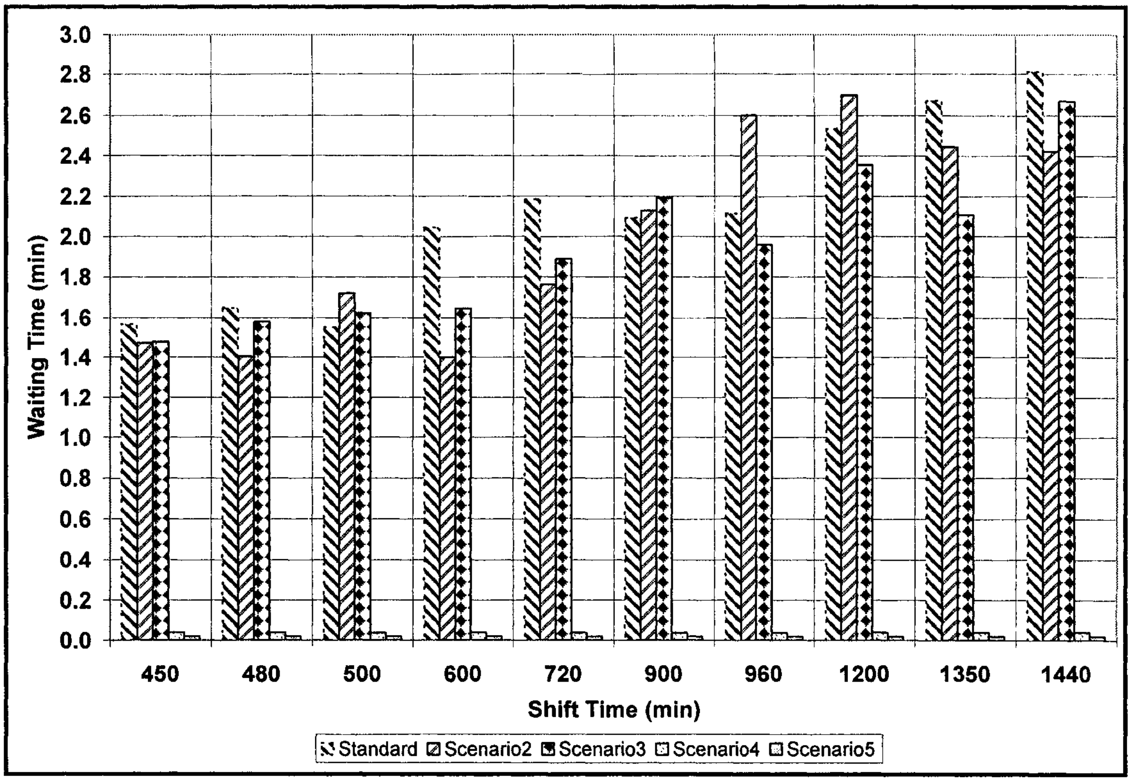


Fig. 6.23 Waiting Times of the CycEx CBCS Option

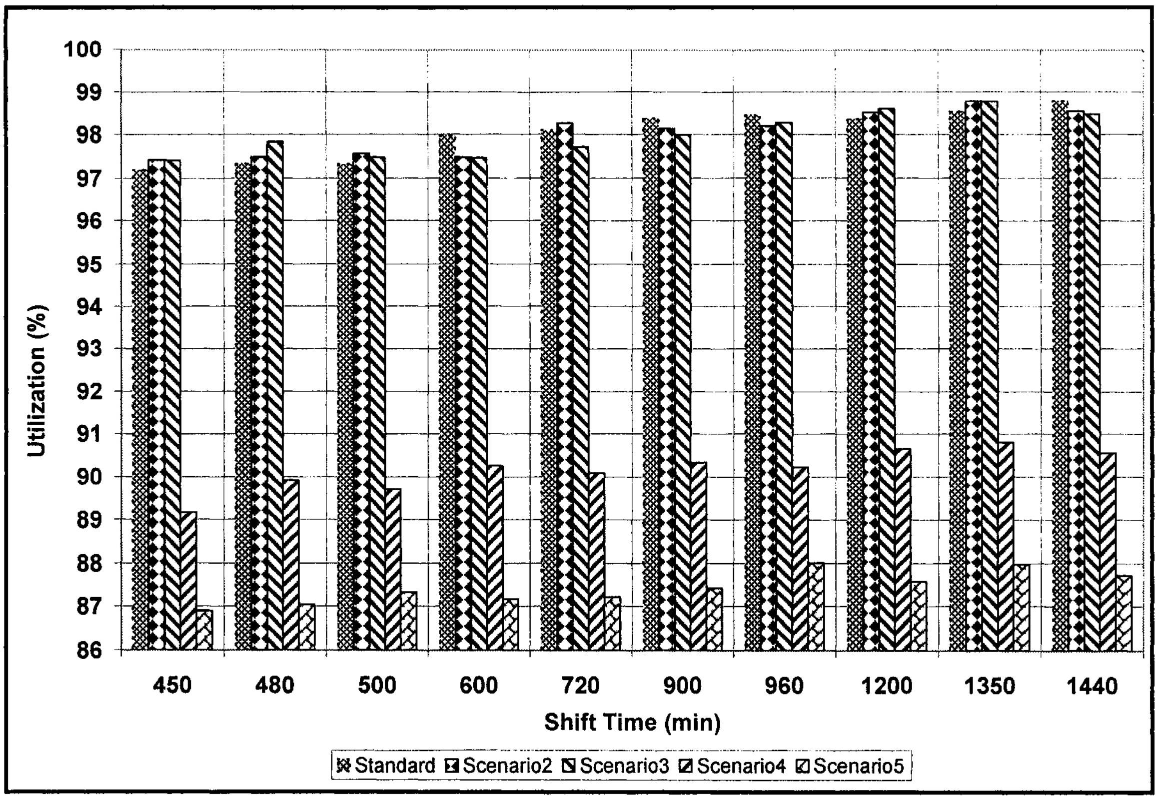


Fig. 6.24 Utilization of the CycEx CBCS Option

The simulated total production figures in column 2 of Appendix G do not take into consideration the frequent moves of the AFS train required to keep up with the shovel movements in the CycEx CBCS option. Accordingly, the total production figures have been multiplied by a job efficiency factor of 0.7. This is to take care of the delays in production due to the need to periodically relocate the hopper/crusher, belt conveyor wagons and the slurrification unit closer to the shovel (giving the estimated production figures in column 3 of Appendix G).

Clogging and delays in continuous mine production systems are usually common when the shovel digging capacity exceeds the joint capacity of the continuous system of apron feeder/crusher and belt conveyor wagons. This is confirmed by the low queue lengths and waiting times that resulted when the shovel digging capacity was reduced in Scenarios 4 and 5 (see Figs. 6.22 and 6.23). Fig. 6.24 shows that the utilization of the CycEx CBCS option was very high ($\geq$ 97.19%) when the O&K RH400 shovels were used in scenarios 1 to 3. Such high utilization levels indicate that the system was over-utilized (i.e. the shovel digging capacity exceeded that of the continuous portion of the system).

By reducing the shovel digging capacities, the system utilizations ranged between 86.9% and 90.65% in scenarios 4 and 5 (see Appendix G). These utilizations are in line with the general industry averages and allow some room for inherent delays in production due to lunch breaks, delays in changeovers at the end of shifts and the effects of inclement weather on production.

The short lengths of the bars for scenarios 4 and 5 in Figs. 6.22 to 6.24 indicate that the production of the CycEx CBCS system efficiency can be maximized by selecting shovels with digging capacities ranging from 39.57 to 68.02 t/cycle. This involves reducing the dipper capacity per pass of the shovels to between 23 and 40 m^3. However, if the present 43.5 m^3 dippers on the O&K RH400 shovels are to be employed, then the crusher and conveyor belt capacities have to be increased by more than 20% to match the shovel capacity.

Fig. 6.25 is a plot of the production from the best six CMS fleets and the five Scenarios of the CycEx CBCS option. The curves for Scenarios 1 to 3 as well those of Options A9, B9 and C9, and Options A34 and C34 overlap.

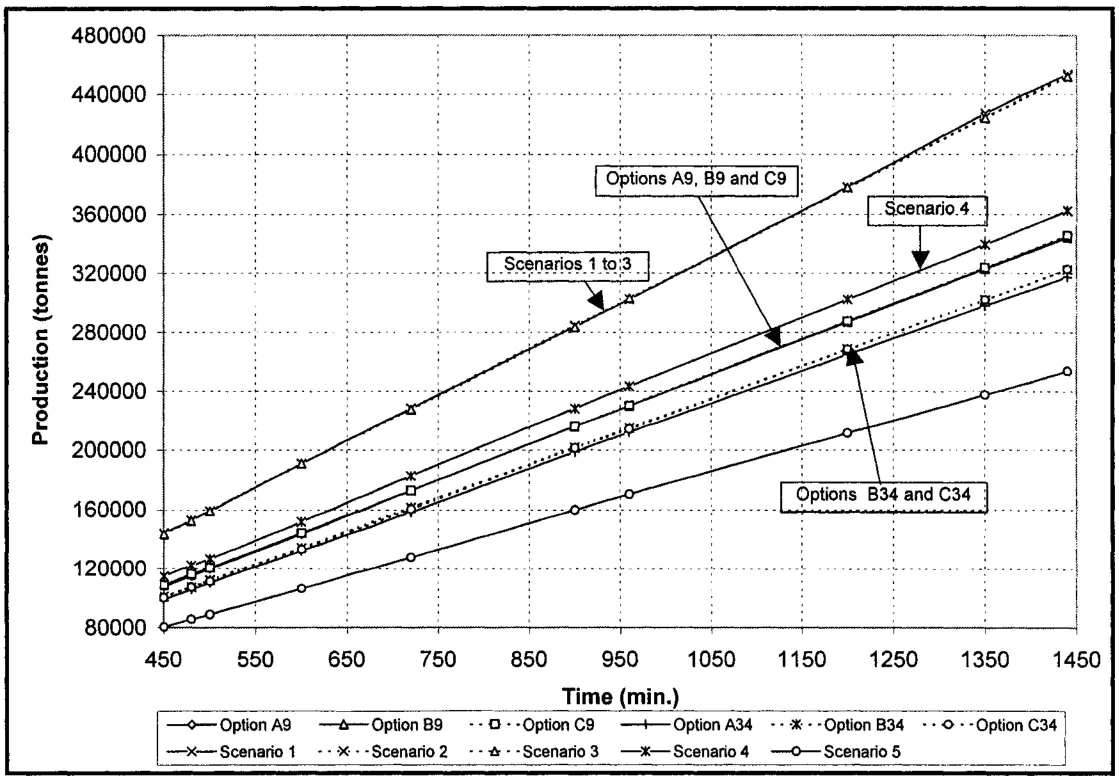


Fig. 6.25 Production of Best Six CMS Fleets and CycEx CBCS Scenarios

From Fig. 6.25, the production from Scenarios 1 to 4 of the CycEx CBCS option are all greater than all the different fleet sizes in the CMS option. It is only when the shovel production capacity is reduced by 40% from the standard (as in Scenario 5) that the production of the CycEx CBCS option falls below the production in the CMS options. Thus the CycEx CBCS option is better suited at meeting the daily production target of 262,000 tonnes than the CMS option.

6.4 Physical Simulation and Dynamic Motion Analysis

The physical simulation and dynamic motion analysis of the CycEx CBCS option are done in this section here. Two problems related to the dynamics of CycEx CBCS system are solved. The first problem is the resolution of the required vectors of joint torques τ_i (i = 2, 3, 4) for a given trajectory point θ_i (i = 2, 3, 4) of the system (Frimpong et al., 2003). The second problem is the calculation of how the mechanism moves under application of a set of joint torques τ_i (i = 2, 3, 4).

One objective of simulating the CycEx CBCS system is to visualize system motion and generate the nominal joint torques for the joint actuators. Simulation is based on the theoretic models in Sections 4.6 and 5.5 and virtual prototype models shown in Fig. 5.10. The CycEx CBCS model is assumed to be moving on a hard homogeneous oil sand terrain. Because the model is analyzed as 3D solid model, the gravity effects are automatically considered when component materials are specified. But the angular velocities of joints need to be applied for driving model. The initial conditions are: static friction coefficient = 0.75, dynamic friction coefficient = 0.7, l_i = 20 m, $\omega_0 = \dot{\omega}_0 = v_0 = 0$ and $\dot{v}_0 = g = (g_x, g_y, g_z)^T$, where $|g|$ = 9.8062 m/s^2. The conveyor belt is considered as a solid-woven belt and wagon material is considered to be steel (Frimpong et al., 2003). As the CycEx CBCS model moves through one complete cycle, a forward and return motion of mobile crusher lasts 80 m within 172,800 s, and corresponds to every belt-wagon angle range from θ_i = 0 to 60° and angular velocity of 1.39 × 10^{-3} d/s. The joint torques are described in terms of time.

Fig. 6.26 shows the joint torques (from joints 2 to 4) applied to joint actuators during two complete cycles. The cyclic changes of torque versus time are established in the figure. For joints 2 and 4, one full motion cycle lasts 15,600s, followed by a uniform torque rise during the next 70,800 s, then a uniform torque return during the next 70,800 s, and a second cycle of torque lasting the final 15,600 s. However, for joint 3, one complete motion cycle consists of a cycle lasting 15,600 s, followed by a uniform torque increase during the 51,900 s, a uniform torque decrease during the 18,900 s, then a uniform torque increase and decrease during next 18,900s and 51,900 s respectively. A second cycle occurs lasting the final 15,600 s after the first cycle. At any given time, the torque decreases from joint 4 through joint 2 to joint 3 (Frimpong et al., 2003). The results indicate that a maximum torque of 1.95×10^8 Nm occurs on joint 4 and joint 3 is subjected to the least torque of about 8×10^7 Nm as the system moves through one full cycle. This means that the component parts of the CycEx CBCS option must be designed to withstand torques up to maximum torque of 1.95×10^8 Nm.

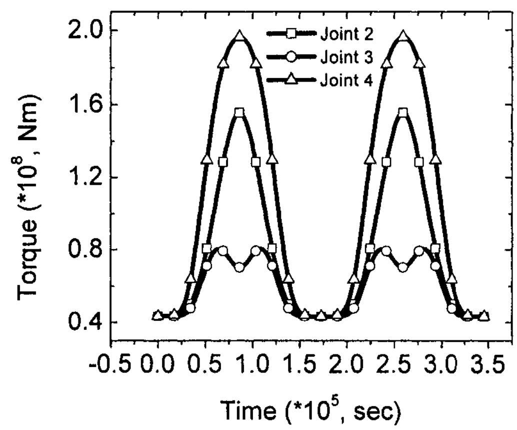


Fig. 6.26 Torque vs Time for CycEx CBCS Option

(Source: Frimpong et al., 2003).

Another objective of simulating the CycEx CBCS option is to analyze the system motion for given torque vectors. The initial parameters used for this simulation are the same as those used in above simulation. Assuming torques, τ_2, τ_3 and τ_4 applied to joints 2 to 4, are as shown in Fig.6.26, the angle θ_i, is determined in terms of time.

Fig. 6.27 shows the cyclic variation the angle θ_i versus time for two cycles. For a full cycle of motion, θ_i increases linearly with time from 0 to 86,400 s, while it decreases linearly with time from 86,400 to 172,800 s. It undergoes a minimum of 0° at time t = 0 corresponding to the initial static equilibrium position. It has a maximum value of 60° at time t = 86,400 s, which corresponds to the end position. The value of θ_i varies from 0° to 60°.

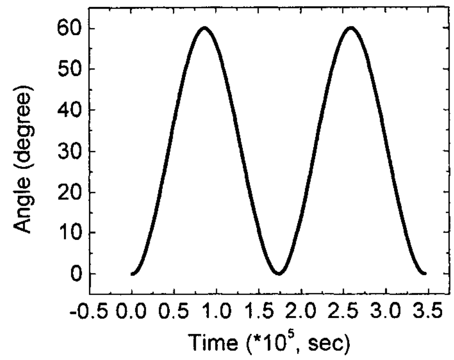


Fig. 6.27 Change in Angle θ_i with time for CycEx CBCS Option

(Frimpong et al., 2003)

Finally, the projection angle of the CycEx CBCS option increased almost linearly up to a maximum of 60° in about 23.50 hr. This shows that the train of belt conveyor wagons are in the fully extended position every 23.50 hr and have to be relocated closer to the shovel to avoid over-stretching the components and subjecting them to higher than designed torques and projection angles.

6.5 Economic Analysis of Mining Options

Mining projects normally involve huge levels of capital outlay with their attendant high investment risks. Accordingly, all new mining projects or modifications to existing projects have to be economically evaluated to assess their viabilities and whether they add value to the company. Economic analysis is one of the best tools for evaluating and comparing different projects or investments options. Various economic evaluation criteria are commonly used alone or in combination to determine the acceptability or attractiveness of projects and to aid in the selection of the best investment ventures from a number of options. Some of the economic evaluation criteria used in this work are the Net Present Value (NPV), Profitability Index (PI), Internal Rate of Return (IRR) and Discounted Payback Period (DPBP). This section presents the details of the economic analysis of the CMS and CycEx CBCS options. Even though the cost of production per tonne of the two options may be used to determine the better option, it is much better to assess both options using the four evaluation economic criteria outlined earlier on to give a global picture of the two options. As well, it is necessary to asses the effect of changes in the discount rate at various percentages because with most new operations where many variables are not well known, most investors often apply high discount rates at the start of operations. Over the years as more and more information is gathered and various parameters known to some high level of certainty, these discount rates are reduced to reflect the level of confidence in the information gathered on the project. In addition, it is also necessary to assess the impact of provincial and federal taxes on the viability of the project at various tax levels up to 100% because due to the unpredictable nature of mineral prices and other economic indices on the world as well as the frequent changes political regimes in some states and provinces, there is the likelihood that some governments will tend to impose higher than expected taxes on operating mining companies in order to balance their accounts and to meet increasing public demand for the provision of certain facilities and services. Thus a comprehensive economic and risk analysis is required on the all possible scenarios ranging from the best case scenario to the worst case scenario for proper comparison of the various mining options and to make informed decisions on them with a higher degree of confidence.

Table 6.4 summarizes some of the input data used in the economic analysis of the CMS and CycEx CBCS options. In Syncrude, about one barrel of oil (Sweet Syncrude Blend) is obtained for every two tonnes of ore mined. The production cost per tonne of the CMS option is normally distributed with a mean of $1.386 and a standard deviation of $0.07. The production cost per tonne of the CycEx CBCS option is also normally distributed with a mean of $0.779 and a standard deviation of $0.023. The various costs used in the calculations were assumed to vary by ±25% of their mean values. The Double Declining Balance (DDB) method of depreciation is used by Syncrude. The depletion allowance is taken as the minimum of 5% of gross revenue or 10% of Pre-Capital Cost Allowance (PreCCA) while the Bank of Canada average exchange rate of the US dollar to the Canadian dollar for the past 7 years was obtained is 1.486.

Appendix H contains the details of the economic analysis on the CMS and CycEx CBCS options using information supplied by the Syncrude authorities, quotations from equipment manufacturers, Federal and Provincial sources, and Bank of Canada webpages (Anon., 2003e; Anon., 2003f).

Table 6.4 Input Data for Economic Analysis

Item	CMS Option	CycEx CBCS Option
Daily Production (tonnes/day)	262000	262000
No. of Working days/yr	365	365
Price/barrel (US$)	29.80 ± 2.67	29.80 ± 2.67
Cost per 320-tonne truck	$2,600,000.00	-
Cost per 360-tonne truck	$3,000,000.00	-
Shovel (O&K RH200)	$10,000,000.00	$10,000,000.00
Crusher (mobile)	$22,000,000.00	13,000,000.00
Belt conveyor wagons (20 m)	-	2,500,000.00
Mobile transfer conveyor	-	9,800,000.00
Federal tax rate (%)	15 ± 5.3	15 ± 5.3
Provincial tax rate (%)	30 ± 8.2	30 ± 8.2
Interest rate (%)	6.2 ± 1.4	6.2 ± 1.4
Royalties (%)	5.00	5.00
Currency exchange rate	1.486 ± 1.325	1.486 ± 1.325
Discount rate (%)	15	15
Inflation rate (%)	2.82 ± 1.26	2.82 ± 1.26

Table 6.5 summarizes the results of the economic analysis conducted on the two mining options in this work. The results in Table 6.5 show that both mining options are viable with high net present values (≥ $\$3.20 \times 10^{10}$), profitability indices (> 19%) and internal rate of returns (> 29.02%) and extremely short discounted payback periods (≤ 3.24 months). An allowance of 25% of the cost is given for contingencies.

Table 6.5 Summary of Economic Analysis of Mining Options

Economic Parameter	Mining Option		Ratio
	CMS	CycEx CBCS	(CBCS/CMS)
Net Present Value (NPV) ($)	3.20×10^{10}	4.06×10^{10}	1.27
Profitability Index (PI) (unitless)	19.37	43.37	2.24
Internal Rate of Return (IRR, %)	29.02	33.37	1.15
Discounted Payback Period (DPBP, yr)	0.27	0.16	0.59

From Table 6.5, the CycEx CBCS option is clearly more economically viable than the CMS option. Its NPV is 1.27 times that of the CMS option. The PI and IRR of the CycEx CBCS option are respectively 2.24 and 1.13 times that of the CMS option. As well, the

CycEx CBCS option has almost half the DPBP of the CMS option. Against a discount rate of 15% set by the company, the CycEx CBCS option is clearly the better option to invest in.

Table 6.6 summarizes the total operating costs of the CMS and CycEx CBCS options obtained from calculations. The results show that the CMS option has an operating cost of $1.386 per tonne ($2.774/barrel) while that of the CycEx CBCS option is $0.779/tonne ($1.558/barrel). Thus the unit operating cost of the CMS option is about 1.78 times that of the CycEx CBCS option.

Table 6.6 Summary of Operating Cost of CMS and CycEx CBCS Options

Type of Equipment	No. of Units	Maintenance Cost ($\$ \times 10^6$)	Operator Cost ($/min.)	Total No. Operators	Total Operator Cost ($\$ \times 10^6$)	Total Cost ($\$ \times 10^6$)
CMS Option						
Trucks (360 t unit)	24	14.40	1.60	36	30.27	55.84
Shovel (O&K RH200)	6	22.80	2.20	18	20.81	54.52
Crusher	1	7.66	1.60	12	10.09	22.18
Total Operating Cost ($\$ \times 10^6$)			13.25 ± 2.26			
Operating Cost per tonne ($/tonne)			1.386 ± 0.07			
CycEx CBCS Option						
Belt conveyor wagons (20 m)	18	6.75	1.6	9	7.57	17.90
Mobile transfer conveyor	2	2.94	1.6	6	5.05	9.98
Hydrotransport Pipelines	1	0.15	1.6	6	5.05	6.49
Shovel (O&K RH200)	6	3.80	2.2	12	13.87	22.09
Mobile Crusher & Slurrification unit	1	1.80	1.6	15	12.61	18.02
Total Operating Costs ($\$ \times 10^6$)			74.49 ± 4.66			
Operating Cost per tonne ($/tonne)			0.779 ± 0.023			

6.6 Risk Characterization and Sensitivity Analysis

Every mining investment venture faces many risks in terms of the huge levels of capital outlay required, the timing of cash inflows and outflows. Other factors that increase the risk include frequent and unpredictable changes in the market prices of commodities and equipment, type of legislations on taxes, laws on environmental protection, the prevailing political environment within the country in which the project is located, and the general consciousness of the people. Risk analysis is required to quantify the level of uncertainty in the venture and to assess the likelihood of the venture achieving certain targets under varying economic and technical conditions. What-if analyses are also conducted to determine how the NPV is affected by changes in price of oil, discount rate, unit operating cost and level of provincial and federal taxes using the Toprank module of the @Risk software. Evaluation of the effect of factors such as either a 25% drop in the price of the major commodity or the introduction of a new processing technology that could lead to a reduction in the unit operation cost by 50% on the profitability of the venture has to be done. Risk characterization and sensitivity analyses of the CMS and the CycEx CBCS options are done in this section to determine the viability of both mining options (Anon., 1997). The following are the details of the risk and sensitivity analyses done on the two mining options.

6.6.1 Risk Charaterization of Mining Options

Table 6.7 summarizes the major statistical factors obtained from the risk characterization of the CMS and CycEx CBCS options. The results show that the value of the NPV of the CMS option at the 5th percentile is \$2.39 $\times$ 10^{10} while that at the 95th percentile is \$4.01 $\times$ 10^{10}. For the CycEx CBCS option the NPV varies between \$3.03 $\times$ 10^{10} (at the 5th percentile) and \$5.09 $\times$ 10^{10} (at the 95th percentile).

Table 6.7 Risk Characterization of Mining Options

Statistic Parameter	CMS Option	CycEx CBCS Option
Minimum ($\$ \times 10^{10}$)	1.19	1.71
Maximum ($\$ \times 10^{10}$)	5.19	6.38
Mean ($\$ \times 10^{10}$)	3.20	4.06
Standard deviation ($\$ \times 10^{9}$)	4.92	6.25
Variance ($\$ \times 10^{19}$)	2.42	3.90
Skewness	-1.82×10^{-04}	-6.19×10^{-04}
Kurtosis	3.002365	2.992046
Mode ($\$ \times 10^{10}$)	3.23	4.07
5% Percentile ($\$ \times 10^{10}$)	2.39	3.03
10% Percentile ($\$ \times 10^{10}$)	2.57	3.26
90% Percentile ($\$ \times 10^{10}$)	3.83	4.86
95% Percentile ($\$ \times 10^{10}$)	4.01	5.09

Figs. 6.28 and 6.29 show the risk characterization of NPV and the probability of success of the two mining options. Fig. 6.28 shows that for the CMS, the mean NPV is \$3.20 $\times$ 10^{10} and a standard deviation of \$4.92 $\times$ 10^{9}. The minimum and maximum values of the

NPV are \$1.19 $\times$ 10^{10} and \$5.19 $\times$ 10^{10} respectively. For the CycEx CBCS option, the mean, standard deviation, minimum and maximum values of the NPV are \$4.06 $\times$ 10^{10}, \$6.25 $\times$ 10^{9}, \$1.71 $\times$ 10^{10}, \$6.38 $\times$ 10^{10} respectively. Thus the NPV of the CycEx CBCS option has a larger variance than that of the CMS option. From Fig. 6.29, there is a 15% probability that the NPV of the CMS and CycEx CBCS options will be $\leq$ \$2.69 $\times$ 10^{10} and $\leq$ \$3.42 $\times$ 10^{10} respectively.

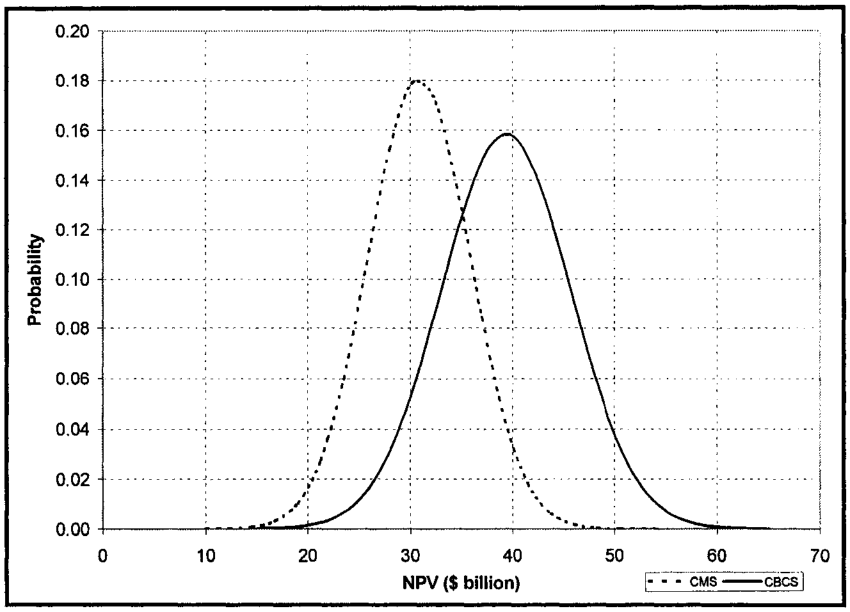


Fig. 6.28 Risk Characterization of NPV of Mining Options

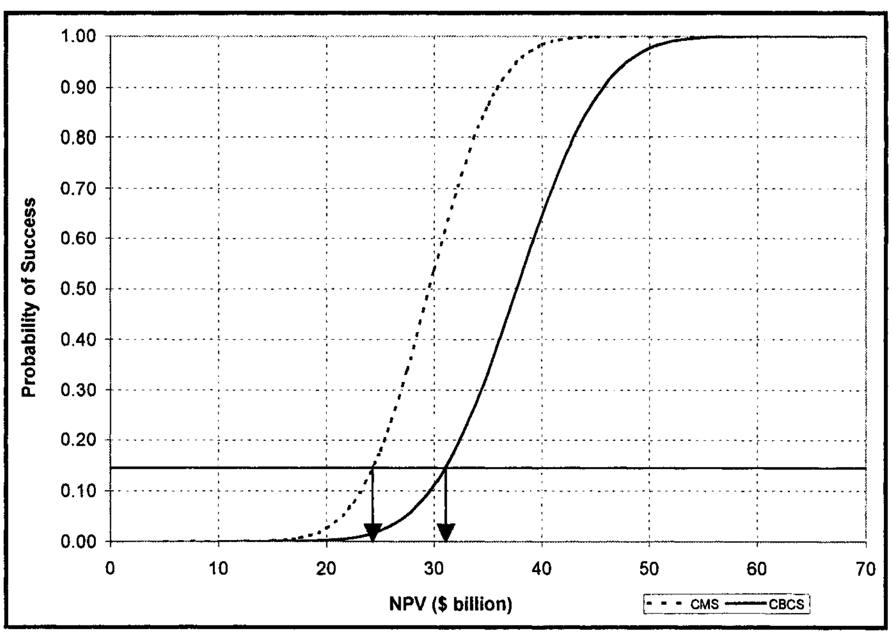


Fig. 6.29 Probability of Success of Mining Options

6.6.1.1 Risk Characterization with Varying Oil Prices

Fig. 6.30 shows the relationship between the mine NPV and the price of oil per barrel on the market. By extrapolation, it shows that when the mine employs the CMS option, the NPV is zero when the price of oil is about US$8.65/barrel. In the same way, when the mine employs the CycEx CBCS option, the NPV is zero when the price of oil is US$2.71/barrel. This indicates that the CMS option requires higher oil prices to remain viable than the CycEx CBCS option.

Fig. 6.31 shows the relationship between IRR and the price of oil per barrel. It shows that the IRR of the CycEx CBCS option is greater than that of the CMS option at all oil prices. As oil prices are presently averaging US$30/barrel, the CycEx CBCS option clearly has a higher return on investment than the CMS option for oil sands mining at Syncrude.

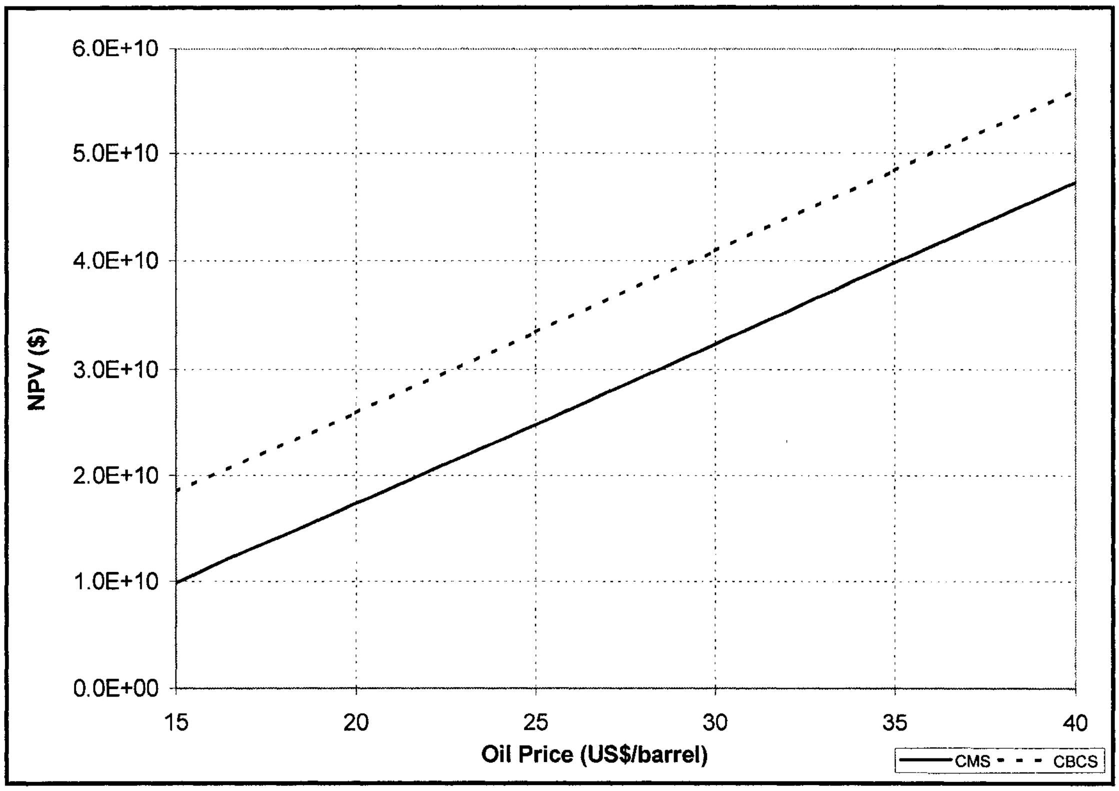


Fig. 6.30 Rate of Change of NPV with Price of Oil

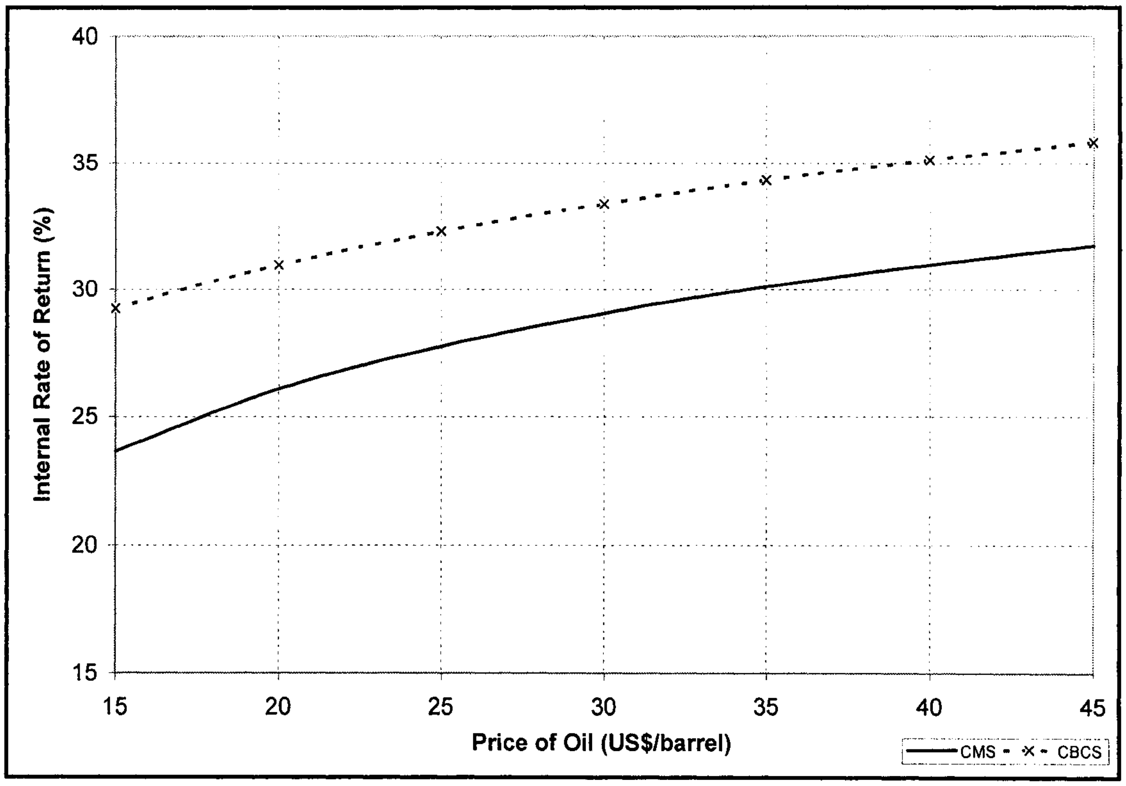


Fig. 6.31 Rate of Change of IRR with Oil Price

Figs. 6.32 and 6.33 are the risk characterization of the NPV of the mine at varying oil prices using the CMS and CycEx CBCS options respectively. Fig. 6.32 shows that at an oil price of US$15, the variance in the NPV is much smaller than those at higher oil prices. As expected, both Figs. 6.32 and 6.33 show that as the oil price increases, the NPV also increases (i.e. oil price has a positive impact on the NPV). Figs. 6.34 and 6.35 are the cumulative probability functions (CPFs) of the mine NPV at various oil prices. From Fig. 6.34, there is 15% probability that the NPV for the CMS option will be less than or equal to $7.62 billion (at an oil price of US$15/bll) and $39.75 billion (when the oil price is US$40/bll). From Fig. 6.35, there is 15% probability that the NPV of the CycEx CBCS option will be ≤ $14.20 billion (when oil price is US$15/bll) and ≤ $46.99 billion (when oil price is US$40/bll). This means that there is 85% chance that the NPV will exceed those values at the stated oil prices in both mining options. Appendix I summarizes the risk characterization of CMS and CycEx CBCS options at different oil prices using @Risk.

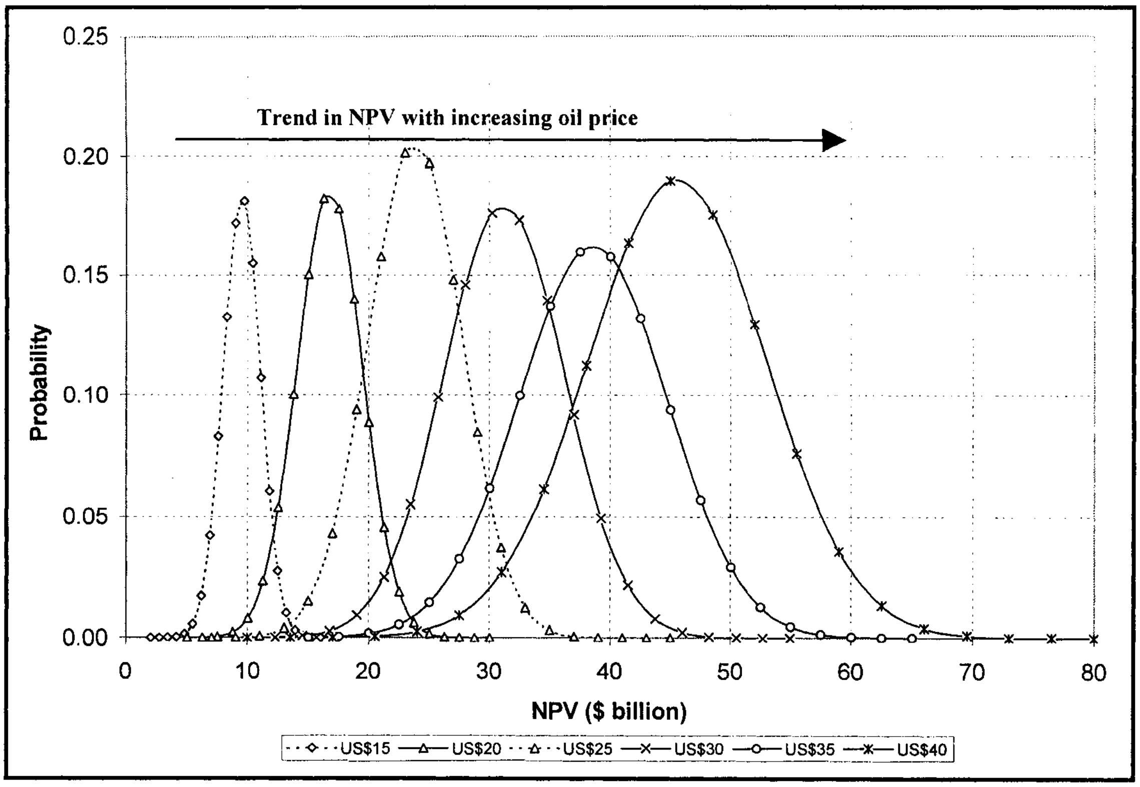


Fig. 6.32 Risk Characterization of NPV at varying Oil Prices (CMS option)

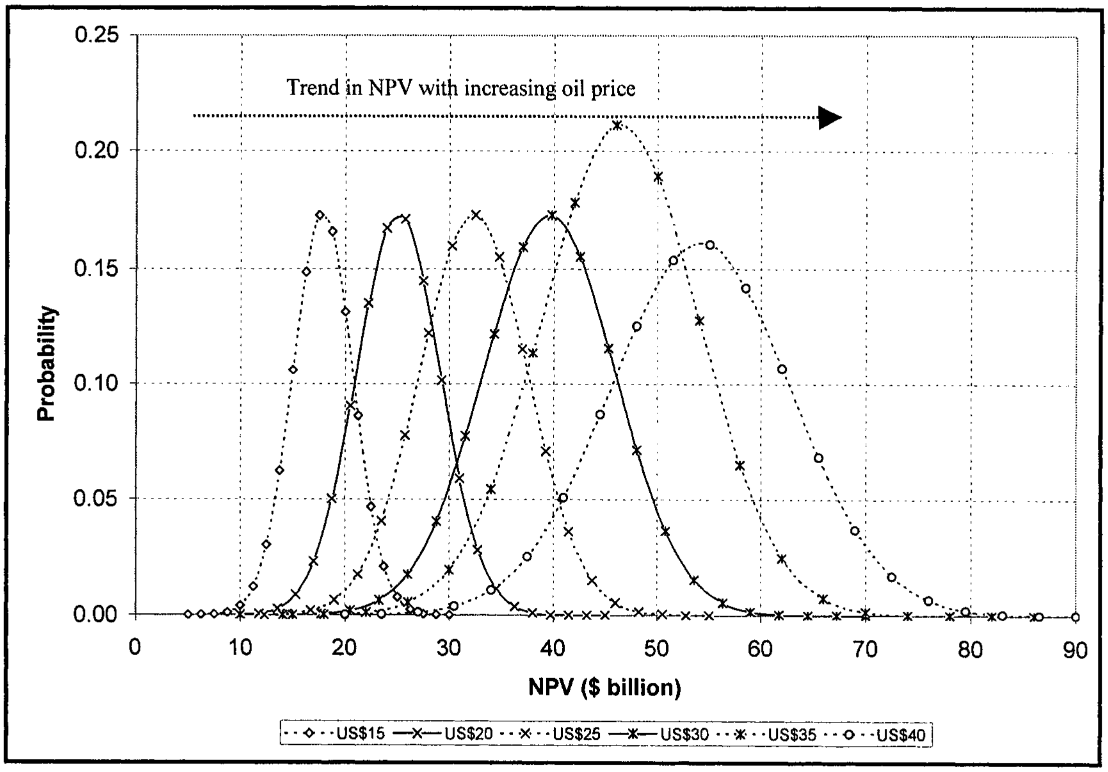


Fig. 6.33 Risk Characterization of NPV at varying Oil Prices (CycEx CBCS option)

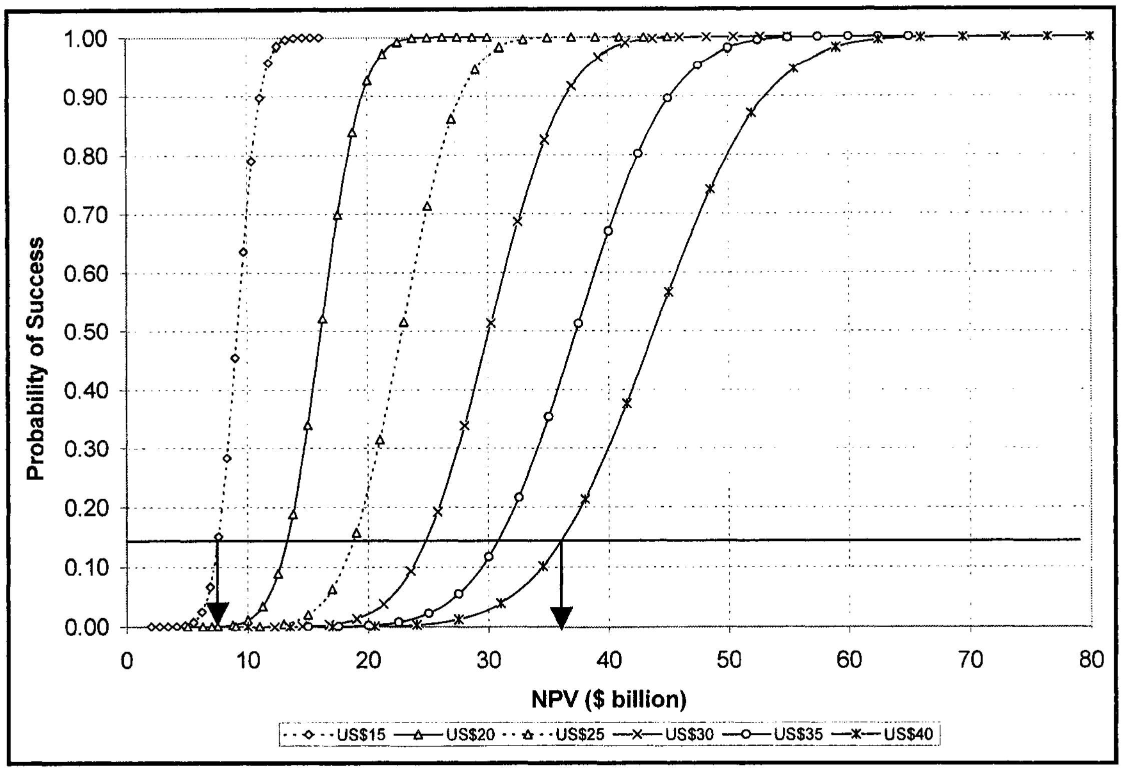


Fig. 6.34 Probability of Success of CMS option at different Oil Prices

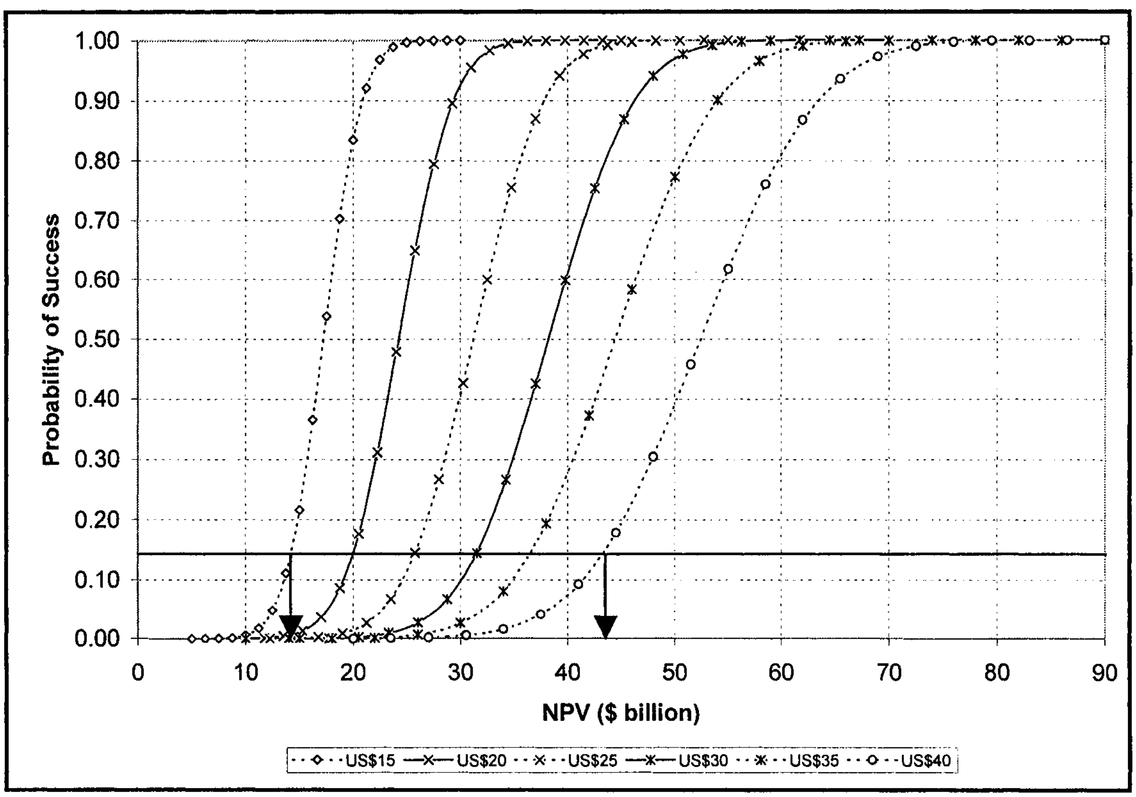


Fig. 6.35 Probability of Success of CycEx CBCS Option at different Oil Prices

6.6.1.2 Risk Characterization with Varying Operating Costs

Fig. 6.36 plots the NPV of the CMS and CycEx CBCS options at varying operating costs. The gradients of the curves show that the NPV of the CMS option is affected more by changes in the operating cost than the CycEx CBCS option. As well, Fig. 6.36 shows that the NPV of the CMS option is zero at an operating cost of \$10.62/barrel while that of the CycEx CBCS option is \$21.68/barrel. Thus the CycEx CBCS option has a wider of operating cost range than the CMS option.

Figs. 6.37 and 6.38 are the CPFs of the mine NPV at various operating costs for the CMS and CycExc CBCS options respectively. From Fig. 6.37, there is ≤ 15% probability that the NPV for the CMS option will fall between \$2.21 billion (at an operating cost of \$10/barrel) and \$35.61 billion (when the operating cost is \$0.10/barrel) for the CMS option. In the same way, from Fig. 6.38, there is ≤ 15% probability that the NPV of the CycEx CBCS option will fall between \$2.96 billion (when the operating cost is US\$20/barrel) and \$36.52 billion (when the operating cost is US\$0.10/barrel) for the CycEx CBCS option. This shows that the probabilities of success or failure of the two mining options with respect to operating cost are almost the same.

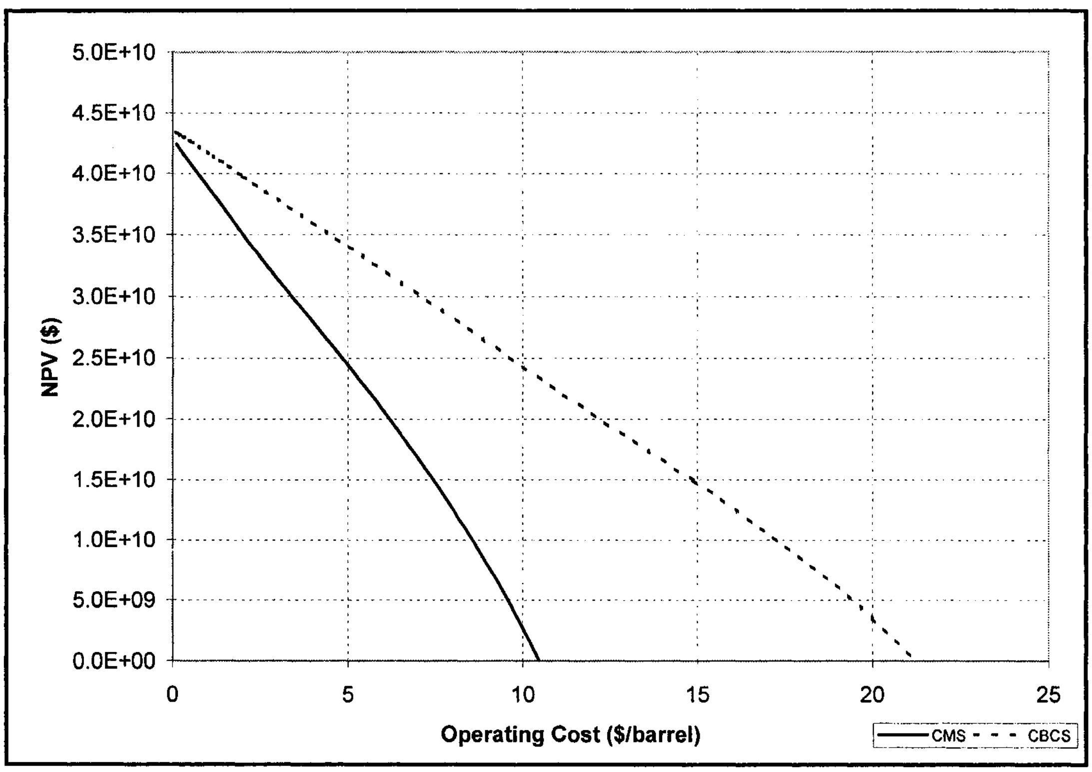


Fig. 6.36 NPV vs. Operating Costs (CMS option)

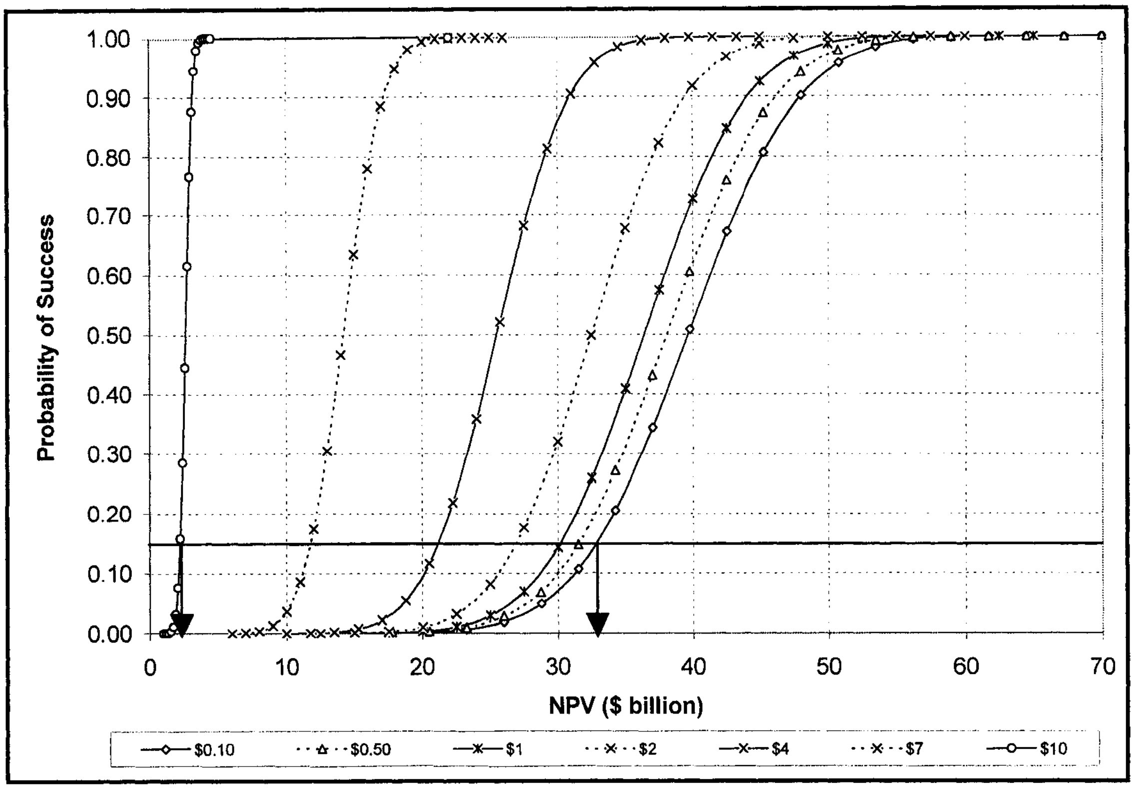


Fig. 6.37 Probability of Success of CMS option at different Operating Costs

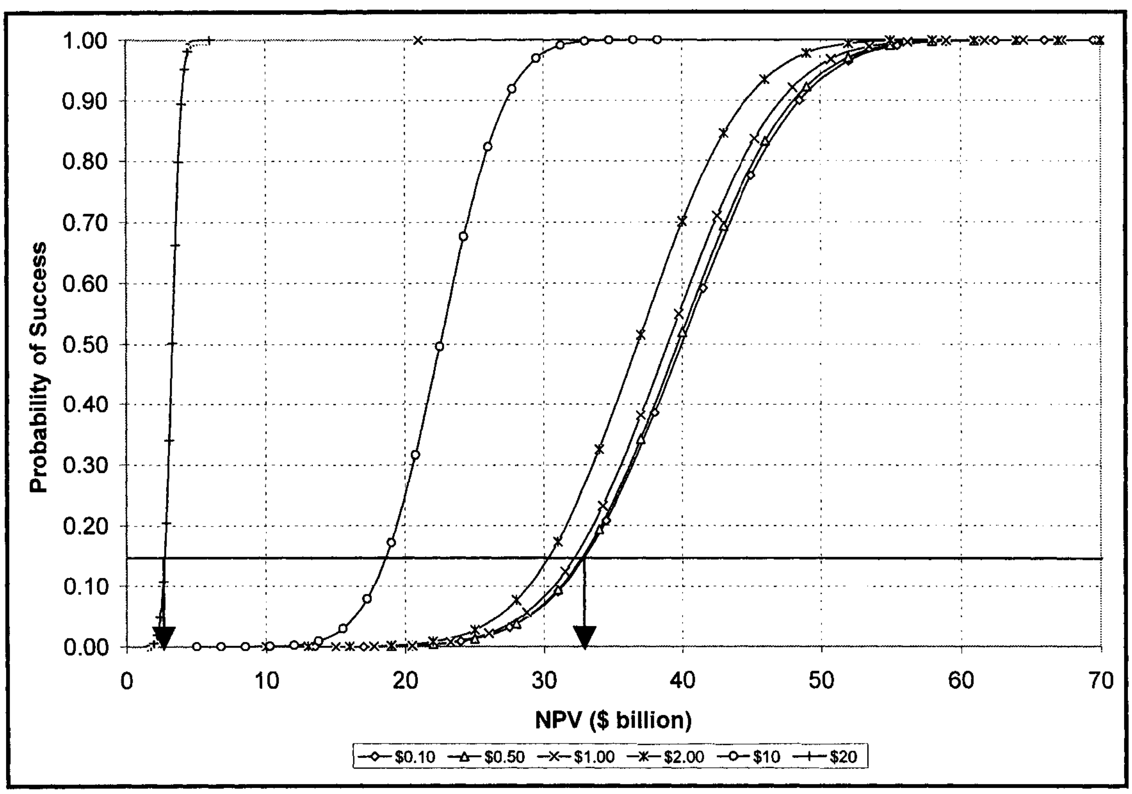


Fig. 6.38 Probability of Success of CycEx CBCS Option at different Operating Costs

It is also worth noting in both Figs. 6.37 and 6.38 that the CPFs of the NPV at an operating costs above \$10/barrel and \$20/barrel respectively have very narrow ranges. This implies that the probability of success at the 15th and 85th percentiles were almost equal in both cases at those operating costs. This indicates that at operating costs above \$10/barrel, the NPV of the CMS option falls into an unstable and risky economic environment in terms of chances of success. For the CycEx CBCS option, this occurs at operating costs above \$20/barrel. Thus it would be safer to maintain the unit operating costs in the CMS and CycEx CBCS options below \$10/barrel and \$20/barrel respectively.

6.6.1.3 Risk Characterization with Varying Tax Rates

Fig. 6.39 shows the relationship between NPV and taxes (sum of provincial and federal taxes). It shows that the level of provincial and federal taxes has a negative impact on the NPV of the mine under the two mining options. The slightly larger gradient of the CycEx CBCS option indicates that it is affected more by the level of taxes than the CMS option. Fig. 6.39 also shows that the difference between the NPV of the CMS and CycEx CBCS options becomes larger and larger as the tax rate decreases.

Table 6.8 summarizes the results of the probability of success at 15th percentile of the two mining options at different tax rates. It shows that at a low tax rate of 5%, there is 85% chance that the NPV of the CMS option will exceed $\$45.15 \times 10^9$ while that for the CycEx CBCS option will exceed $\$57.14 \times 10^9$. When the tax rate is increased to 60%, there is 15% chance that NPV of the CMS option will be less than or equal to $\$20.08 \times 10^9$ while that of the CycEx CBCS option will be $\leq \$25.53 \times 10^9$.

Table 6.8 Chance of Success of Mining Options at 15th percentile at different Tax Rates

	NPV (\$ billion)	
Provincial and Federal Tax Rate	CMS Option	CycEx CBCS Option
5%	45.15	57.14
10%	42.87	54.26
15%	40.59	51.39
30%	33.76	42.77
Base Case (45%)	26.92	34.15
60%	20.08	25.53
85%	8.69	11.17
100%	1.85	2.55

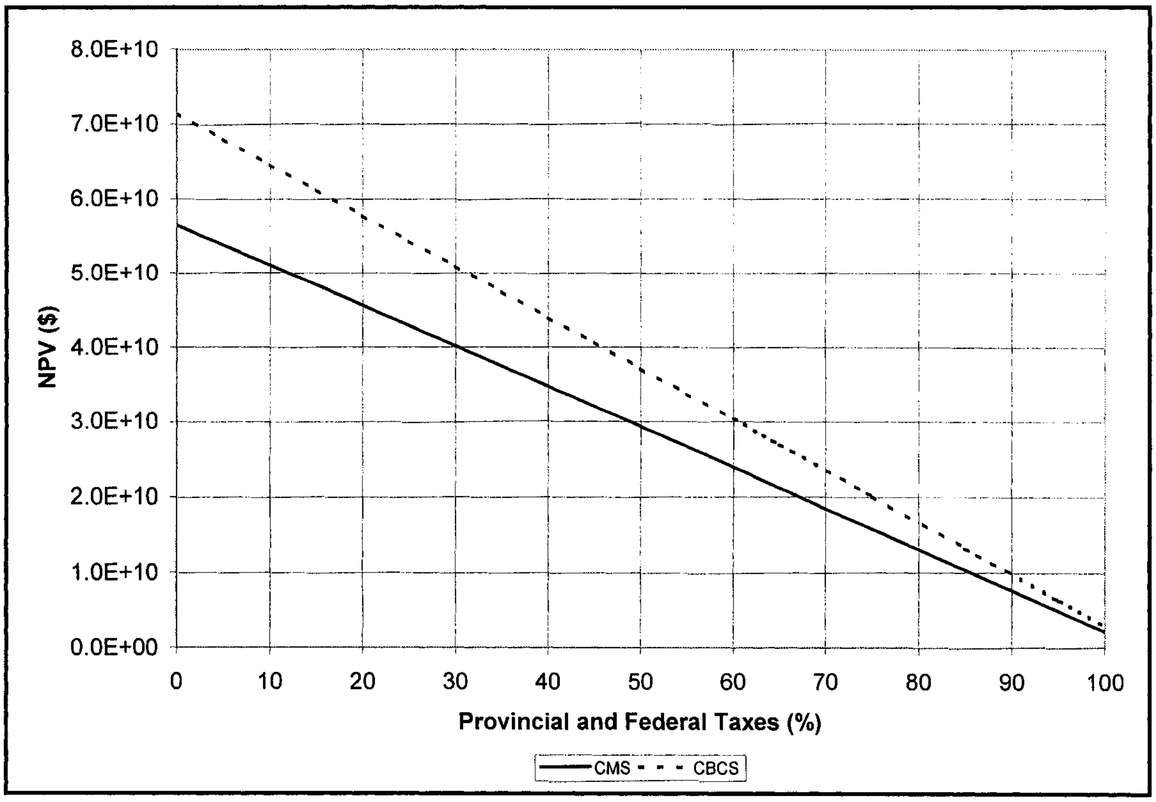


Fig. 6.39 NPV vs. Provincial and Federal Taxes for Mining Options

6.6.2 Sensitivity Analysis of Mining Options

The TopRank module of @Risk software was used to conduct sensitivity analyses of the various input parameters on the NPV of the mine if it employs any of the two mining options. Using the autovary function in Toprank, the base values of the input parameters that directly affect the NPV of the mine (e.g. oil price, production capacity, discount rate, level of provincial and federal taxes) are varied between ±25% to determine how sensitive the NPV is to changes in each of the input variables. All the input variables are assumed to be normally distributed.

Figs. 6.40 and 6.41 are the tornado graphs of the results of what-if analysis of the various input parameters on the NPV of the CMS and CycEx CBCS options respectively. Fig. 6.40 shows that the discount rate, daily ore production, scheduled number of working days per year, price of oil/barrel, exchange rate, production time and, provincial and federal taxes are some of the input parameters that have the most significant effects on the NPV of the CMS option. For example, when the value of the discount rate is varied by ±25%, it has between –19.09% and +19.09% effect on the NPV of the CMS option while a similar change in the price of oil affects the NPV by ±6.96%. The number of 36-tonne trucks has the least effect of ±1.40% on the NPV of the CMS option.

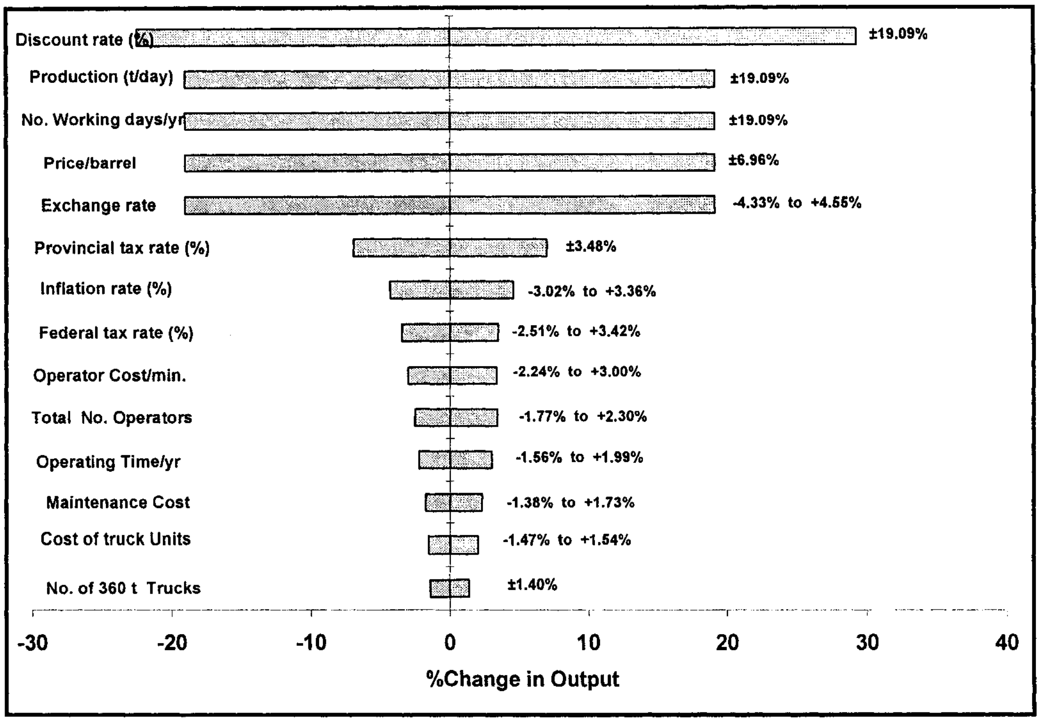


Fig. 6.40 Tornado Graph of NPV of CMS Option

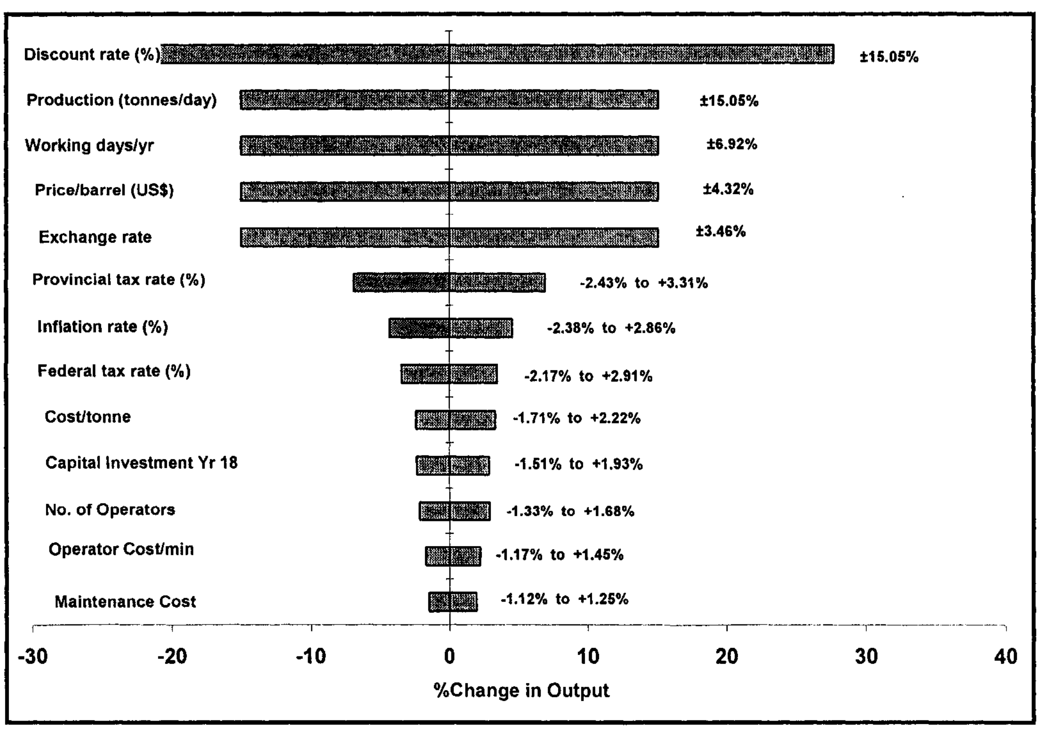


Fig. 6.41 Tornado Graph of NPV of CycEx CBCS Option

In Fig. 6.41, input variables that have the greatest effect on the NPV are the discount rate, the production capacity, number of scheduled working days, oil price, and the exchange rate between the Canadian and US dollars. Thus with the CycExc CBCS option, when the value of the discount rate is varied by ±25%, it affects the value of the NPV most by -±15.05%. The input parameters that have marginal effects on the NPV of the CycEx CBCS option are the maintenance cost, operator cost/min., number of operators, and the level of capital investments made within the last three years of the mine life.

Table 6.9 summarizes the results of the top twelve most significant inputs on the NPV of the CMS and CycEx CBCS options. It shows the percentage changes in the maximum and minimum NPV values at the values of the input values given in columns 4 and 6. Thus when the discount rate is 12.94%, the maximum NPV value increases by 29.20% for the CMS option.

When the discount rate is 17.06%, there is a 22.52% decrease in the minimum value of the NPV for the CMS option. For the CycEx CBCS option, when the discount rate is 12.94%, the value of the maximum NPV increases by 27.65%. As well, when the discount rate is 17.06%, there is 20.99% decrease in the minimum value of the NPV for the CycEx CBCS option. While a ±25% in the operator cost of shovels affects the NPV of the CMS option by between -2.24% and +3.00%, a similar change in the shovel operator costs only affects the NPV of the CycEx CBCS option between -1.17% and +1.45%. This means that even though the same number of shovels are employed in mining in both options, the operator costs of the CycEx CBCS option has a smaller negative impact on the NPV than that of the CMS option.

Figs. 6.42 and 6.43 are the spider graphs of the CMS and CycEx CBCS options respectively. Spider graphs show the percentage change in the base value of the input variables against the percentage change in the base value of the output variables. Both figures show that the daily ore production capacity, number of scheduled working days/year, production time, exchange rate, oil price per barrel have positive impact on the NPV (i.e. an increase in any of them leads to an increase in the NPV of the mine). However, the discount rate, operating costs, inflation rate, federal and provincial tax rates negatively impact the mine NPV.

Table 6.9 Sensitivity of NPV to Marginal Input Variation

Mining Option – CMS					
Rank	Name	% Change in Maximum NPV	When Input Value is	% Change in Minimum NPV	When Input Value is
#1	Discount rate (%)	+29.20%	12.94	-22.52%	17.06
#2	Production (tonnes/day)	+19.09%	297912.50	-19.09%	226087.50
#3	No. Working days/yr	+19.09%	415.03	-19.09%	314.97
#4	Price/barrel (US$)	+19.09%	33.88	-19.09%	25.72
#5	Exchange rate	+19.09%	1.69	-19.09%	1.28
#6	Provincial tax rate (%)	+6.96%	25.89	-6.96%	34.11
#7	Inflation rate (%)	+4.55%	3.21	-4.33%	2.43
#8	Federal tax rate (%)	+3.48%	12.94	-3.48%	17.06
#9	No. of trucks required	+1.40%	20.71	-1.40%	27.29
#10	Total Operating Cost	+1.49%	12.08	-1.20%	15.92
#11	Truck Operator Cost/min.	+1.31%	1.38	-1.31%	1.82
#12	Trucks Operating Time/yr	+1.31%	453555.70	-1.31%	597644.30
Mining Option – CycEx CBCS					
#1	Discount rate (%)	+27.65%	12.94	-20.99%	17.06
#2	Production (tonnes/day)	+15.05%	297912.50	-15.05%	226087.50
#3	No. Working days/yr	+15.05%	415.03	-15.05%	314.97
#4	Price/barrel	+15.05%	33.88	-15.05%	25.72
#5	Exchange rate	+15.05%	1.69	-15.05%	1.28
#6	Provincial tax rate (%)	+6.92%	25.89	-6.92%	34.11
#7	Inflation rate (%)	+4.54%	3.21	-4.32%	2.43
#8	Federal tax rate (%)	+3.46%	12.94	-3.46%	17.06
#9	Cost of Belt conveyor wagons	+0.43%	2157323	-0.43%	2842677
#10	No. Belt conveyor wagons	+0.43%	15.53	-0.43%	20.47
#11	Operator cost (shovel)/min.	+0.24%	1.90	-0.24%	2.50
#12	Working time (shovel)	+0.24%	453555.70	-0.24%	597644.30

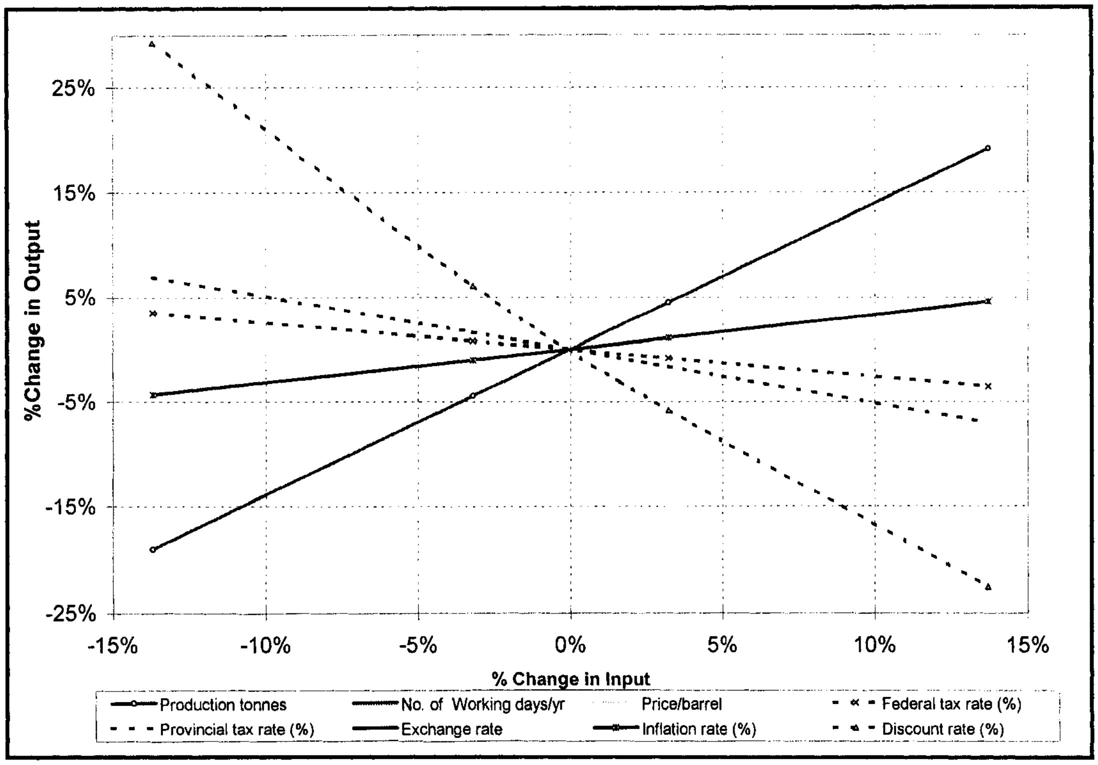


Fig. 6.42 Spider Graph for NPV of CMS Option

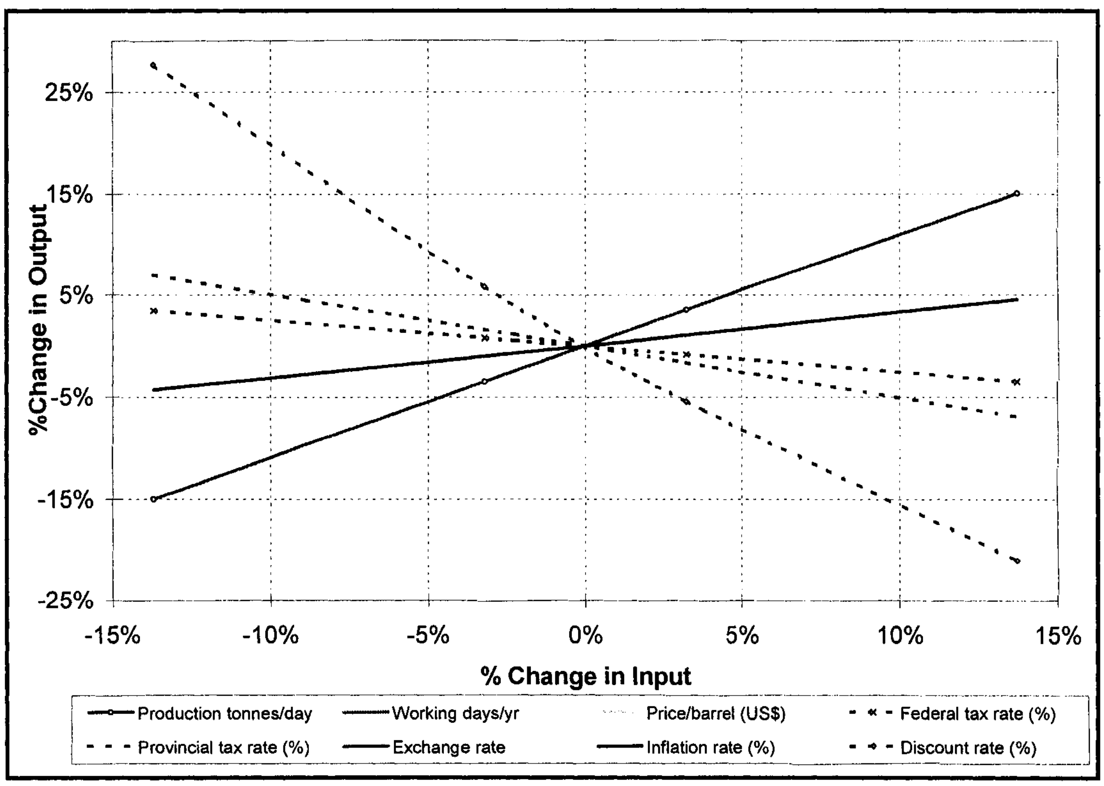


Fig. 6.43 Spider Graph for NPV of CycEx CBCS Option

From the gradients of the plots in Figs. 6.42 and 6.43, the discount rate has a greater negative impact on the NPV than the provincial tax rate. As well, the scheduled production time and capacity have greater positive impact than the exchange rate and price of oil. Thus the input variables that greatly affect the NPV in both mining options are the discount rate, operating costs, production rate, oil price per barrel exchange rate, and the federal and provincial tax rates.

6.7 Summary

In this chapter, performance simulation modeling and analysis was done on the two mining methods. The geometric volumes of circular and elliptical pits, dynamics of the material flow within the pits and the changes in the dimensions of the pits in real-time using mathematical modeling and parabolic partial differential equations were calculated. Simulation of the mining operations as well as comprehensive economic, risk characterization and sensitivity analyses were conducted on the two mining options. The computer software packages used in the modeling, verification, experimentation and simulations included Matlab, Visual SLAM, Simphony, ADAMS and @Risk. It has been shown that the geometric mine design and multi-bench material flow characterization of the AFS which involve continuous pit expansion and materials handling operations within the pit are best handled using PDE models with their associated boundary conditions.

The results of the geometric volume calculations show that at the scheduled production rate of 262,000 tonnes/day, the ore within benches 1 to 3 will be completely mined out between 6.25 and 8.03 years with circular pit configurations. With elliptical pit configurations, the ore in the three benches will be excavated between 6.39 and 8.17 years. It takes between 0.08 to 3.79 days to mine the ore materials within each incremental pushback on bench #1. As well, the rate of expansion in the circular pit dimensions on bench #1 using the CMS option was determined as 0.048 m/hr while that using an elliptical pit on bench #1 was 0.046 m/hr.

The simulated results of the CMS option show that the optimum truck fleets are Option A9 (operating over 1,094.85 min.), Option B34 (operating over 1094.85 min.) and C9 (operating over 1092.49 min.). The first six best options involve the use of either 360-tonne trucks only or in combination with 320-tonne trucks all with either one or two simultaneous dumping sites at the crusher location. The average unit cost for the best 18 options (six in each scenario) is \$1.386/tonne (\$2.774/barrel).

With the CycEx CBCS option, the simulated results show that the targeted production of 262,000 t/day can be achieved by operating shift periods ranging from 820 min. to 1300 min. The unit cost of employing the CycEx CBCS option was calculated to be \$0.779/t (\$1.558/barrel). The simulation results for the two mining options show that the production from Scenarios 1 to 4 of the CycEx CBCS option are all higher than all the options in the CMS option. Thus the CycEx CBCS option is better suited at meeting the daily production target of 262,000 tonnes than the CMS option in most cases.

The results of cost analysis on both mining options show that the NPV of the CMS option is $\$3.20 \times 10^{10}$ while that for the CycEx CBCS option is $\$4.06 \times 10^{10}$. The profitability index for the CMS and CycEx CBCS options are 19.37 and 43.37 respectively. The internal rate of return of the CMS option was calculated to be 29.02% while that of the CycEx CBCS option is 33.37%. Discounted payback periods for the CMS and CycEx CBCS options are 0.27 years and 0.16 years respectively. These economic parameters show that the CycEx CBCS option is more economically viable than the CMS option.

The results of risk characterization of the two mining options show that there is 85% probability that the NPVs of the CMS and CycEx CBCS options will be greater than $\$2.69 \times 10^{10}$ and $\$3.42 \times 10^{10}$ respectively. The results of sensitivity analyses on the mining options show that the daily ore production capacity, number of scheduled working days/year, production time, exchange rate and oil price per barrel have the largest positive impact on the NPV. However, the discount rate, federal and provincial tax rates, and inflation rate negatively impact the mine NPV. Thus the input variables that significantly affect the NPV significantly in both mining options are the discount rate, production rate, oil price per barrel, exchange rate, and the federal and provincial tax rates. In general, changes in input parameters such as discount rate, scheduled mine production capacity and time, price of oil per barrel and total operating costs had less impact on the NPV of the CycEx CBCS option than the CMS option.

CHAPTER 7

CONCLUSIONS AND RECOMMENDATIONS

7.0 Conclusions

This research initiative is a pioneering effort to develop continuous ellipsoid and spheroid processes for characterizing the geometrical layouts and material extraction process using the novel AFS technology. The research associated with the AFS technology occupies a frontier within the paradigm of economic extraction of the Athabasca oil sands deposits. Based on the work carried out and the results obtained, the following conclusions are drawn:

1. First and foremost, this research investigation has achieved the objectives defined in Section 1.3 of this thesis report. Conceptual AFS models for oil sands slurry transportation from the production faces have been outlined as a precursor for the detailed study. Mathematical models governing the evolution of the surface mine layouts associated with one AFS method have been developed using parabolic partial differential equations (PDEs) and their boundary conditions. Continuous simulation of these processes is carried out based on the underlying stochastic processes and field constraints within Visual SLAM with AweSIM, Matlab, Simphony and the ADAMS simulation environments. The continuous processes are also compared with the current cyclic method at Syncrude. Production sequence and schedules of material flow from multi-bench, multi-face mining operations are developed and tested for various operating paradigms of the two mining systems.
2. Comprehensive review and analysis of the potential AFS options showed that the Cyclic Excavator Conveyor Belt Control System (CycEx CBCS) is the most promising and it formed the basis for modeling, simulation, experimentation and analysis of the study.
3. The continuous ellipsoid and spheroid extraction problems associated with continuous surface mining have been solved for the AFS technology using parabolic PDEs and their boundary conditions. The dynamic volumes of materials as a result of the dynamic evolution of the mining layouts are generated for strategic and tactical mine plans. The AFS equipment geometry and space requirements are also generated to fit within the space constraints of the environment. This analysis has been comprehensively carried out for a surface mine layout with three operating benches, currently operating at Syncrude. The rate of expansion in the circular pit dimensions on bench #1 using the CMS option was determined as 0.048 m/hr while that using an elliptical pit on bench #1 was 0.046 m/hr.

4. The kinematics and dynamic evaluation of the AFS system requires robust, reliable and effective production equipment. The required torque, displacement and projection angles required to maintain a stable operating unit, have been generated for the AFS machinery. The results show that the component parts of the CycEx CBCS option must be designed to withstand torques up to 1.95×10^8 Nm. As well, the projection angle of the CycEx CBCS option increased almost linearly up to a maximum of 60^o in about 23.50 hr. This shows that the train of belt conveyor wagons are in the fully extended position every 23.50 hr and have to be relocated closer to the shovel to avoid over-stretching the components and subjecting them to higher than designed torques and projection angles.
5. Dynamic simulation of the various mining systems shows that the optimum truck fleets for the CMS option are Option A9 (operating over 1094.85 min.), Option B34 (operating over 1094.85 min.) and C9 (operating over 1092.49 min.). These fleets involve the use of either 360-tonne trucks only or in combination with 320-tonne trucks all with either one to three simultaneous dumping sites at the crusher location. For all scenarios in the CycEx CBCS option, the targeted production of 262,000 t/day can be achieved by operating shift periods ranging from 820 min. to 1300 min.
6. The AFS technology could achieve a 44% reduction in the current production cost of $13.78/bbl to $7.72/bbl, with robust long-term economic outlook, compared to the 5-year industry target of $9.00/bbl.
7. The long-term stability and sustainability of the AFS technology were also analyzed using quantitative risk modeling and analysis. The results show that there is 85% probability that the NPVs of the CMS and CycEx CBCS options will be greater than $\$2.69 \times 10^{10}$ and $\$3.42 \times 10^{10}$ respectively.

7.1 Main Contributions

The principal contributions of this study are:

1. The AFS mining technology is a new surface mining method that has a potential to reduce the production cost of mining the Athabasca oil sands deposits, which is currently over six times that of conventional oil cost. This research investigation marks the first comprehensive examination of the geometrical AFS design and space requirements.
2. This research initiative is a pioneering effort to develop continuous ellipsoid and spheroid processes for characterizing material extraction process associated with the

novel AFS technology. The development of the continuous ellipsoid and spheroid processes for continuous surface mining operations, which provides solutions for material excavation and haulage problems are major contributions to strategic and tactical plans. To date no research has been undertaken to solve the problems outlined and solved in this study. The successful implementation of the AFS technology will be determined by complete understanding of material flows from multi-bench, multi-face operations, vis à vis the space and geometry requirements of the equipment components.

3. The expansion of this research frontier will also create the basis and the impetus for expanded research by other researchers in mining methods and production engineering.

7.2 Recommendations

On the basis of the research carried out and the results obtained and discussed, the following recommendations are made:

1. A research study must be undertaken to develop the AFS equipment-oil sands formation interactions to understand the ground pressures and formation bearing capacities required for optimum equipment performance.
2. A virtual simulator must be developed for simulating the AFS machinery extensively covering various operating paradigms. The simulation must deal with machine-formation interactions, ground pressures and formation bearing capacity, and machine components interfaces.
3. The dynamic evolution of the layouts using the 3D ellipsoid and spheroid processes must be automated to simulate material excavation processes under various operating paradigms. This will allow for comprehensive strategic and tactical mine plans for efficient and effective AFS production and productivity.
4. The final implementation of the AFS technology must be preceded by a pilot scale prototype of the technology. This will allow for complete testing and study of the equipment and the associated interfaces for correcting any deviations and errors before implementation.

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APPENDICES

APPENDIX A

EQUIPMENT DIMENSIONS

Type	Dimensions (m)
Average Bench Height, H	13
Bench slope angle, θ	50^o
Average Bench Width, B	80
Berm width, b_{rw}	3.0
Minimum clearance between trucks	3.0
Radius of level floor of shovel (RLF)	24.35
Width of RH 400 hydraulic shovel	9.06
Dumping radius of shovel, DR	8.9
Dipper capacity of O&K RH 400 hydraulic shovel	43.25 m^3
Dumping height of shovel	9
Height of shovel cab	10.2
Maximum Scaling height of shovel, H_s	20.2
Dumping radius of shovel	8.9
Width of largest truck	10
Maximum grade of haul roads/ramps	8^o
Length of bucket wheel excavator (Krupp BWE 1420)	193
Radius of clearance of BWE	75
Length of slurrification unit, L_{cf}	54
Length of hopper-crusher unit, L_{hc}	54
Length of one conveyor belt wagon, L_{cw}	20
Length of one flexible pipeline unit, L_{fp}	60
Dimensions of Base Mine	5 km × 7 km × 60 m deep
Dimensions of North Mine	6 km × 10 km × 76 m deep
Dimensions of Aurora Mine	5 km × 2.5 km × 70 m deep
Grade of Oil sands mined	10 – 18% (by wt.)

APPENDIX B

VOLUME OF MATERIALS EXCAVATED FROM BENCH #1 OF CIRCULAR AND ELLIPTICAL PITS

Length Incremental Pushback = 1.92 m

Increment Number	Circular Pit				Elliptical Pit			
	Expansion (m)	Pit Radius (m)	Volume (m^3)	Time days)	Major Axis (m)	Minor Axis (m)	Volume Excavated (m^3)	Time (days)
1	1.92	92.83	13600.00	0.11	91.02	41.02	9644.11	0.08
2	3.84	94.75	13900.00	0.11	91.02	41.02	9943.84	0.08
3	5.76	96.67	14200.00	0.11	91.02	41.02	10243.78	0.08
4	7.68	98.59	14500.00	0.12	91.03	41.03	10543.89	0.08
5	9.60	100.51	14800.00	0.12	91.03	41.03	10844.13	0.09
6	11.50	102.41	15100.00	0.12	91.03	41.03	11144.49	0.09
7	13.40	104.31	15400.00	0.12	91.03	41.03	11444.94	0.09
8	15.40	106.31	15700.00	0.13	91.04	41.04	11745.48	0.09
9	17.30	108.21	16000.00	0.13	91.04	41.04	12046.08	0.10
10	19.20	110.11	16300.00	0.13	91.04	41.04	12346.74	0.10
11	21.10	112.01	16600.00	0.13	91.04	41.04	12647.45	0.10
12	23.00	113.91	16900.00	0.14	91.05	41.05	12948.20	0.10
13	25.00	115.91	17200.00	0.14	91.05	41.05	13248.99	0.11
14	26.90	117.81	17500.00	0.14	91.05	41.05	13549.81	0.11
15	28.80	119.71	17800.00	0.14	91.05	41.05	13850.66	0.11
16	30.70	121.61	18100.00	0.15	91.06	41.06	14151.53	0.11
17	32.60	123.51	18400.00	0.15	91.06	41.06	14452.43	0.12
18	34.60	125.51	18700.00	0.15	91.06	41.06	14753.35	0.12
19	36.50	127.41	19000.00	0.15	91.06	41.06	15054.28	0.12
20	38.40	129.31	19300.00	0.15	91.06	41.06	15355.23	0.12
21	40.30	131.21	19600.00	0.16	91.07	41.07	15656.19	0.13
22	42.20	133.11	19900.00	0.16	91.07	41.07	15957.17	0.13
23	44.20	135.11	20200.00	0.16	91.07	41.07	16258.15	0.13
24	46.10	137.01	20500.00	0.16	91.07	41.07	16559.15	0.13
25	48.00	138.91	20800.00	0.17	91.08	41.08	16860.15	0.14
26	49.90	140.81	21100.00	0.17	91.08	41.08	17161.16	0.14
27	51.80	142.71	21400.00	0.17	91.08	41.08	17462.18	0.14
28	53.80	144.71	21700.00	0.17	91.08	41.08	17763.21	0.14
29	55.70	146.61	22000.00	0.18	91.09	41.09	18064.24	0.14
30	57.60	148.51	22300.00	0.18	91.09	41.09	18365.28	0.15
31	59.50	150.41	22600.00	0.18	91.09	41.09	18666.32	0.15
32	61.40	152.31	22900.00	0.18	91.09	41.09	18967.37	0.15
33	63.40	154.31	23200.00	0.19	91.10	41.10	19268.42	0.15
34	65.30	156.21	23500.00	0.19	91.10	41.10	19569.48	0.16
35	67.20	158.11	23800.00	0.19	91.10	41.10	19870.54	0.16
36	69.10	160.01	24100.00	0.19	91.10	41.10	20171.60	0.16
37	71.00	161.91	24400.00	0.20	91.11	41.11	20472.66	0.16
38	73.00	163.91	24700.00	0.20	91.11	41.11	20773.73	0.17

APPENDIX B (continued)

VOLUME OF MATERIALS EXCAVATED FROM BENCH #1 OF CIRCULAR AND ELLIPTICAL PITS

Increment Number	Circular Pit				Elliptical Pit			
	Expansion (m)	Pit Radius (m)	Volume (m^3)	Time days)	Major Axis (m)	Minor Axis (m)	Volume Excavated (m^3)	Time (days)
39	74.90	165.81	25000.00	0.20	91.11	41.11	21074.80	0.17
40	76.80	167.71	25300.00	0.20	91.11	41.11	21375.87	0.17
41	78.70	169.61	25600.00	0.21	91.12	41.12	21676.95	0.17
42	80.60	171.51	25900.00	0.21	91.12	41.12	21978.02	0.18
43	82.60	173.51	26200.00	0.21	91.12	41.12	22279.10	0.18
44	84.50	175.41	26500.00	0.21	91.12	41.12	22580.18	0.18
45	86.40	177.31	26800.00	0.22	91.13	41.13	22881.27	0.18
46	88.30	179.21	27100.00	0.22	91.13	41.13	23182.35	0.19
47	90.20	181.11	27400.00	0.22	91.13	41.13	23483.43	0.19
48	92.20	183.11	27700.00	0.22	91.13	41.13	23784.52	0.19
49	94.10	185.01	28000.00	0.22	91.13	41.13	24085.61	0.19
50	96.00	186.91	28300.00	0.23	91.14	41.14	24386.70	0.20
51	97.90	188.81	28600.00	0.23	91.14	41.14	24687.79	0.20
52	99.80	190.71	28900.00	0.23	91.14	41.14	24988.88	0.20
53	102.00	192.91	29200.00	0.23	91.14	41.14	25289.97	0.20
54	104.00	194.91	29500.00	0.24	91.15	41.15	25591.06	0.21
55	106.00	196.91	29800.00	0.24	91.15	41.15	25892.16	0.21
56	108.00	198.91	30100.00	0.24	91.15	41.15	26193.25	0.21
57	109.00	199.91	30400.00	0.24	91.15	41.15	26494.35	0.21
Total Volume Excavated (m^3)			1,254,000.00	10.05			1,029,732.66	8.25
Days			10.05				8.25	
Years			0.03				0.02	

APPENDIX C

SUMMARY OF VOLUME OF MATERIALS EXCAVATED FROM PITS

D1 Volume of Materials Excavated from a Circular Frustum using CMS Option

Bench	Volume Excavated (m^3)	Time to Mine		Time Started (days)
		Days	Years	
Bench #1	365,861,870.26	2,932.48	8.03	0.00
Bench #2	344,123,995.33	2,758.25	7.56	174.23
Bench #3	322,610,143.98	2,585.81	7.08	172.44

D2 Volume of Materials Excavated from a Circular Frustum using CycEx CBCS Option

Bench	Volume Excavated (m^3)	Time to Mine		Time Started (days)
		Days	Years	
Bench #1	364,869,978.70	2,924.53	8.01	0.00
Bench #2	323,391,425.90	2,592.07	7.10	332.46
Bench #3	284,406,367.86	2,279.59	6.25	312.48

D3 Volume of Materials Excavated from an Elliptical Frustum using CMS Option

Bench	Volume Excavated (m^3)	Time to Mine		Time Started (days)
		Days	Years	
Bench #1	372,131,189.97	2,982.73	8.17	0.00
Bench #2	350,209,631.43	2,807.02	7.69	175.71
Bench #3	328,508,188.32	2,633.08	7.21	173.94

D4 Volume of Materials Excavated from an Elliptical Frustum using CycEx CBCS

Bench	Volume Excavated (m^3)	Time to Mine		Time Started (days)
		Days	Years	
Bench #1	372,977,079.55	2,989.51	8.19	0.00
Bench #2	331,049,834.69	2,653.45	7.27	336.06
Bench #3	291,196,597.75	2,334.02	6.39	319.43

APPENDIX D

Summary of Simulated Results of Fleets Options in Discrete Shovel-Truck System

Rank	Option	Combination of Trucks			Time	Total Cost	Unit Cost
Scenario 1: One dumping site at the crusher location							
Rank	Option	240-t	320-t	360-t	(min.)	($)	($/t)
1	Option A9	0	0	12	1094.00	124071310.19	1.297
2	Option A34	0	6	3	1187.79	127642275.62	1.335
3	Option A33	0	6	6	1137.25	132968059.00	1.390
4	Option A13	6	0	6	1160.71	137356165.09	1.436
5	Option A10	0	0	15	1090.99	138071599.06	1.444
6	Option A28	0	9	3	1166.47	139306082.55	1.457
7	Option A21	0	12	0	1193.78	145294383.20	1.519
8	Option A1	9	0	3	1199.64	145490979.77	1.521
9	Option A6	15	0	0	1143.59	146110213.65	1.528
10	Option A35	0	6	9	1135.49	148461103.69	1.552
Scenario 2: Two dumping sites at the crusher location							
Rank	Option	Combination of Trucks			Time	Total Cost	Unit Cost
		240-t	320-t	360-t	(min.)	($)	($/t)
1	Option B34	0	6	3	1094.85	85101793.01	0.890
2	Option B9	0	0	12	1086.46	88655827.45	0.927
3	Option B33	0	6	6	1171.21	91354335.85	0.955
4	Option B13	6	0	6	1137.25	91452155.91	0.956
5	Option B10	0	0	15	1084.17	93116133.97	0.974
6	Option B28	0	9	3	1158.04	94443567.29	0.988
7	Option B21	0	12	0	1164.70	95725076.96	1.001
8	Option B1	9	0	3	1078.91	97002278.04	1.014
9	Option B25	6	6	0	1144.94	97896269.84	1.024
10	Option B29	0	9	6	1190.96	99889327.76	1.045
Scenario 3: Three dumping sites at the crusher location							
Rank	Option	Combination of Trucks			Time	Total Cost	Unit Cost
		240-t	320-t	360-t	(min.)	($)	($/t)
1	Option C9	0	0	12	1092.49	124274173.38	1.300
2	Option C34	0	6	3	1171.91	124879354.86	1.306
3	Option C26	6	3	0	1214.14	132149062.97	1.382
4	Option C33	0	6	6	1140.56	134336803.60	1.405
5	Option C13	6	0	6	1157.36	137175066.64	1.434
6	Option C10	0	0	15	1085.74	137285001.40	1.436
7	Option C28	0	9	3	1161.81	138811529.29	1.452
8	Option C21	0	12	0	1190.85	145230359.98	1.519
9	Option C25	6	6	0	1179.25	145401783.71	1.520
10	Option C1	9	0	3	1198.55	145884234.79	1.526

APPENDIX E

SIMULATION RESULTS OF CURRENT MINING SYSTEM (CMS)

Scenario 1	**One Server at Crusher location**										Trucks Combinations			Shift Time (min)	Total Cost/min
Time (min.)	450	480	500	600	720	900	960	1200	1350	1440	240 t	320 t	360 t		
Option A1	97920	104400	108840	130800	156960	196320	209400	262080	294720	314400	3	0	1	1199.64	121279.32
Option A2	100800	107640	111960	134400	161040	201120	214320	267960	301560	321120	3	0	2	1173.33	133095.65
Option A3	100920	107400	111960	133920	160200	199920	212880	265920	299280	319080	4	0	2	1182.26	147681.85
Option A4	102960	109800	114120	136320	163440	203880	217200	271200	304560	324840	4	0	3	1159.11	159089.85
Option A5	104760	111720	116160	138960	166320	206280	219720	274680	308400	329040	4	0	4	1144.63	171223.41
Option A6	94080	100320	104640	125280	150240	187440	199920	256560	281760	300720	5	0	0	1143.59	127764.03
Option A7	94800	101040	105120	126240	151200	188640	201120	251040	282240	301440	6	0	0	1232.38	151832.15
Option A8	95280	101520	105840	126720	151680	189360	202080	252480	283200	302400	7	0	0	1252.69	168716.49
Option A9	108000	114840	119880	143640	172800	216000	230040	287280	323280	344880	0	0	4	1094.00	113410.27
Option A10	108720	115920	120600	144360	173160	216360	230760	288000	324000	345600	0	0	5	1090.99	126556.79
Option A11	109800	117000	121680	145440	173880	217440	231840	289440	325440	347040	0	0	6	1085.67	139333.67
Option A12	110160	117000	122040	145440	174240	217440	231480	289080	324720	349960	0	0	7	1087.17	152938.54
Option A13	101640	108600	113040	135240	162840	203280	216840	270840	305400	325680	2	0	2	1160.71	118337.94
Option A14	85920	91800	95520	114480	137760	172440	183720	229920	258720	275520	2	0	1	1367.57	122556.26
Option A15	103080	109920	114720	137280	164280	204720	218520	272520	306600	327240	3	0	3	1153.24	145044.45
Option A16	95120	101600	105760	126800	152480	190480	203200	254000	286160	304880	3	1	0	1237.80	124607.27
Option A17	96560	103040	107360	128640	154400	193200	206160	257280	289280	308640	3	2	0	1237.31	139293.81
Option A18	97200	103680	107680	128960	154560	192960	205520	256720	288960	307920	4	2	0	1222.13	151614.93
Option A19	98400	104880	109120	130320	156240	194640	207360	258880	291040	310480	4	3	0	1224.57	166500.96
Option A20	99520	106240	110560	131760	157600	195920	208880	260320	292400	311600	4	4	0	1214.55	179603.44
Option A21	98560	105280	109760	131840	158080	198080	210880	263360	296640	316480	0	4	0	1193.78	121709.46
Option A22	99840	106560	110720	132800	158720	198400	211520	264320	297280	317120	0	5	0	1189.45	135433.52
Option A23	100480	106880	111360	133120	159360	198720	215160	264640	297600	317440	0	6	0	1187.19	149314.41
Option A24	101120	107520	112320	134080	160320	200000	213120	265920	299200	319040	0	7	0	1182.18	162762.41
Option A25	96560	103280	107520	129040	155040	193440	206400	257840	287960	309120	2	2	0	1220.72	123410.60
Option A26	85040	90720	94400	113600	136080	170480	182240	228080	257120	274160	2	1	0	1375.77	122702.45
Option A27	98240	104880	109040	130080	155920	194480	207360	258960	291120	310400	3	3	0	1214.07	151134.91
Option A28	101240	107880	112160	134600	161680	202240	215440	269560	303560	324000	0	3	1	1166.47	119424.88
Option A29	103440	110160	114560	137400	164640	205960	219360	273800	307760	328240	0	3	2	1147.98	131693.95
Option A30	103960	111280	115280	138320	165280	206280	219960	274280	307960	328280	0	4	2	1145.74	145082.02
Option A31	105840	112200	116880	139800	167120	207880	221400	276760	310480	331000	0	4	3	1136.01	157864.63
Option A32	107680	114200	118960	141720	169240	210040	224360	278720	313080	333520	0	4	4	1126.18	170392.22
Option A33	104000	111160	115560	138400	165880	207760	221320	276400	310760	331800	0	2	2	1137.25	116920.19
Option A34	99040	105680	110360	132360	158360	198680	212000	264680	298000	317360	0	2	1	1187.79	107461.95
Option A35	105680	112480	117240	140040	167240	208720	221960	276720	311440	331760	0	2	3	1135.49	130746.78

Scenario 2	Two servers at crusher location										Truck Combinations			Shift	
Time (min.)	450	480	500	600	720	900	960	1200	1350	1440	240 t	320 t	360 t	Time (min)	Total Cost/min
Option B1	98960	105360	109440	130800	156960	196080	209280	261720	294360	314040	3	0	1	1201.29	121720.52
Option B2	101040	108480	112560	134880	161520	201120	214680	268320	301320	321360	3	0	2	1171.72	133180.45
Option B3	101760	108240	112320	135000	161280	200520	213960	266400	300240	320040	4	0	2	1179.86	147651.47
Option B4	104040	110640	114840	137400	164160	203880	217320	270960	304560	324720	4	0	3	1159.91	159464.39
Option B5	105600	112440	117000	139680	166800	207000	221040	272400	309240	329760	4	0	4	1151.40	172499.45
Option B6	95280	101520	105360	126240	151440	188880	200880	251520	282960	301680	5	0	0	1201.29	134483.78
Option B7	96000	101760	105840	126960	151920	189360	201840	252000	283200	302120	6	0	0	1248.08	154050.88
Option B8	96240	102480	106560	127440	152400	189840	202320	252480	284160	302640	7	0	0	1245.08	167974.93
Option B9	108720	115920	120600	144360	172800	216000	230040	286920	322920	344160	0	0	4	1094.85	113748.16
Option B10	109800	117800	122040	145800	174240	217440	231840	289080	325080	346320	0	0	5	1086.46	126279.54
Option B11	109800	117360	122040	146160	174600	217800	232200	289800	325440	347040	0	0	6	1084.17	139388.68
Option B12	111240	118800	123480	147600	176400	219600	233640	290880	326880	348120	0	0	7	1078.91	152023.32
Option B13	102240	108600	113520	135480	163200	203520	217440	271440	305280	325440	2	0	2	1158.04	118330.46
Option B14	87840	93600	97680	116880	140760	175800	187440	234960	264000	281640	2	0	1	1339.67	120361.65
Option B15	103920	111000	115680	138000	165600	206160	219720	273480	307320	327720	3	0	3	1148.75	144741.45
Option B16	95760	102000	106320	126960	152560	190800	203520	254400	286720	305680	3	1	0	1235.27	124635.31
Option B17	97360	103680	108800	128960	154800	192800	205600	257280	289920	308640	3	2	0	1221.69	137814.02
Option B18	97840	104160	108400	129520	155120	193280	205760	256720	288720	307920	4	2	0	1224.75	152220.21
Option B19	99600	105600	110640	131840	157440	196000	209120	260400	292480	311760	4	3	0	1207.48	164453.71
Option B20	100480	106560	110800	132320	158160	196800	209280	260880	293600	312960	4	4	0	1205.13	178485.92
Option B21	99520	105600	110400	132480	158400	198080	210880	264000	297280	317120	0	4	0	1190.96	121694.21
Option B22	100800	107200	111360	133760	160000	199360	212160	264640	297600	317760	0	5	0	1187.93	135530.79
Option B23	101440	107840	112000	133760	160000	199360	212800	265280	298240	318400	0	6	0	1185.00	149308.92
Option B24	102080	109120	113280	135360	161600	201280	214080	266880	299520	319360	0	7	0	1177.82	162430.54
Option B25	96800	103520	108080	129360	155040	194000	206640	258080	290640	310240	2	2	0	1218.06	123419.96
Option B26	87760	93120	97120	116240	139760	174480	185920	232640	261680	279520	2	1	0	1351.61	120856.23
Option B27	98800	105280	109680	130800	156480	195360	208240	259440	292000	311280	3	3	0	1211.79	151128.65
Option B28	101240	107880	112160	134960	162000	202600	216080	269920	303200	323000	0	3	1	1164.70	119508.65
Option B29	104160	110840	115240	138080	165320	205960	220080	274480	309400	329560	0	3	2	1144.94	131606.81
Option B30	104960	111640	116280	139280	166280	206280	219960	273920	307960	327920	0	4	2	1146.98	145500.81
Option B31	106160	113240	117560	140160	167840	208560	222360	276760	310800	331320	0	4	3	1134.88	157966.81
Option B32	108000	115200	119280	142400	169920	211080	225040	279400	314120	334880	0	4	4	1123.18	170194.41
Option B33	104360	111480	115560	138360	166240	207400	221320	276400	310760	331480	0	2	2	1137.25	117179.84
Option B34	101040	107360	112080	134360	161360	201680	215040	268400	302040	322400	0	2	1	1171.21	106229.68
Option B35	106040	113200	117240	140400	167920	208720	222680	277760	311720	332520	0	3	3	1131.33	143999.32

Scenario 3	Three servers at Crusher location										Trucks Combinations			Shift	
Time (min.)	450	480	500	600	720	900	960	1200	1350	1440	240 t	320 t	360 t	Time (min)	Total Cost/min
Option C1	98400	105120	109440	131280	156920	196320	209280	262320	294960	314400	3	0	1	1198.55	121717.06
Option C2	101760	108480	113280	135120	161520	202320	215880	269280	302640	322680	3	0	2	1167.28	132942.10
Option C3	102240	108960	112920	135000	162000	201840	215160	268080	301680	321840	4	0	2	1172.43	146988.53
Option C4	103320	110400	114840	137280	164160	204360	217200	271440	304920	325080	4	0	3	1158.23	159497.80
Option C5	105580	112680	117360	140040	167160	207720	221400	275400	309480	327760	4	0	4	1140.44	171118.22
Option C6	95040	101040	105360	126240	150960	188880	201120	251520	282960	301680	5	0	0	1250.00	140222.60
Option C7	96240	102240	106560	127440	152400	190080	202560	252960	284640	303600	6	0	0	1242.80	153683.66
Option C8	96240	102720	107040	127920	152640	190320	202560	252720	284160	302880	7	0	0	1244.27	168150.96
Option C9	108720	116280	120600	144360	172800	216000	230400	287640	323640	345600	0	0	4	1092.49	113752.65
Option C10	109800	117000	122040	145800	174600	217440	232200	289080	324720	346320	0	0	5	1085.74	126443.90
Option C11	110520	118080	122400	146160	175320	218520	232920	290160	326160	347760	0	0	6	1081.93	139347.97
Option C12	111960	119160	123840	147600	176040	219240	234000	291600	326520	348120	0	0	7	1076.67	151953.06
Option C13	101280	108480	113160	135480	162480	203640	217200	271680	306120	325680	2	0	2	1157.36	118524.44
Option C14	88440	94440	98160	118080	141960	177480	189000	236880	266160	284160	2	0	1	1328.69	119678.43
Option C15	103920	110880	115080	137880	165360	206040	219360	273360	307320	328320	3	0	3	1149.51	145099.80
Option C16	95520	101760	106000	127280	152240	190240	202960	254160	285840	305120	3	1	0	1237.12	125104.31
Option C17	97040	103600	107680	128800	154640	193120	206160	257280	289280	308400	3	2	0	1222.13	138141.98
Option C18	97440	103680	108240	129280	155040	193200	205760	256960	288960	307680	4	2	0	1223.63	152359.75
Option C19	99360	105600	109920	131200	156720	194880	207840	259680	291120	310720	4	3	0	1211.07	165218.80
Option C20	100560	106960	111040	132320	158400	197600	210240	261520	293360	312880	4	4	0	1202.26	178334.88
Option C21	99520	106240	110400	132800	158720	198720	211520	264000	296960	316480	0	4	0	1190.85	121954.83
Option C22	100800	107520	111680	133760	160320	199680	212800	264960	297920	317760	0	5	0	1186.38	135625.22
Option C23	100800	107520	112000	133440	160000	199680	212480	264960	297920	317760	0	6	0	1186.46	149764.19
Option C24	102080	108480	112640	134720	161600	200640	213760	266560	299520	319360	0	7	0	1179.27	162896.90
Option C25	97440	103840	108400	129600	155360	194000	207200	258960	291200	310240	2	2	0	1179.25	123300.47
Option C26	87520	93840	97120	117120	140320	175200	186960	233600	263200	280480	2	1	0	1214.14	108841.35
Option C27	99040	105280	109680	131040	157040	195200	208160	259840	291680	311120	3	3	0	1343.92	167913.46
Option C28	101880	108840	113120	135600	162680	202920	216760	270560	304520	324320	0	3	1	1161.81	119478.26
Option C29	103800	110880	114880	137400	164960	205600	219040	273480	308080	328240	0	3	2	1149.39	132380.62
Option C30	103960	110680	115280	137960	165000	205640	219320	273000	306960	327640	0	4	2	1150.82	146250.26
Option C31	106840	113600	118240	140880	168160	209200	222720	277720	311800	332720	0	4	3	1131.40	157740.91
Option C32	108360	115520	119640	142800	170280	211400	225360	280080	314120	335520	0	4	4	1120.70	170074.93
Option C33	103720	110520	114960	138360	165560	206680	220320	275720	310440	330480	0	2	2	1140.56	117781.12
Option C34	100320	107360	111360	133320	160320	201360	214320	268320	302000	322640	0	2	1	1171.91	106560.43
Option C35	106040	112520	117240	140400	167600	208720	223040	277440	311720	332840	0	3	3	1131.88	144328.18

APPENDIX F

SIMULATION RESULTS OF CYCLIC EXCAVATOR CONVEYOR BELT CONTROL SYSTEM

APPENDIX F

SIMULATION RESULTS OF CYCLIC EXCAVATOR CONVEYOR BELT CONTROL SYSTEM

Scenario 1: Standard Case
Shovel cycle time: uniformly distributed between 0.88 and 1.25 minutes; Shovel capacity: uniformly distributed between 61.83 and 85.02 t/hr; Conveyor belt capacity: normally distributed with a mean of 101.67 t/min and a standard deviation of 8.33t/min (6,100 t/hr, std dev = 500 t/hr); Apron feeder/crusher capacity: uniformly distributed between 80 and 91.67 t/hr

Shift Time	Total Production (tonnes)		Productivity (t/hr)		Shovel Utilization (%)		Queue Length		Waiting Time (min.)	
(min.)	Mean	Standard Deviation	Mean	Standard Deviation	Mean	Standard Deviation	Mean	Standard Deviation	Mean	Standard Deviation
450	207650.74	24155.54	2222.9	14.43	97.66	2.13	1.37	0.98	1.47	1.06
					97.12	1.53	1.18	0.48	1.26	0.51
					96.55	2.40	1.28	1.05	1.37	1.12
	206244.81	23964.59	2189.13	14.64	97.90	2.35	1.53	0.90	1.64	0.96
					96.21	1.75	0.90	0.50	0.96	0.53
					97.65	1.45	2.54	1.49	2.73	1.60
	205958.48	23524.73	2191.09	15.43	97.77	1.41	1.42	0.70	1.51	0.75
					97.67	1.35	1.72	0.57	1.84	0.61
					96.22	3.24	1.25	0.56	1.33	0.59
Average	206618.01	23881.62	2201.04	14.83	97.19	1.96	1.47	0.80	1.57	0.86
480	218110.08	24.899.53	2226.41	14.56	97.21	1.27	0.90	0.61	0.97	0.65
					97.48	0.81	1.34	0.35	1.43	0.38
					97.72	1.32	0.79	0.16	0.84	0.17
	218005.45	25298.88	2201.50	15.41	96.20	2.08	1.31	0.68	1.40	0.73
					98.32	1.38	1.75	1.30	1.87	1.39
					97.29	2.15	2.21	1.56	2.36	0.71
	219377.00	25455.40	2219.83	14.87	96.59	2.38	1.72	1.28	1.83	1.37
					97.91	2.27	1.89	1.13	2.01	1.20
					97.39	1.51	1.99	1.36	2.13	1.46
Average	218497.51	25377.14	2215.91	14.95	97.35	1.69	1.54	0.94	1.65	0.90

APPENDIX F

SIMULATION RESULTS OF CYCLIC EXCAVATOR CONVEYOR BELT CONTROL SYSTEM

Scenario 2: Conveyor belt capacity increased
Shovel cycle time: uniformly distributed between 0.88 and 1.25 minutes; Shovel capacity: uniformly distributed between 61.83 and 85.02 t/hr; Conveyor belt capacity: normally distributed with a mean of 122.01 and a standard deviation of 10.48 min. (7,320 t/hr, std dev = 600 t/hr); Apron feeder/crusher capacity: uniformly distributed between 100.64 and 135.03 t/hr).

Shift Time	Total Production (tonnes)		Productivity (t/hr)		Shovel Utilization (%)		Queue Length		Waiting Time (min.)	
(min.)	Mean	Standard Deviation	Mean	Standard Deviation	Mean	Standard Deviation	Mean	Standard Deviation	Mean	Standard Deviation
450	205077.48	23749.44	2292.05	13.69	97.37	1.59	1.34	0.45	1.43	0.47
					96.39	1.69	1.35	0.73	1.44	0.78
					98.73	0.77	2.26	1.06	2.42	1.12
	205473.35	23760.56	2295.04	14.43	96.48	1.93	0.94	0.45	1.00	0.48
					97.79	2.07	1.66	0.49	1.77	0.52
					97.36	2.68	1.43	0.46	1.53	0.49
	207040.03	24152.08	2319.13	13.33	97.51	1.04	1.01	0.76	1.08	0.81
					96.56	1.86	1.45	1.20	1.55	1.29
					98.52	0.92	1.85	0.85	1.97	0.89
Average	205863.62	23887.36	2302.07	13.82	97.41	1.62	1.48	0.72	1.58	0.76
480	218370.74	25027.17	2313.18	13.62	98.06	1.19	1.58	0.82	1.68	0.88
					98.87	0.56	1.59	1.15	1.70	1.22
					96.79	1.20	1.51	0.71	1.61	0.76
	217310.43	25168.54	2286.77	13.93	98.75	0.64	1.85	0.72	1.98	0.77
					97.84	1.28	0.92	0.25	0.98	0.27
					98.43	0.87	1.45	0.56	1.56	0.61
	218432.33	25139.29	2302.93	13.07	97.17	1.75	1.99	0.65	2.13	0.71
					97.41	2.52	1.87	1.04	2.00	1.11
					97.21	1.75	1.47	0.54	1.57	0.57
Average	218037.83	25111.67	2300.96	13.54	97.84	1.31	1.58	0.72	1.69	0.77

APPENDIX F

SIMULATION RESULTS OF CYCLIC EXCAVATOR CONVEYOR BELT CONTROL SYSTEM

Scenario 3: Conveyor belt capacity reduced
Shovel cycle time: uniformly distributed between 0.88 and 1.25 minutes; Shovel capacity: uniformly distributed between 61.83,85.02 t/hr; Conveyor belt capacity: normally distributed with a mean of 61.01 t/hr and a standard deviation of 5.03 t/hr (2,440 t/hr, std dev = 300 t/hr); Apron feeder/crusher capacity: uniformly distributed between 59.42 and 67.05 t/hr.

Shift Time	Total Production (tonnes)		Productivity (t/hr)		Shovel Utilization (%)		Queue Length		Waiting Time (min.)	
(min.)	Mean	Standard Deviation	Mean	Standard Deviation	Mean	Standard Deviation	Mean	Standard Deviation	Mean	Standard Deviation.
450	205953.99	23782.13	2138.96	17.06	96.81	2.04	1.45	0.7	1.54	0.75
					97.30	1.24	0.96	0.85	1.02	0.89
					97.24	2.10	0.85	0.32	0.91	0.34
	205946.58	23871.73	2140.60	16.60	97.98	0.67	1.05	0.39	1.12	0.41
					96.98	2.11	1.55	1.29	1.66	1.39
					97.80	0.88	1.56	0.57	1.67	0.60
	203920.58	23511.44	2131.55	16.29	98.54	0.79	1.95	1.66	2.10	1.80
					97.68	1.15	1.37	1.17	1.46	1.27
					96.46	1.55	1.65	0.86	1.75	0.91
Average	205273.72	23721.77	2137.04	16.65	97.42	1.39	1.38	0.87	1.47	0.93
480	217957.96	25297.75	2166.36	16.60	96.84	2.18	2.03	1.47	2.17	1.57
					98.06	1.04	0.84	0.60	0.90	0.64
					96.99	1.68	1.53	0.99	1.63	1.05
	218631.22	25082.20	2156.79	15.94	98.63	0.72	1.53	0.73	1.63	0.78
					96.64	1.38	1.36	1.31	1.45	1.40
					97.33	1.26	0.87	0.67	0.93	0.72
	217919.45	25265.39	2164.19	16.38	97.79	1.39	1.30	0.47	1.39	0.51
					97.87	0.66	1.21	0.82	1.29	0.88
					97.26	1.13	1.17	0.29	1.25	0.31
Average	218169.54	25215.11	2162.45	16.31	97.49	1.27	1.32	0.82	1.40	0.87

APPENDIX F

SIMULATION RESULTS OF CYCLIC EXCAVATOR CONVEYOR BELT CONTROL SYSTEM

Scenario 4: Shovel capacity reduced

Shovel cycle time: uniformly distributed between 0.84 and 1.11 minutes; Shovel capacity: uniformly distributed between 48.46 and 68.02 t/hr; Conveyor belt capacity: normally distributed with a mean of 101.67 t/hr and standard deviation of 8.33 t/hr. (6,100 t/hr, std dev = 500 t/hr); Apron feeder/crusher capacity uniformly distributed between 80 and 91.67 t/hr

Shift Time	Total Production (tonnes)		Productivity (t/hr)		Shovel Utilization (%)		Queue Length		Waiting Time (min.)	
(min.)	Mean	Standard Deviation	Mean	Standard Deviation	Mean	Standard Deviation	Mean	Standard Deviation	Mean	Standard Deviation
450	164063.53	18881.53	1864.65	10.86	89.50	2.20	0.04	0.00	0.04	0.01
					89.68	1.96	0.04	0.00	0.04	0.01
					90.44	0.91	0.04	0.01	0.05	0.01
	164096.58	18979.30	1816.94	10.82	90.59	1.17	0.04	0.01	0.04	0.01
					87.70	2.49	0.05	0.01	0.05	0.01
					87.76	2.05	0.03	0.01	0.04	0.01
	163981.62	18842.15	1852.51	11.00	89.61	1.80	0.04	0.01	0.04	0.01
					88.77	1.73	0.03	0.01	0.04	0.01
					88.55	2.26	0.03	0.01	0.04	0.01
Average	164047.24	18900.99	1844.70	10.89	89.18	1.84	0.04	0.01	0.04	0.01
480	175413.61	20380.29	1879.85	11.07	90.63	0.69	0.04	0.01	0.04	0.01
					89.82	2.77	0.04	0.01	0.04	0.01
					90.36	2.01	0.04	0.01	0.04	0.01
	174372.84	20448.30	1878.53	11.29	89.27	1.53	0.04	0.00	0.04	0.00
					90.65	0.76	0.04	0.01	0.04	0.01
					89.92	1.34	0.04	0.01	0.04	0.01
	174091.31	20271.45	1875.40	10.28	90.32	1.70	0.05	0.01	0.05	0.01
					88.25	2.21	0.04	0.00	0.04	0.00
					90.10	1.85	0.04	0.00	0.04	0.00
Average	174625.92	20366.68	1877.93	10.88	89.92	1.65	0.04	0.01	0.04	0.01

APPENDIX F

SIMULATION RESULTS OF CYCLIC EXCAVATOR CONVEYOR BELT CONTROL SYSTEM

Scenario 5: Shovel capacity reduced
Shovel cycle time: uniformly distributed between 0.81 and 1.08 min.; Shovel capacity: uniformly distributed between 39.57 and 54.41 t/hr; Conveyor belt capacity: normally distributed with a mean of 10167 t/hr and standard deviation of 8.33 t/hr (6100 t/hr, std dev = 500 t/hr); Apron feeder/crusher capacity: uniformly distributed between 80 and 91.67 t/hr.

Shift Time	Total Production (tonnes)		Productivity (t/hr)		Shovel Utilization (%)		Queue Length		Waiting Time (min.)	
(min.)	Mean	Standard Deviation	Mean	Standard Deviation	Mean	Standard Deviation.	Mean	Standard Deviation	Mean	Standard Deviation
450	133559.76	15336.42	1553.17	7.36	86.93	1.08	0.02	0.00	0.03	0.00
					86.71	1.48	0.02	0.00	0.02	0.00
					87.23	1.07	0.02	0.00	0.02	0.00
	131419.59	15072.02	1520.19	8.12	87.57	1.83	0.02	0.01	0.02	0.01
					86.87	1.64	0.02	0.00	0.02	0.00
					87.20	1.46	0.02	0.00	0.02	0.00
	131390.63	15175.41	1540.94	7.70	86.93	1.24	0.02	0.00	0.02	0.00
					86.11	2.83	0.02	0.00	0.02	0.00
					86.57	1.76	0.02	0.00	0.02	0.00
Average	132123.33	15194.62	1538.10	7.73	86.90	1.60	0.02	0.00	0.02	0.00
480	141065.48	16196.82	1561.39	7.79	87.46	1.09	0.02	0.00	0.02	0.00
					86.94	1.85	0.02	0.00	0.02	0.00
					86.95	1.57	0.02	0.00	0.02	0.00
	138425.60	16072.79	1540.25	7.41	86.91	1.21	0.02	0.01	0.02	0.01
					87.62	1.13	0.02	0.00	0.02	0.00
					87.37	1.06	0.02	0.00	0.02	0.00
	139926.02	16107.08	1533.29	7.95	86.96	0.78	0.02	0.00	0.02	0.00
					87.17	1.43	0.02	0.00	0.02	0.00
					86.01	2.41	0.02	0.00	0.02	0.00
Average	139805.70	16125.56	1544.98	7.72	87.04	1.39	0.02	0.00	0.02	0.00

APPENDIX G

Summary of Key Data from Simulation of CycEx CBCS Option

Shift Time (min.)	Total Production (tonnes)	Estimated Production (tonnes)	Shovel Utilization (%)	Queue Length	Waiting Time (min.)
Scenario 1: Standard Operating conditions					
450	206618.01	144632.61	97.19	1.47	1.57
480	218497.51	152948.26	97.35	1.54	1.65
500	227493.50	159245.45	97.33	1.45	1.55
600	272460.24	190722.17	97.96	1.91	2.05
720	326015.10	228210.57	98.09	2.05	2.19
900	405329.56	283730.69	98.37	1.96	2.09
960	431912.33	302338.63	98.46	1.99	2.12
1200	540307.33	378215.13	98.35	2.37	2.53
1350	610142.38	427099.67	98.54	2.50	2.67
1440	647739.38	453417.56	98.78	2.64	2.81
Scenario 2: Conveyor belt capacity increased by 20%					
Shift Time (min.)	Total Production (tonnes)	Estimated Production (tonnes)	Shovel Utilization (%)	Queue Length	Waiting Time (min.)
450	205863.62	144104.53	97.41	1.48	1.58
480	218037.83	152626.48	97.84	1.58	1.69
500	227274.41	159092.08	97.48	1.62	1.74
600	272541.97	190779.38	97.46	1.64	1.76
720	326237.82	228366.47	97.72	1.89	2.02
900	406423.52	284496.46	97.99	2.19	2.34
960	431715.97	302201.18	98.26	1.96	2.09
1200	539396.77	377577.74	98.61	2.35	2.51
1350	606640.79	424648.55	98.77	2.11	2.26
1440	645955.14	452168.59	98.48	2.67	2.53
Scenario 3: Conveyor belt capacity reduced by 20%					
Shift Time (min.)	Total Production (tonnes)	Estimated Production (tonnes)	Shovel Utilization (%)	Queue Length	Waiting Time (min.)
450	205273.72	143691.60	97.42	1.38	1.47
480	218169.54	152718.68	97.49	1.32	1.40
500	227597.88	159318.51	97.56	1.61	1.72
600	272716.89	190901.82	97.49	1.27	1.40
720	325233.36	227663.35	98.25	1.65	1.76
900	404876.79	283413.75	98.13	1.99	2.13
960	432713.23	302899.26	98.19	2.43	2.60
1200	539705.04	377793.53	98.52	2.53	2.70
1350	605876.44	424113.51	98.78	2.36	2.44
1440	645404.30	451783.01	98.55	2.27	2.42

APPENDIX G (continued)

Summary of Key Data from Simulation of CycEx CBCS Option

Shift Time (min.)	Total Production (tonnes)	Estimated Production (tonnes)	Shovel Utilization (%)	Queue Length	Waiting Time (min.)
Scenario 4: Shovel digging capacity reduced by 20%					
450	164047.24	114833.07	89.18	0.04	0.04
480	174625.92	122238.14	89.92	0.04	0.04
500	181276.02	126893.21	89.71	0.04	0.04
600	217062.30	151943.61	90.25	0.04	0.04
720	260927.05	182648.94	90.09	0.04	0.04
900	326105.58	228273.91	90.33	0.04	0.04
960	347782.90	243448.03	90.23	0.04	0.04
1200	431724.83	302207.38	90.65	0.04	0.04
1350	485030.05	339521.04	90.81	0.04	0.04
1440	517708.66	362396.06	90.56	0.04	0.04
Scenario 5: Shovel digging capacity reduced by 40%					
Shift Time (min.)	Total Production (tonnes)	Estimated Production (tonnes)	Shovel Utilization (%)	Queue Length	Waiting Time (min.)
450	132123.33	92486.33	86.90	0.02	0.02
480	139805.70	97863.99	87.04	0.02	0.02
500	145622.87	101936.01	87.32	0.02	0.02
600	174099.72	121869.80	87.17	0.02	0.02
720	208401.66	145881.16	87.22	0.02	0.02
900	259807.66	181865.36	87.44	0.02	0.02
960	277156.32	194009.43	88.01	0.02	0.02
1200	345625.25	241937.68	87.59	0.02	0.02
1350	388677.79	272074.45	87.99	0.02	0.02
1440	413949.63	289764.74	87.74	0.02	0.02

APPENDIX H

SUMMARY ECONOMIC ANALYSIS ON AFS OPTIONS

CMS Option C9

Year	0	1	2	3	4	5	6
Operating Time (min.)		1,092.49	1,092.49	1,092.49	1,092.49	1,092.49	1,092.49
Production (tonnes)		9.56E+07	1.15E+08	1.38E+08	1.65E+08	1.98E+08	2.38E+08
Production (barrels)		4.78E+07	5.74E+07	6.89E+07	8.26E+07	9.91E+07	1.19E+08
Gross Revenue ($)		2.12E+09	2.61E+09	3.22E+09	3.98E+09	4.91E+09	6.05E+09
Royalties		1.06E+08	1.31E+08	1.61E+08	1.99E+08	2.45E+08	3.03E+08
Net Revenue		2.01E+09	2.48E+09	3.06E+09	3.78E+09	4.66E+09	5.75E+09
Operating Expenses ($)		5.30E+08	6.54E+08	8.07E+08	9.96E+08	1.23E+09	1.52E+09
Operating Profit ($)		1.59E+09	1.96E+09	2.42E+09	2.98E+09	3.68E+09	4.54E+09
Expected Capital Investment ($)	1.23E+08	9.00E+07					1.09E+08
LP1 (Shovel & Crusher)		1.23E+07	1.11E+07	1.00E+07	9.00E+06	8.10E+06	7.29E+06
LP2 (Trucks - 1st to 4th sets)		3.00E+07	2.00E+07	1.33E+07	8.89E+06	5.93E+06	3.95E+06
Interest		1.32E+07	1.24E+07	1.16E+07	1.09E+07	1.02E+07	9.61E+06
PreCCA		1.57E+09	1.95E+09	2.40E+09	2.97E+09	3.67E+09	4.53E+09
Depr_Base		1.23E+08	2.10E+08	2.07E+08	2.08E+08	2.08E+08	3.17E+08
Amortization		4.23E+07	3.11E+07	2.33E+07	1.79E+07	1.40E+07	1.12E+07
Depreciation		3.48E+06	5.92E+06	5.85E+06	5.85E+06	5.85E+06	8.94E+06
Depletion		1.06E+08	1.31E+08	1.61E+08	1.99E+08	2.45E+08	3.03E+08
Capital Cost Allowance (CCA)		1.52E+08	1.68E+08	1.90E+08	2.23E+08	2.65E+08	3.23E+08
Before Tax Cash Flow (BTCF)		1.42E+09	1.78E+09	2.21E+09	2.75E+09	3.40E+09	4.21E+09
TAX		6.40E+08	8.00E+08	9.96E+08	1.24E+09	1.53E+09	1.89E+09
After Tax Cash Flow (ATCF)		7.82E+08	9.78E+08	1.22E+09	1.51E+09	1.87E+09	2.31E+09
Net Cash Flow (NCF)		8.92E+08	1.11E+09	1.38E+09	1.72E+09	2.12E+09	2.63E+09
PV of NCF		7.75E+08	8.43E+08	9.11E+08	9.81E+08	1.06E+09	1.13E+09
PVCI	2.13E+08	0.00E+00	0.00E+00	0.00E+00	0.00E+00	0.00E+00	2.53E+08

APPENDIX H (continued)

SUMMARY ECONOMIC ANALYSIS ON AFS OPTIONS

CMS Option C9

Year	7	8	9	10	11	12	13	14
Operating Time (min.)	1,092.49	1,092.49	1,092.49	1,092.49	1,092.49	1,092.49	1,092.49	1,092.49
Production (tonnes)	2.86E+08	3.43E+08	4.11E+08	4.93E+08	5.92E+08	7.11E+08	8.53E+08	1.02E+09
Production (barrels)	1.43E+08	1.71E+08	2.06E+08	2.47E+08	2.96E+08	3.55E+08	4.26E+08	5.12E+08
Gross Revenue ($)	7.47E+09	9.22E+09	1.14E+10	1.40E+10	1.73E+10	2.14E+10	2.64E+10	3.25E+10
Royalties	3.74E+08	4.61E+08	5.69E+08	7.02E+08	8.66E+08	1.07E+09	1.32E+09	1.63E+09
Net Revenue	7.10E+09	8.76E+09	1.08E+10	1.33E+10	1.64E+10	2.03E+10	2.50E+10	3.09E+10
Operating Expenses ($)	1.87E+09	2.31E+09	2.85E+09	3.51E+09	4.34E+09	5.35E+09	6.60E+09	8.14E+09
Operating Profit ($)	5.60E+09	6.91E+09	8.53E+09	1.05E+10	1.30E+10	1.60E+10	1.98E+10	2.44E+10
Expected Capital Investment ($)						1.29E+08		
LP1 (Shovel & Crusher)	6.56E+06	5.90E+06	5.31E+06	4.78E+06	4.30E+06	3.87E+06	3.49E+06	3.14E+06
LP2 (Trucks - 1st to 4th sets)	3.64E+07	2.43E+07	1.62E+07	1.08E+07	7.20E+06	4.80E+06	4.31E+07	2.87E+07
PreCCA	5.59E+09	6.90E+09	8.51E+09	1.05E+10	1.30E+10	1.60E+10	1.97E+10	2.44E+10
Depr_Base	3.14E+08	3.14E+08	3.14E+08	3.14E+08	3.14E+08	4.43E+08	4.39E+08	4.40E+08
Amortization	4.30E+07	3.02E+07	2.15E+07	1.56E+07	1.15E+07	8.67E+06	4.66E+07	3.18E+07
Depreciation	8.85E+06	8.85E+06	8.85E+06	8.85E+06	8.85E+06	1.25E+07	1.24E+07	1.24E+07
Depletion	3.74E+08	4.61E+08	5.69E+08	7.02E+08	8.66E+08	1.07E+09	1.32E+09	1.63E+09
Capital Cost Allowance (CCA)	4.25E+08	5.00E+08	5.99E+08	7.26E+08	8.86E+08	1.09E+09	1.38E+09	1.67E+09
Before Tax Cash Flow (BTCF)	5.16E+09	6.40E+09	7.92E+09	9.78E+09	1.21E+10	1.49E+10	1.84E+10	2.27E+10
TAX	2.32E+09	2.88E+09	3.56E+09	4.40E+09	5.44E+09	6.71E+09	8.27E+09	1.02E+10
After Tax Cash Flow (ATCF)	2.84E+09	3.52E+09	4.35E+09	5.38E+09	6.65E+09	8.20E+09	1.01E+10	1.25E+10
Net Cash Flow (NCF)	3.22E+09	3.99E+09	4.93E+09	6.09E+09	7.52E+09	9.28E+09	1.14E+10	1.41E+10
PV of NCF	1.21E+09	1.30E+09	1.40E+09	1.51E+09	1.62E+09	1.74E+09	1.86E+09	2.00E+09
PVCI	0.00E+00	0.00E+00	0.00E+00	0.00E+00	0.00E+00	6.91E+08	0.00E+00	0.00E+00

APPENDIX H (continued)
SUMMARY ECONOMIC ANALYSIS ON AFS OPTIONS
CMS Option C9

Year	15	16	17	18	19	20	Sum
Operating Time (min.)	1,092.49	1,092.49	1,092.49	1,092.49	1,092.49	1,092.49	
Production (tonnes)	1.23E+09	1.47E+09	1.77E+09	2.12E+09	2.55E+09	3.06E+09	1.79E+10
Production (barrels)	6.14E+08	7.37E+08	8.84E+08	1.06E+09	1.27E+09	1.53E+09	8.93E+09
Gross Revenue ($)	4.01E+10	4.95E+10	6.11E+10	7.54E+10	9.30E+10	1.15E+11	5.96E+11
Royalties	2.01E+09	2.48E+09	3.05E+09	3.77E+09	4.65E+09	5.74E+09	2.98E+10
Net Revenue	3.81E+10	4.70E+10	5.80E+10	7.16E+10	8.83E+10	1.09E+11	5.67E+11
Operating Expenses ($)	1.00E+10	1.24E+10	1.53E+10	1.89E+10	2.33E+10	2.87E+10	1.49E+11
Operating Profit ($)	3.01E+10	3.71E+10	4.58E+10	5.65E+10	6.97E+10	8.60E+10	4.47E+11
Expected Capital Investment ($)				6.46E+07			3.93E+08
LP1 (Shovel & Crusher)	2.82E+06	2.54E+06	2.29E+06	2.06E+06	1.85E+06	1.67E+06	1.08E+08
LP2 (Trucks - 1st to 4th sets)	1.91E+07	1.28E+07	8.51E+06	5.67E+06	2.15E+07	1.44E+07	3.36E+08
Interest	1.20E+07	1.13E+07	1.08E+07	1.02E+07	1.39E+07	1.35E+07	2.31E+08
PreCCA	3.01E+10	3.71E+10	4.58E+10	5.65E+10	6.97E+10	8.60E+10	4.47E+11
Depr_Base	4.40E+08	4.40E+08	4.40E+08	5.04E+08	5.02E+08	5.02E+08	6.99E+09
Amortization	2.20E+07	1.53E+07	1.08E+07	7.73E+06	2.34E+07	1.60E+07	4.44E+08
Depreciation	1.24E+07	1.24E+07	1.24E+07	1.42E+07	1.42E+07	1.42E+07	1.97E+08
Depletion	2.01E+09	2.48E+09	3.05E+09	3.77E+09	4.65E+09	5.74E+09	2.98E+10
Capital Cost Allowance (CCA)	2.04E+09	2.50E+09	3.08E+09	3.79E+09	4.69E+09	5.77E+09	3.05E+10
Before Tax Cash Flow (BTCF)	2.80E+10	3.46E+10	4.27E+10	5.27E+10	6.50E+10	8.02E+10	4.16E+11
TAX	1.26E+10	1.56E+10	1.92E+10	2.37E+10	2.93E+10	3.61E+10	1.87E+11
After Tax Cash Flow (ATCF)	1.54E+10	1.90E+10	2.35E+10	2.90E+10	3.58E+10	4.41E+10	2.29E+11
Net Cash Flow (NCF)	1.74E+10	2.15E+10	2.66E+10	3.28E+10	4.04E+10	4.99E+10	2.59E+11
PV of NCF	2.14E+09	2.30E+09	2.47E+09	2.65E+09	2.84E+09	3.05E+09	3.38E+10
PVCI	0.00E+00	0.00E+00	0.00E+00	7.99E+08	0.00E+00	0.00E+00	1.74E+09

Net Present Value (NPV) = $3.20E+10
Internal Rate of Return (IRR) = 29.02

Profitability Index (PI) = 19.37
Discounted Payback Period (DPBP) = 0.27

APPENDIX H (continued)

SUMMARY ECONOMIC ANALYSIS ON AFS OPTIONS

CycEx CBCS – Scenario 4

Year	0	1	2	3	4	5	6
Operating Time (min.)		1,171.21	1,171.21	1,171.21	1,171.21	1,171.21	1,171.21
Production (tonnes)		1.34E+08	1.61E+08	1.93E+08	2.32E+08	2.79E+08	3.34E+08
Production (barrels)		6.72E+07	8.06E+07	9.67E+07	1.16E+08	1.39E+08	1.67E+08
Gross Revenue ($)		2.50E+09	3.08E+09	3.80E+09	4.69E+09	5.78E+09	7.14E+09
Royalties		1.25E+08	1.54E+08	1.90E+08	2.34E+08	2.89E+08	3.57E+08
Net Revenue		2.37E+09	2.93E+09	3.61E+09	4.45E+09	5.49E+09	6.78E+09
Operating Expenses ($)		1.10E+09	1.36E+09	1.68E+09	2.07E+09	2.55E+09	3.15E+09
Operating Profit ($)		1.39E+09	1.72E+09	2.12E+09	2.62E+09	3.23E+09	3.99E+09
Expected Capital Investment ($)	3.13E+08						6.68E+07
LP1 (Shovel & Field conveyor)	1.08E+08	1.08E+07	9.70E+06	8.73E+06	7.85E+06	7.07E+06	6.36E+06
LP2 (Mobile crusher & transfer conveyor)	1.50E+08	2.01E+07	1.74E+07	1.51E+07	1.31E+07	1.13E+07	9.81E+06
LP3 (Belt wagons - 1st to 4th sets)	5.50E+07	1.83E+07	1.22E+07	8.15E+06	5.43E+06	3.62E+06	2.41E+06
PreCCA		1.37E+09	1.70E+09	2.10E+09	2.60E+09	3.21E+09	3.97E+09
Depr_Base		1.08E+08	3.10E+08	3.04E+08	3.05E+08	3.05E+08	3.71E+08
Amortization		3.08E+07	2.71E+07	2.38E+07	2.09E+07	1.84E+07	1.62E+07
Depreciation		3.04E+06	8.75E+06	8.59E+06	8.59E+06	8.59E+06	1.05E+07
Depletion		1.25E+08	1.54E+08	1.90E+08	2.34E+08	2.89E+08	3.57E+08
Capital Cost Allowance (CCA)		1.59E+08	1.90E+08	2.22E+08	2.64E+08	3.16E+08	3.83E+08
Before Tax Cash Flow (BTCF)		1.22E+09	1.51E+09	1.88E+09	2.34E+09	2.90E+09	3.59E+09
TAX		5.47E+08	6.81E+08	8.47E+08	1.05E+09	1.30E+09	1.61E+09
After Tax Cash Flow (ATCF)		6.69E+08	8.32E+08	1.04E+09	1.29E+09	1.59E+09	1.97E+09
Net Cash Flow (NCF)		7.96E+08	9.95E+08	1.23E+09	1.53E+09	1.89E+09	2.34E+09
PV of NCF		6.93E+08	7.52E+08	8.11E+08	8.74E+08	9.41E+08	1.01E+09
PVCI	3.13E+08	0.00E+00	0.00E+00	0.00E+00	0.00E+00	0.00E+00	1.55E+08

APPENDIX H (continued)

SUMMARY ECONOMIC ANALYSIS ON AFS OPTIONS

CycEx CBCS – Scenario 4

Year	7	8	9	10	11	12	13
Operating Time (min.)	1,171.21	1,171.21	1,171.21	1,171.21	1,171.21	1,171.21	1,171.21
Production (tonnes)	4.01E+08	4.81E+08	5.78E+08	6.93E+08	8.32E+08	9.98E+08	1.20E+09
Production (barrels)	2.01E+08	2.41E+08	2.89E+08	3.47E+08	4.16E+08	4.99E+08	5.99E+08
Gross Revenue ($)	8.80E+09	1.09E+10	1.34E+10	1.65E+10	2.04E+10	2.52E+10	3.11E+10
Royalties	4.40E+08	5.43E+08	6.70E+08	8.27E+08	1.02E+09	1.26E+09	1.55E+09
Net Revenue	8.36E+09	1.03E+10	1.27E+10	1.57E+10	1.94E+10	2.39E+10	2.95E+10
Operating Expenses ($)	3.89E+09	4.80E+09	5.92E+09	7.30E+09	9.01E+09	1.11E+10	1.37E+10
Operating Profit ($)	4.92E+09	6.07E+09	7.49E+09	9.24E+09	1.14E+10	1.41E+10	1.73E+10
Expected Capital Investment ($)						8.12E+07	
LP1 (Shovel & Field conveyor)	5.73E+06	5.15E+06	4.64E+06	4.17E+06	3.76E+06	3.38E+06	3.04E+06
LP2 (Mobile crusher & transfer conveyor)	8.50E+06	7.37E+06	6.39E+06	5.53E+06	4.80E+06	4.16E+06	3.60E+06
LP3 (Belt wagons - 1st to 4th sets)	2.23E+07	1.48E+07	9.90E+06	6.60E+06	4.40E+06	2.93E+06	2.71E+07
Interest	1.51E+07	1.42E+07	1.33E+07	1.25E+07	1.17E+07	1.10E+07	1.25E+07
PreCCA	4.90E+09	6.05E+09	7.47E+09	9.22E+09	1.14E+10	1.41E+10	1.73E+10
Depr_Base	3.70E+08	3.70E+08	3.70E+08	3.70E+08	3.70E+08	4.51E+08	4.49E+08
Amortization	1.42E+07	1.25E+07	1.10E+07	9.71E+06	8.55E+06	7.54E+06	6.65E+06
Depreciation	1.04E+07	1.04E+07	1.04E+07	1.04E+07	1.04E+07	1.27E+07	1.26E+07
Depletion	4.40E+08	5.43E+08	6.70E+08	8.27E+08	1.02E+09	1.26E+09	1.55E+09
Capital Cost Allowance (CCA)	4.65E+08	5.66E+08	6.92E+08	8.47E+08	1.04E+09	1.28E+09	1.57E+09
Before Tax Cash Flow (BTCF)	4.44E+09	5.49E+09	6.78E+09	8.38E+09	1.03E+10	1.28E+10	1.58E+10
TAX	2.00E+09	2.47E+09	3.05E+09	3.77E+09	4.66E+09	5.75E+09	7.09E+09
After Tax Cash Flow (ATCF)	2.44E+09	3.02E+09	3.73E+09	4.61E+09	5.69E+09	7.02E+09	8.67E+09
Net Cash Flow (NCF)	2.89E+09	3.57E+09	4.41E+09	5.44E+09	6.72E+09	8.30E+09	1.02E+10
PV of NCF	1.09E+09	1.17E+09	1.25E+09	1.35E+09	1.44E+09	1.55E+09	1.66E+09
PVCI	0.00E+00	0.00E+00	0.00E+00	0.00E+00	0.00E+00	4.34E+08	0.00E+00

APPENDIX H (continued)
SUMMARY ECONOMIC ANALYSIS ON AFS OPTIONS
CycEx CBCS – Scenario 4

Year	14	15	16	17	18	19	20	Sum
Operating Time (min.)	1,171.21	1,171.21	1,171.21	1,171.21	1,171.21	1,171.21	1,171.21	
Production (tonnes)	1.44E+09	1.72E+09	2.07E+09	2.48E+09	2.98E+09	3.58E+09	4.29E+09	2.51E+10
Production (barrels)	7.19E+08	8.62E+08	1.03E+09	1.24E+09	1.49E+09	1.79E+09	2.15E+09	1.25E+10
Gross Revenue ($)	3.83E+10	4.73E+10	5.84E+10	7.20E+10	8.88E+10	1.10E+11	1.35E+11	7.03E+11
Royalties	1.92E+09	2.36E+09	2.92E+09	3.60E+09	4.44E+09	5.48E+09	6.76E+09	3.51E+10
Net Revenue	3.64E+10	4.49E+10	5.54E+10	6.84E+10	8.44E+10	1.04E+11	1.28E+11	6.68E+11
Operating Expenses ($)	1.69E+10	2.09E+10	2.58E+10	3.18E+10	3.92E+10	4.84E+10	5.97E+10	3.10E+11
Operating Profit ($)	2.14E+10	2.64E+10	3.26E+10	4.02E+10	4.96E+10	6.12E+10	7.55E+10	3.93E+11
Expected Capital Investment ($)					9.86E+07			2.47E+08
LP1 (Shovel & Field conveyor)	2.74E+06	2.46E+06	2.22E+06	2.00E+06	1.80E+06	1.62E+06	1.46E+06	9.46E+07
LP2 (Mobile crusher & transfer conveyor)	3.12E+06	2.71E+06	3.52E+06	3.52E+06	3.52E+06	3.52E+06	3.52E+06	1.50E+08
LP3 (Belt wagons - 1st to 4th sets)	1.80E+07	1.20E+07	8.02E+06	5.35E+06	3.56E+06	3.29E+07	2.19E+07	2.40E+08
Interest	1.17E+07	9.69E+06	9.09E+06	8.53E+06	8.00E+06	8.73E+06	8.19E+06	2.53E+08
PreCCA	2.14E+10	2.64E+10	3.26E+10	4.02E+10	4.96E+10	6.12E+10	7.55E+10	3.92E+11
Depr_Base	4.49E+08	4.49E+08	4.49E+08	4.49E+08	5.47E+08	5.44E+08	5.44E+08	7.88E+09
Amortization	5.86E+06	5.17E+06	5.74E+06	5.51E+06	5.31E+06	5.14E+06	4.97E+06	2.45E+08
Depreciation	1.26E+07	1.26E+07	1.26E+07	1.26E+07	1.54E+07	1.54E+07	1.54E+07	2.22E+08
Depletion	1.92E+09	2.36E+09	2.92E+09	3.60E+09	4.44E+09	5.48E+09	6.76E+09	3.51E+10
Capital Cost Allowance (CCA)	1.93E+09	2.38E+09	2.94E+09	3.62E+09	4.46E+09	5.50E+09	6.78E+09	3.56E+10
Before Tax Cash Flow (BTCF)	1.95E+10	2.40E+10	2.96E+10	3.66E+10	4.51E+10	5.57E+10	6.87E+10	3.57E+11
TAX	8.76E+09	1.08E+10	1.33E+10	1.65E+10	2.03E+10	2.51E+10	3.09E+10	1.61E+11
After Tax Cash Flow (ATCF)	1.07E+10	1.32E+10	1.63E+10	2.01E+10	2.48E+10	3.06E+10	3.78E+10	1.96E+11
Net Cash Flow (NCF)	1.26E+10	1.56E+10	1.92E+10	2.37E+10	2.93E+10	3.61E+10	4.46E+10	2.32E+11
PV of NCF	1.79E+09	1.92E+09	2.06E+09	2.21E+09	2.37E+09	2.54E+09	2.72E+09	3.02E+10
PVCI	0.00E+00	0.00E+00	0.00E+00	0.00E+00	1.22E+09	0.00E+00	0.00E+00	1.81E+09

Net Present Value (NPV) = $\$2.84 \times 10^{10}$

Internal Rate of Return (IRR) = 27.48

Profitability Index (PI) = 16.68

Discounted Payback Period = 0.45

APPENDIX I

Summary of Risk Characterization of Mining Options at different Oil Prices

J1 - Summary of Risk Characterization of CMS Option at different Oil Prices

Price of Oil (US$/bll)	15	20	25	30	35	40
Profitability Index (PI)	6.66	10.96	15.25	19.54	23.83	28.13
Internal Rate of Return	23.63	26.09	27.77	29.06	30.11	30.99
Discounted Payback Period (yr)	0.84	0.49	0.36	0.27	0.23	0.18
Minimum NPV ($ × 10^9)	3.63	7.05	9.55	12.97	16.54	19.52
Maximum NPV ($ × 10^{10})	1.62	2.77	3.99	5.19	6.26	7.79
Mean NPV ($ × 10^{10})	0.99	1.74	2.48	3.23	3.98	4.73
Standard deviation ($ × 10^9)	1.52	2.67	3.82	4.97	6.12	7.27
Variance ($ × 10^{18})	2.31	7.13	14.61	24.73	37.50	52.95
Skewness (× 10^{-5})	-2.64	26.69	-30.46	10.00	16.46	231.26
Kurtosis	3.0091	2.9969	3.0002	2.9968	2.9904	3.0009
Mode ($ × 10^{10})	0.99	1.74	2.46	3.25	4.02	4.76
5% Percentile ($ × 10^{10})	0.74	1.30	1.86	2.41	2.97	3.53
10% Percentile ($ × 10^{10})	0.79	1.39	1.99	2.60	3.20	3.80
90% Percentile ($ × 10^{10})	1.18	2.08	2.97	3.87	4.77	5.66
95% Percentile ($ × 10^{10})	1.24	2.18	3.11	4.05	4.99	5.93

J2 - Summary of Risk Characterization of CBCS Option at different Oil Prices

Price of Oil (US$/bll)	15	20	25	30	35	40
Profitability Index (PI)	20.27	28.07	35.88	43.68	51.49	59.29
Internal Rate of Return	29.27	31.00	32.33	33.41	34.33	35.13
Discounted Payback Period (yr)	0.33	0.24	0.18	0.16	0.13	0.12
Minimum NPV ($ × 10^{10})	0.73	1.01	1.37	1.63	2.00	2.30
Maximum NPV ($ × 10^{10})	3.09	4.24	5.32	6.78	7.89	9.32
Mean NPV ($ × 10^{10})	1.85	2.60	3.34	4.09	4.84	5.59
Standard deviation ($ × 10^9)	2.84	3.99	5.149	6.30	7.45	8.60
Variance ($ × 10^{18})	8.08	15.96	26.47	39.66	55.48	73.99
Skewness (× 10^{-5})	189.80	83.92	3.34	155.61	191.91	36.31
Kurtosis	3.0073	3.0028	2.9922	3.0045	2.9985	3.0129
Mode ($ × 10^{10})	1.82	2.61	3.36	4.06	4.83	5.56
5% Percentile ($ × 10^{10})	1.38	1.94	2.50	3.06	3.62	4.18
10% Percentile ($ × 10^{10})	1.48	2.08	2.69	3.29	3.88	4.49
90% Percentile ($ × 10^{10})	2.21	3.11	4.00	4.90	5.80	6.69
95% Percentile ($ × 10^{10})	2.32	3.25	4.19	5.13	6.07	7.00