Geometallurgical Analysis - Implications on Operating Flexibility

(A Case for Geometallurgy for Orapa A/K1 deposit)

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DECLARATION

I declare that this research report is my own unaided work. It is being submitted to the Graduate Degree of Engineering to the University of the Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination to any other University.

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31st day of July year 2013

ABSTRACT

Historically, many mining investments have demonstrated inability to meet projected cash-flows despite detailed and costly project evaluations. The risk in mining projects can be attributed to the fact that decisions are made in situations involving high levels of uncertainty. The cash-flow uncertainty is driven by commodity price fluctuations and by the uncertainty of the geometries and grades of ore deposits. Accurately predicting future prices can be very difficult, however, better knowledge of the orebody will allow for improved strategic planning and an ability to build in flexibility into the design of the operation to deal with veiled uncertainty, orebody variability and operational constraints.

Geometallurgy provides a platform for cross-functional collaboration between geology and metallurgy, providing better inputs to mine planning and strategic decision making. Understanding variability in the orebody enables optimum operational designs and extraction methods that maximize value recovery. Geometallurgical programs allow for material characterization that is based on metallurgical responses instead of just geological zones, enabling informed design decisions and building in flexibility to handle variability. On the other hand, metallurgical design decisions made without full appreciation of the resource can limit ability to deal with variability, constraining the process right from design. Geometallurgical risk exists as a result of uncertainty in metallurgical characteristics of the ore resulting in treatability challenges.

Projects such as the Cawse Nickel Project in Western Australia and the Voorspoed Diamond Mine in South Africa are given as illustrations of geometallurgical risk at various phases of projects. Canahuire Project in Peru and the Kemi Chromite and Ferrochrome mine in Finland exemplify how successful geometallurgical programs have and can be implemented.

This report demonstrates a case for geometallurgy at the Orapa A/K1 deposit. Two rock types comprising the kimberlite, SVK_M and NPK_GG, previously not apparent in geological models, have introduced a constraint in the Orapa 2 treatment plant. This treatability challenge is due to the generation of non-settling slurries from the rock types. A metallurgical design decision in the selection of the thickeners has limited the plant's ability to deal with the changes in the ore blend. The Orapa case proves how design decisions can limit flexibility to deal with orebody variability, constraining the process from achieving the design capacity and limiting forecast cash-flows. The case demonstrates existence of geometallurgical risk and illustrates the consequences of this risk in operational and financial terms.

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1 INTRODUCTION

The orebody is the fundamental asset of a mining investment and profitability depends on its optimal exploitation. Within the scope of prefeasibility and feasibility studies, project evaluation assesses the economics of the resource, determining the optimal operational design. Effective operational design includes decision making on the mining and treatment methods that have the greatest impact on overall project economics. The 'optimal' design offers the most favourable economic option, or the best possible scenario that maximizes value of the asset. The basis of project valuation and the subsequent operational design is the resource model. Financial evaluation models such as the mostly used discounted (DCF) are based on the resource model information. A scenario offering the best financial returns through a satisfactory net present value (NPV) and or an internal rate of return (IRR) exceeding the hurdle rate, can be selected as the design option. The resource model, a representation of the orebody, is thus a primary input into project valuation, operational design as well as the investment decisions into the project.

The resource model represents a spatial distribution of mineralization, and is the basis for all value options that are considered during the design optimisation process. The mining schedules for tonnes and grade, and the subsequent financial analysis are based on the spatial estimate. To effectively evaluate a resource, the model must effectively represent the geoscientific information of the orebody, and variability of attributes and parameters that determine value of the resource. Over and above the *in situ* tonnes of ore and the associated grade, serious consideration should be given to other parameters that drive profitability, such as throughput rates determinants, mining and processing costs drivers and those that determine metallurgical recovery. Inadequacy of geological characterization and resource estimation imply inaccuracy of the spatial estimate and uncertainty of the resource. On the same note, failure to identify value determinants for consideration during operational design also introduces uncertainty. Resource uncertainty implies reduced confidence on financial outcomes of an investment, entailing risk emanating from the resource. Revenue projections are based on forecasting grade, throughput rate and recovery efficiency, and resource uncertainty throws off predictions of these parameters, thus introducing risk to the cash-flows. Metallurgical plants are designed specific to the orebody, bringing forth a process that will deliver the most desirable throughput rate and recovery efficiency to optimize the economics of the project. Changes in ore body attributes such as variability in the ore mineralization, texture, or weathering profiles, can have implications on the behaviour of ore in plant circuitry subsequently affecting process efficiencies and throughput rate. The operational design aims to optimize value recovery per unit time, and ore treatability challenges have adverse implications on the project costs and NPV. An improved understanding of the ore body and knowledge of the material characterization can reduce the risk due to resource uncertainty, or resource risk.

Geometallurgy is an approach that can reduce project risk, by integrating geology, operational design, mine planning and metallurgy with the aim of improving understanding of the resource. Complexities associated with ore body variability, are factored into the operational design providing further understanding of resource economics and minimizing risk through provision of flexibility options. Ore characteristics or material attributes that can affect throughput rate and process efficiencies are identified and built into spatial modelling, forming the basis of the geometallurgical approach.

Resource exploitation is both spatial and temporal in nature, and since operational design is done without full elaboration of the operating conditions, resource risk is a challenge to some mining operations. When uncertainties are realized during the life of the project, the operation should have ability to adapt and remain sustainable, and continue to deliver to the expectations of the shareholders. In order for the operation to maximize on earnings, it should have *flexibility* to reduce or avoid the impact of the risk. Thus, flexibility can be considered to be the ability or capacity of the operation to deal with consequential uncertainty. Flexibility can be built into the design, and the geometallurgical approach can inform the strategic options that can ensure profitability of the operation.

Metallurgical designs are based on the understanding of the ore characterization and ore dressing studies. Geometallurgical approach provides a platform for the metallurgical design team to further understand the spatial and temporal variability of the ore body. Metallurgical design decisions that either lack or do not pay due consideration to ore body knowledge tend to overlook flexibility requirements in the treatment plants. Bids to contain capital expenditure in plant construction can result in design decisions that eventually destroy operating flexibility and introduce constraints. On realization of resource uncertainty, project profitability will be at risk due to reduced throughput rates and protracted capital expenditures under the guise of process improvement initiatives. Such interventions, a result of inadequate metallurgical design decisions and geometallurgical involvements, will negatively impact on the NPV of the project.

This study reviews some mining projects such as the Cawse Nickel Project in Western Australia, the Voorspoed Diamond Mine in South Africa to illustrate geometallurgical risk at various phases of projects. The Cawse Nickel project is an illustration for inappropriate plant designs for the laterite ores, while the Voorspoed Diamond Mine demonstrates project ramp-up challenges due to inadequate geometallurgical understanding of the orebody. Canahuire Project in Peru and the Kemi Chromite and Ferrochrome mine in Finland are reviewed to exemplify application of geometallurgical programs. The projects are cited as illustrations of the value and importance of geometallurgy at various stages of the mining projects.

This report consequently ascertains a case for geometallurgy at Orapa Mine A/K1 deposit. From the extraction and analysis of process data, the report aims to demonstrate introduction of a constraint in the process due to a sudden realization of orebody characteristics not indicated in the resource model, illustrating existence of geometallurgical risk at this operation. New rock types, SVK_M and NPK_GG, were encountered as sub-classifications of the previously modelled geological domains and they introduced a constraint in the slurry handling section due to generation of non-settling slurries. The report also shows how metallurgical design decisions and misinterpretation of ore dressing studies results have limited the operation's flexibility to deal with the risk. As a result, a throughput risk currently exists and is threatening full realisation of forecasted cash-flows in the life of mine plan. A considerable injection of capital is required to remove the constraint existing in the slurry handling section of the plant.

2 LITERATURE REVIEW

The mining business environment is inherently risky. This is due to associated uncertainties that can be technical, political, environmental, social or financial in nature and can thus affect the overall economics of the project.

Geometallurgy is an important concept that aims to address the main source of technical risk in mining investments, the ore body. It is an integrated approach to enhancing ore body knowledge, by bringing together operational design, mine planning, geology and mineral processing. Its application on the strategic or tactical levels can reduce risk of protracted capital expenditure aimed at optimizing project economics due to consequential resource uncertainty. When inherent uncertainties of the business are realized during the life of the project, the operation should have ability to adapt and sustain operating margins, and continue to deliver to the expectations of the shareholders. Flexibility is the operation’s in-built ability to cost effectively deal with adverse uncertainty and remain profitable, as well as capitalize on positive changes in the operating environment to maximize on earnings.

2.1 Risk and uncertainty in mining

Historically, mining investments have demonstrated difficulties in predicting cash-flows. Ramani (1989) estimated that 66% of mining projects had failed to achieve projected cash-flows in given times, and he alluded this to the fact that though approvals of mining investments are a result of strict proposal evaluations, decisions are made in situations involving high levels of uncertainty.

2.1.1 Defining risk as a product of uncertainty

The international standard, ISO 31000 (2009), Risk Management – Principles and guidelines offers the definition of risk as the 'effect of uncertainty on objectives'. In the definition, notes are made to the effect of illustrating that:

(i) An effect is a divergence from the expected outcome
(ii) Risk is often characterized by reference to potential events and consequences
(iii) Risk is frequently stated as a combination of likelihood of an event and the consequences of the occurrence.
(iv) Uncertainty refers to inadequacy of information about the event, its possibility of occurrence and impact.

Macfarlane (2006) explains that the concept of risk can be summarized through posing two questions:

• What is the probability or chance of an occurrence (something happening or not happening)?
• What is the outcome or consequence of this if it does (or does not) happen?

This gives direction to the primary mathematical definition of risk:

Risk = Probability of occurrence x consequence of occurrence

Risk is generally defined as the probability of an outcome multiplied by its impact. Davies (1997) refers to risk as the expected value of an uncertainty and develops the definition to be represented by equation (1) below;

$R_i = P_i \bullet C_i$ ……………………… (1)

Where;

R_i Risk of aspect -i,

P_i Probability of occurrence of aspect -i,

C_i Consequence or impact of aspect –i, given occurrence P_i

Risk can be inferred as a consequence of uncertainty. Uncertainty in mining has internal and external sources. Kazakidis has illustrated internal and external uncertainties in mining operations using Figure 1 below.

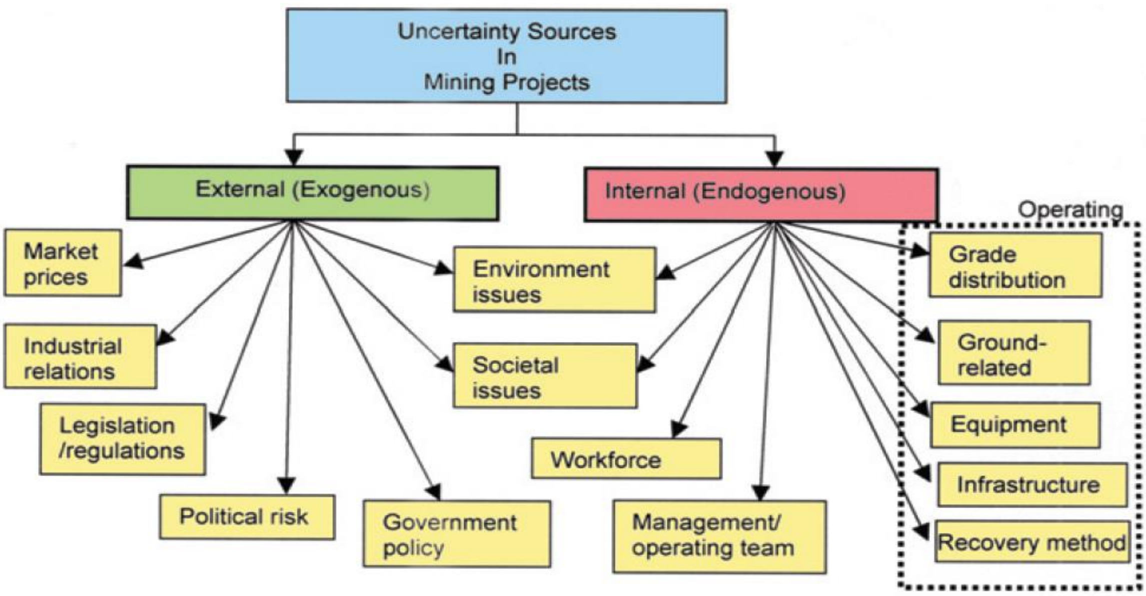


Figure 1: Sources of uncertainty in mining (after Kazakidis and Scoble, 2003)

External sources of uncertainty are predominantly to do with the market and economic factors in the operating environment, i.e. the metal prices, impact of exchange rates and inflation. Political events present political risk, government policy and legislation or regulations can impact on the trade policies, ownership and exploitation of the resource.

The internal sources of uncertainties include the resource model itself, as it is based on limited sampling information and also susceptible to estimation bias and errors. The mining methods, treatment methods and available equipment may present operational challenges at a certain point in the life of the mining project. Events such as strike actions, environmental incidents and decisions by the management or operating team may also present a certain level of risk to the project.

At the feasibility study phase, effort is put in to consider uncertainty during the development of a financial model of a project; a greater amount of it is spent on capital and operating expenditure estimations, the most appropriate discount rate and commodity price forecasts. More often than not, this is at the expense of uncertainty consideration of the most critical internal source of uncertainty, the primary asset of the investment, the resource, which sometimes gets much less attention. (Day *et al*, 1999). The mine planning and financial evaluation teams work on the assumption that the resource model is reliable and accurate, and thus due consideration is not paid to the levels of uncertainty and the fact that the model is an estimate.

2.1.2 Risk factors in mining projects

In order to be able to understand the risks that mineral projects have to deal with, it is important to distinguish the varying risk types. Broadly, risks can be differentiated into systematic and non-systematic risks.

- **Systematic risks** are coupled with the behaviour of financial and capital markets. They relate to market uncertainty, and the way that they impact on the cost of capital. Though knowledge of such risks can influence decision making on the project, the risks are not under direct control of the mineral resource management team. However, deep understanding of these risks and sufficient knowledge of how they can be mitigated will offer intelligible approach to the adjusting of the discount rates in terms of the capital asset pricing model, the after tax cost of debt, weighted average cost of capital and other aspects affecting financing options of the investment. (Macfarlane, 2006).

The systematic risk usually refers to the risk of financial performance of the venture. The risk relates to project's inability to payback interests and capital from the cash-flow. The risk sources can be the financier or external uncertainties such as increases in interest rates, market collapses, unfavourable exchange rate regimes, and commodity price fluctuations. The usual practice, in general, is to account for the financial risks in the financial arrangement and subsequently factor it in the cost of capital. This approach, however, does not adequately address the risk factors associated with that particular operation. (Macfarlane, 2006).

- **Unsystematic risks** are associated with the inputs to the business, and unlike systematic risks these are specific risks that can be understood and treated at source. Macfarlane emphasizes that 'it is *not* appropriate to deal with these through adjustments to discount rates', but rather mineral resource management practitioners should have tangible mitigation plans that demonstrate understanding of these risks.

Unsystematic risks are technical in nature, and a risk management strategy should be in existence to identify and manage such risks. Various techniques are available to identify, prioritize and mitigate the risks to acceptable levels. Minimal residual risk can then remain, which can be predominantly systematic risk. (Macfarlane, 2006).

The table below shows some of the risk factors encountered in mining investments.

Table 1: Risk Factors in mining (after Kazakidis, 2001)

Evaluation Parameter	Risk Considerations	Classification/
Geotechnical	Lithology	Unsystematic risk/ Technical Risk/ Operational Risk
	Geophysics, Groundwater	
	Ore genesis	
Mineral Occurrence	Continuity of ore zones	
	Mineral occurrence within ore zones	
	Economic mineral occurrence within ore zones	
Ore body configuration	Dip, plunge, size, shape	
Economic	Minable ore tonnes, Orebody grade	
	Mineral value	
	Capital & Operating costs	
Safety/ regulatory	Degree of mechanization, labour intensity of method	
	Ventilation, refrigeration	
	Ground support requirements	
	Dust controls, noise controls, gas controls	
Environmental	Groundwater contamination	
	Subsistence potential, Dust controls, noise controls	
Political	Labour costs, political influences	Systematic risk/ Financial risk
Market dynamics	Commodity price fluctuations	
Macro-economic conditions	Taxation, Exchange rate regime, Inflation, interest rates	

2.1.3 Resource risk

In an operating environment, risk can be defined as a measure of probability and consequence of not achieving a goal. (McGill, 2005). Mining projects are inherently risky since the financial returns cannot be guaranteed, due to their high capital intensity, high operating costs, volatility of commodity markets, political risk and the uncertainty associated with the underlying asset, the mineral resource. The traditional approach to valuing projects is to use the discounted cash-flows (DCF) method to determine the NPV and the IRR. Under uncertainty, risk considerations can be done either through risk adjusted discount rate approach, certainty equivalent approach or the capital asset pricing model (CAPM) approach. (Kazakidis, 2001). The pre-feasibility stage of the project considers various scenarios to develop a business case for exploiting the deposit. Though some uncertainty related to the mining project can be dealt with through design improvisations, in most cases this is inadequate to fully address all risk considerations.

For resource projects, uncertainty or risk, can be demonstrated by the difference between the predicted cash-flows and the realized cash-flows. (Deraisme, Farrow, 2003). Project cash-flow projections are developed from the optimized mine plan, with the resource model as the primary input. The key assumption is that the model is reliable; this however, is rarely the case. (Bertoli, Jackson, Vann, 2002). A typical project optimisation process comprises thorough geological and operational modeling. Within the context of a set of assumptions, key controllable variables are identified. The variables are held in various combinations, to identify an option that offers maximum value. The resource or block model is now considered a ‘reality’ or to be unchanging at this stage. Metallurgical testing sampling plans, throughput and engineering designs are prepared based on the model. (Hanson, Stange, Whittle, 2007).

Realistically, since an estimate is being considered, it implies that a degree of uncertainty does exist in the optimized mine plan. (Dohm, 2003). The uncertainty of the resource is a result of limited geoscientific information on the orebody, leading to limited understanding of the geological characterization, and reduced confidence in quantification of tonnages, grades, and metal estimates. Unforeseen geotechnical conditions can disrupt the mining sequence leading to prolonged delays and or failure to adhere to plan.

In the case of grade and tonnages estimation errors, application of cut-off grades may thus lead to misallocation of material. Ore can be misallocated as waste or sent to low grade stockpiles, and vice-versa, waste can possibly find its way to the treatment plant.

Uncertainty on the geological structure results in poor guidance to grade control, pit optimisation and prediction of orebody geometry and orientation, resulting in the operation being constrained by shortage of payable face. Ore dressing studies and material characterization inform design options for the treatment plants. Uncertainty in geological material characterization implies that rock types can possibly be encountered during the life of the project, which may present treatability challenges to the processing plants. Expected recovery efficiencies may not be achieved, due to erroneous expectations dependent on the rock type mineralogy and the subsequent plant design. (Berry, McCarthy, 2006).

Consequently, decisions that are made based on the analysis of geoscientific information have major impact on the operational success and financial viability of the project, as well as operational safety and sustainability. Broadly, geoscientific inputs and interpretation of the information are necessary for:

- Orebody delineation, that is, interpretation of tonnes, grade, rock structure, continuity, density, quality, hardness and other important parameters required for the process.
- Design of mining methods and associated design parameters such as development, production rate, stability and rock reinforcement and dilution.
- Ore processing plant design methods and associated parameters such as crushing, grinding, solid-liquid separation, recovery, and deleterious products.
- Design of materials handling techniques such as ore and waste transportation.
- Design of waste disposal methods, rehabilitation and environmental management. (Berry *et al*, 2006).

Geological or geoscientific data present the basis for much mine design, short- to long-term planning, plant design, operations management and sustainability, and estimation of capital and operating costs. (Berry *et al*, 2006).

Accordingly, geological information provides essential inputs into decision making, risk assessment and risk management. Once critical decisions have been made on erroneous information, the operation should have the flexibility to react so that it remains sustained through effective management of constraints, and be able to maintain desirable profit margins. This is the concept of flexibility, and means that the operation can recover well in time through availability of options to react to the project risk.

2.1.4 Risk profile across mining project life

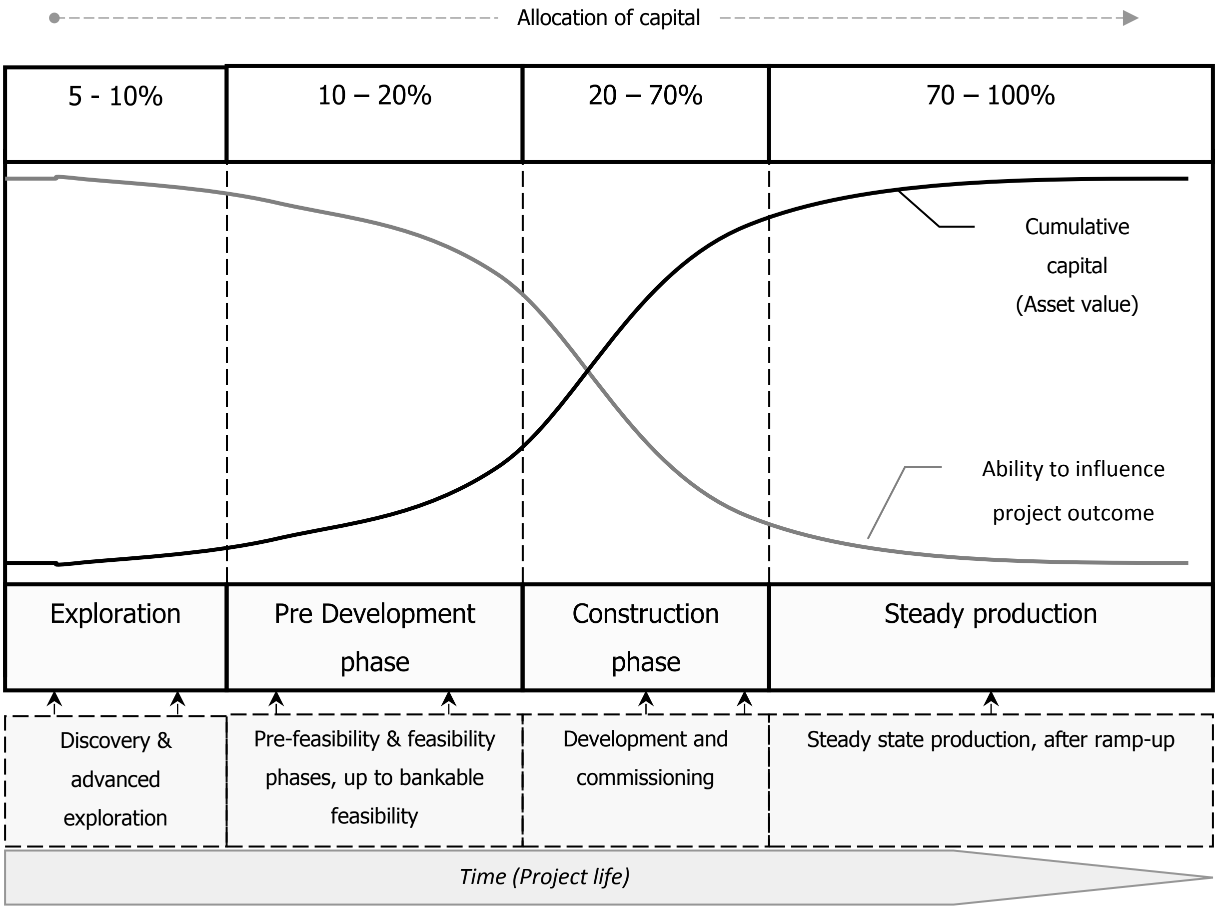


Figure 2: Risk profile across mining project life (after Macfarlane, 2006)

Inherent risk can be considered to be the original risk of an entity in the absence of controls. The inherent risk of mining is gradually reduced as information becomes available and controls and mitigation measures are put in place from the exploration exercise, pre-feasibility and feasibility stages and eventually during the production/ operational phase. The risk remaining after the controls and mitigation measures are in place is the residual risk.

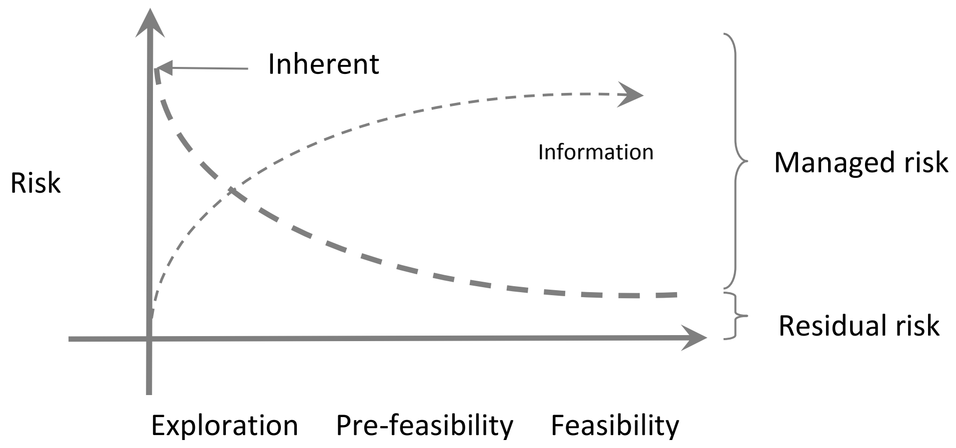


Figure 3: Risk profile across mining project life (Inherent and residual risk)

Flexibility can be considered to be an inverse function of risk, and implies ability to deal with residual risk. It necessitates cost benefit analysis in assessing options to deal with constraints created by uncertainty, and optimum methods of taking advantage of upside risk.

Value is created by being able to deal with downside risk cost effectively, and failure to capitalize on upside risk becomes an opportunity cost. To justify need of flexibility, major areas of uncertainty and their potential impacts need to be understood.

2.1.5 Management of risk in resource projects

The ISO 31000 (2009), Risk Management standard recommends organizations to establish risk management frameworks with the purpose of integrating the risk management processes into the governance structures, strategic planning, reporting systems, policies, values and culture. The risk management process entails systematic application of policies and procedures, methods and practices to actions that seek to communicate, analyse, identify treat monitor and review risks. The risk management framework is the mechanism that offers the basis and structures for designing, establishing, monitoring, reviewing and continuous improvement of risk assessment throughout the enterprise.

The standard also offers the principles or values of risk management, and diagrammatically demonstrates how they are linked to the framework and processes as shown below.

- *Creates value*
- *Integral part organizational processes*
- *Part of decision making*
- *Specifically addresses uncertainty*
- *Structured, systematic and timely*
- *Dynamic, iterative and responsive*
- *Facilitates continual improvement and enhancement of organizational value*

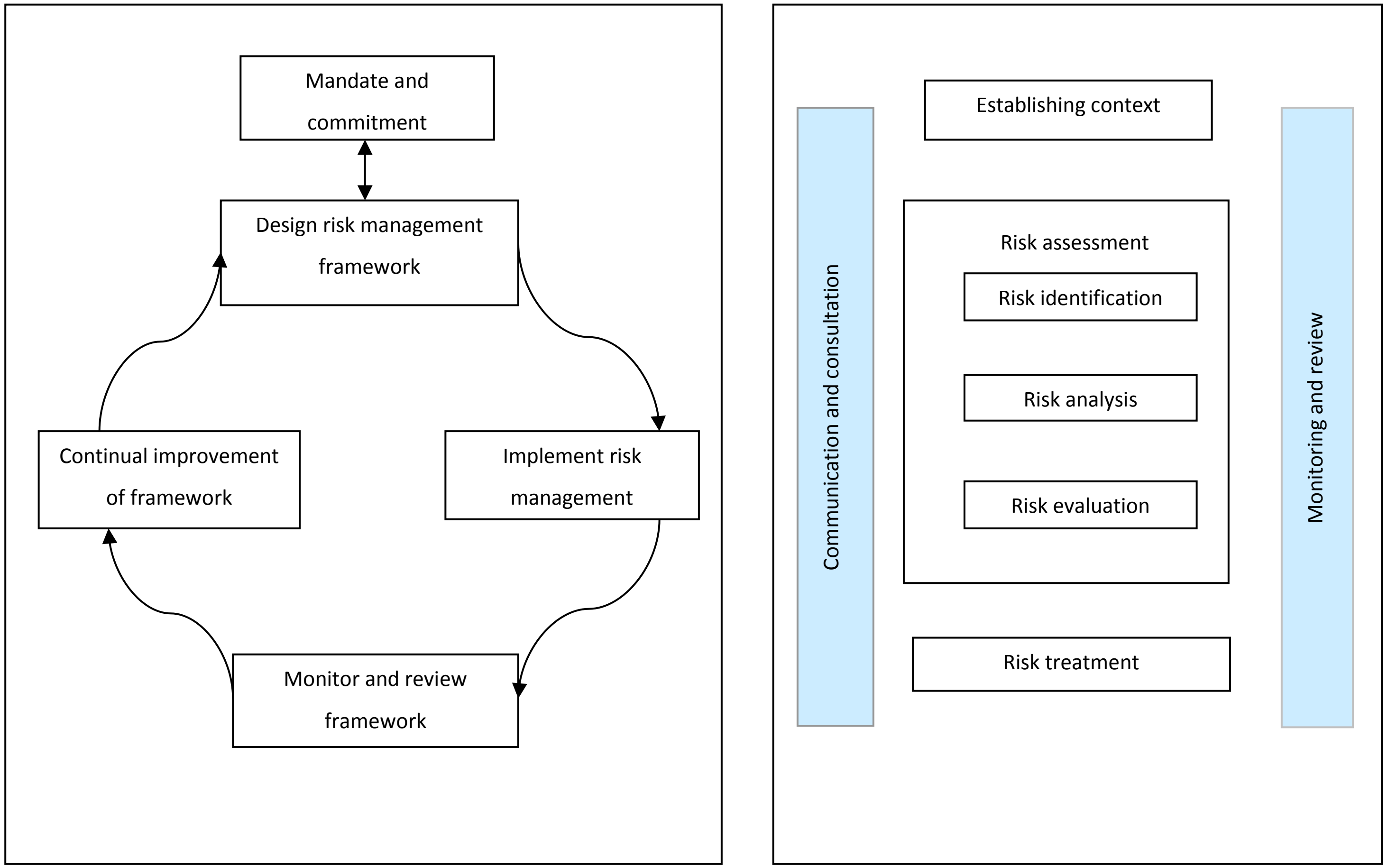


Figure 4: The risk management process (after the 2009 Risk Management Standard, ISO 31000)

Risk management should be considered to be a dynamic, rather than a static set of activities. As illustrated in figure 4 above, proper implementation of the risk management cycle should commence at source and become part of an iterative process that aids decision making.

For risk associated with the resource, it is imperative that risk is understood at source and it becomes an integral part of processes, and that systems that embed the risk management framework are utilized within the enterprise. Risk quantification is crucial to enable decision making, and a variety of methods are available for this purpose. Some of the quantitative resource risks that are available are (McGill, 2005):

- *Sensitivity analysis* assesses the sensitivity of the outcome (e.g. NPV) to variation in the inputs such as capital costs, grade, throughput, operating costs, commodity prices, and macro-economic indicators such as inflation and exchange rate.
- *Classical statistical techniques* Applied in the understanding of resources and reserves, geostatistical tools are applied to predict and infer probable grade values at points that are not sampled.
- *Monte Carlo simulation* predicts likely outcomes through random variation of inputs values. The Monte Carlo simulation creates probability distributions for potential risks associated with the investment and thus realistic outcomes can be visualized. The accuracy of Monte Carlo simulation depends on a truly representative resource model, realistic estimate and clearly defined objectives of the analysis (Heuberger, 2005).
- *Conditional simulation* realizes that resource exploitation is both spatial and temporal. The technique works out or predicts how the deposit might vary in terms of tonnages and grade through 3D-simulated models of reality. Conditional simulation works out probable variations of the resource, and thus illustrates risk at source.
- *The discount rate (DCF models)* DCF models are used to evaluate the rate of return and net present values of a project or competing projects, and are used as a basis for decision making. To demonstrate levels of risk within a project, some practitioners include a factor into the discount rate for the DCF model. The discount rate will include

aspects such as risk-free interest rate, mineral-project risk (represents numerous risks that are associated with the resource), and county risk (represents political risk, policies and governance concerns within the host country for the project). (McGill, 2005).

- *Decision Tree Analysis* Can be used to calculate probabilities of outcomes; it is a decision support tool that uses tree-like graphs of outcomes and their possible consequences. The tool can guide decision making.
- *Real Option Analysis/ valuation (Modern Asset Pricing techniques)* the techniques recognize that management has the ability to change the outcome of the project through decision making and ability to identify opportunities. Unlike the DCF which assumes a 'passive' path, real options analysis or the modern asset pricing techniques recognize management 'flexibility' in making decisions. (Macfarlane, 2006).

Risk assessment involves risk identification, risk analysis and evaluation (quantification). Complete risk management process should entail ability to treat the risk such that there is minimal impact to the enterprise's objectives. The ability to deal with downside risk and also to capitalize on upside risk can be referred to as flexibility.

2.2 Operating flexibility

Uncertainty in mining ventures makes it difficult to guarantee economic performance of the investment. This is due to limited information in the resource, volatility of mineral markets that brings along challenges to precisely predict future revenues, political risks in some countries of operation, high operating expenditure and other uncertainties in the operating environment. There is a necessity for flexibility to strategically and tactically deal with risk, to ensure profitability and overall success of a mining project.

For mining operations to meet the investment returns expected by the shareholders there should be ability to adapt to changes that are a result of the inherent uncertainty in the business. The operation should have in-built ability to cost effectively react to adverse consequential uncertainty and remain profitable, as well as capitalize on positive changes in the operating environment to maximize on earnings.

At strategic decision making and design stage, flexibility should be carefully considered and built into the operating system for future capability to deal with future risks. At the tactical stage, there should be ability to put into effect available operating options depending on the unveiled situations or risks. Flexibility options should be reviewed by cost benefit deliberations. Various alternatives can be evaluated by assessing the costs of a particular flexibility option and the impact it will have in terms sustainable value creation.

2.2.1 The meaning of flexibility

In the context of engineering design, Wikipedia defines flexibility as the *"ability of a system to respond to potential internal or external changes affecting its value delivery, in a timely and cost-effective manner. Thus, flexibility for an engineering system is the ease with which the system can respond to uncertainty in a manner that sustains or increase its value delivery. Uncertainty is a key element in the definition of flexibility. Uncertainty can create both risks and opportunities in a system, and it is with the existence of uncertainty that flexibility becomes valuable". (http://en.wikipedia.org/wiki/flexibility_(engineering), 1984).*

In the mining context, flexibility can therefore be defined as the ability of a mining operation to respond to potential internal or external changes affecting its capability to deliver value, in a timely and cost-effective manner. Flexibility of a mining operation is the ease with which it can respond to uncertainty that results in upside or downside risk in a manner that will sustain or increase its value delivery.

Kazakidis explains that need for flexibility is not only to act as "insurance" against adverse production performance but also to enable the management team to take advantage of opportunities that may develop during the life cycle of the operation. Flexibility is thus capability or ability of a mining operation to sustain performance, conserve a desired cost structure and operating margins, adapt to internal and external changes in operating conditions, and take advantage of opportunities that arise during the life of the project by adjusting operating parameters. This implies management has the opportunity to improve decision-making in an uncertain environment with operating constraints. (Kazakidis et al, 2003).

Hence, flexibility is the planned capacity to accommodate uncertainty or change, and an in-built ability to take tactical advantage of the ever changing situations in a mining environment. Flexibility can be the ultimate decider of profitability and success of a mining project.

2.2.2 The need for creating flexibility

Recognition that there should be ability to react to technical, financial as well as social changes that exist in the dynamic modern business operating environment justifies the need for flexibility in a mining operation. (Minnitt et al, 2007). The inherent risk of mining is gradually reduced as more information becomes available in the project, and as controls and mitigation measures are put in place. This occurs as the project progresses through the exploration exercise, pre-feasibility and feasibility stages and eventually during the production or operational phase. The risk remaining after the controls and mitigation measures are in place is the residual risk.

Flexibility can be considered to be an inverse function of risk, and implies ability to deal with residual risk. It necessitates cost benefit analysis in assessing options to deal with constraints created by uncertainty, and optimum methods of taking advantage of upside risk.

Value is created by being able to deal with downside risk cost effectively, inversely; failure to capitalize on upside risk becomes an opportunity cost. To justify need of flexibility, major areas of uncertainty and their potential impacts need to be understood and these include the resource or mineral asset, the behaviour of the markets, macroeconomics, and operating performance.

2.2.2.1 Resource uncertainty

Uncertainty of the resource is a result of limited geoscientific information. Misallocation of material can result in lower than planned development and production rates, unplanned dilution, reduced metallurgical performance and increases in capital/operating costs. These challenges are directly attributed to decisions made on the basis of inadequate geoscience interpretations and estimates. Historical examples of complications arising from insufficient orebody knowledge include:

• The Mt Keith Nickel Mine in Western Australia: Metallurgical performance in the flotation circuits due to response challenges associated with high talc content in certain ore zones in the early days of the project. This led to extended ramp-up period for the project.

• Horn Island Gold Mine, Queensland: Mining difficulties due to discontinuous mineralization and the ever changing or transition ore surface resulting in unexpectedly higher dilution levels, and over 80 per cent of the ore was being recovered at 65 per cent of the originally expected grade. The mine was closed down in 1989 due to poor performance.

• Big Bell Mine in Western Australia: Unexpectedly high hardness of waste in the pit led to increased mining costs and, lower than expected grades were encountered leading to premature closure 2003.

The above examples illustrate how insufficient knowledge of the orebody can have significant financial impact on a company’s bottom line. Once critical decisions have been made on erroneous information, the operation should have the flexibility to react so that it remains sustained through effective management of constraints and ability to maintain desirable profit margins.

Unforeseen geotechnical conditions can disrupt the mining sequence leading to prolonged delays and or failure to adhere to plan. Flexibility in such a situation means that the operation can recover well in time through availability of options to change the mine plan and still remain profitable.

2.2.2.2 Market uncertainty

Justification of mining projects is usually based on metal price forecasts. Fluctuations imply uncertainties in future market prices and can have a sudden unexpected impact on the project expected cash-flows.

In times of unfavourable metal prices, there is need to increase tonnage throughput and take advantage of economies of scale operating at lower pay limits, for resources with low grade variability. Throughput constraints or bottlenecks should be removed, or optimisation should be done through the constraints. Grade variability in such situations calls for close management of cut-off grades and this can be at the expense of volume and economies of scale. (Woodhall, 2007). In either case, focus should be on cost containment and protection of margins. Favourable prices will dictate need to avail ore reserves. Active management of future reserves or a healthy and on-going development program is needed to keep mining reserves available. The strategy will be to take advantage of the opportunity of high prices and to sustain profits. In the event of high grade variability, the mining mix should be managed. (Woodhall, 2007). An optimum exploitation strategy and dynamic cut-off grade optimisation will reduce the risk of losing consistency in the extraction sequence.

Though hedging strategies have been adopted to guard against unfavourable price trends, shorter-term contracts have become a preference in certain sectors to be able to benefit from the upward price trends. Shorter term contracts also provide flexibility in the event of interruptions of the production plan due to resource risk or equipment breakdowns. Operations that do not have flexibility will fail to adopt the above strategies in order to deal with market changes.

2.2.2.3 Production uncertainty or operating constraints

Unplanned equipment failures can be catastrophic, leading to long production delays. If this happens at the mine, then there are ore shortage delays to the treatment plant. Breakdowns at the treatment plant mean that mining activities have to slow down as there is limited processing capacity. In these situations, redundancy and optimum maintenance practices will reduce the impact and frequency of breakdowns. The process can be constrained from the design stage. In some cases the treatment plant or sections within it are under-sized and the plant fails to cope with the mine's ore production rate and meeting the desired targets. In other cases, the mine fails to supply as per the demands of plant capability resulting in ore shortage delays. This can occur as a result of sub-optimal cycle times for the mining fleet, under sized equipment and low fleet availability. In either of these cases, constraints should be identified and removed through implementation of flexibility options; if costly then there should be optimisation through the constraints. Building in flexibility from the design phase will reduce the impact and occurrences of equipment and infrastructural failure. Surge capacity and control functions are a critical consideration during design to be able to deal with inconsistent material flow or constraints across the value chain.

2.2.2.4 Uncertainty in operating Environment

As previously illustrated in Figure 1, sources of uncertainty can be either external or internal and this section takes a more general approach to their discussion. Issues such as industrial relations, legislation, government policies and social responsibilities can impact on the smooth running of the operation. Political risk should be taken into consideration when assessing viability of the project. All these factors constitute the operating environment and can cause disruptions if they are not understood or managed.

2.2.2.5 Uncertainty in Financial Evaluation (using DCF)

The DCF assumes that all blocks will be extracted and be treated during a certain period of time, and will on average have the same technical risk and that the cash-flow streams are produced at the exact times. No room for management intervention, active asset management or technical risk associated with the production process is in-built adequately in the financial model. Assumptions are made on some parameters in the financial evaluation models such as exchange rate, metal prices and the inflation rate. Understanding limitations of the DCF method when valuing mineral projects and forecasting cash-flows will justify need to build in flexibility due to the uncertainty attached to the NPV. Risk analysis using Monte Carlo Methods attempt to quantify confidence to the NPV values. Monte Carlo might have a limitation in that it may not adequately realize risk emanating from sudden impact events.

The main concern with the DCF is the single solution that it offers, and having closely related options developed around the model. An alternative is the rapidly developing stochastic analyses, where key variables are identified and their sensitivities are tested. (Jager, 2005).

2.2.3 Creating Flexibility

Flexibility is a matter of cost benefit considerations. On creating flexibility, its cost must be ascertained versus its benefit. Flexibility thus improves the value of a mining operation.

Expanded value of operation = DCF value + value provided by flexibility

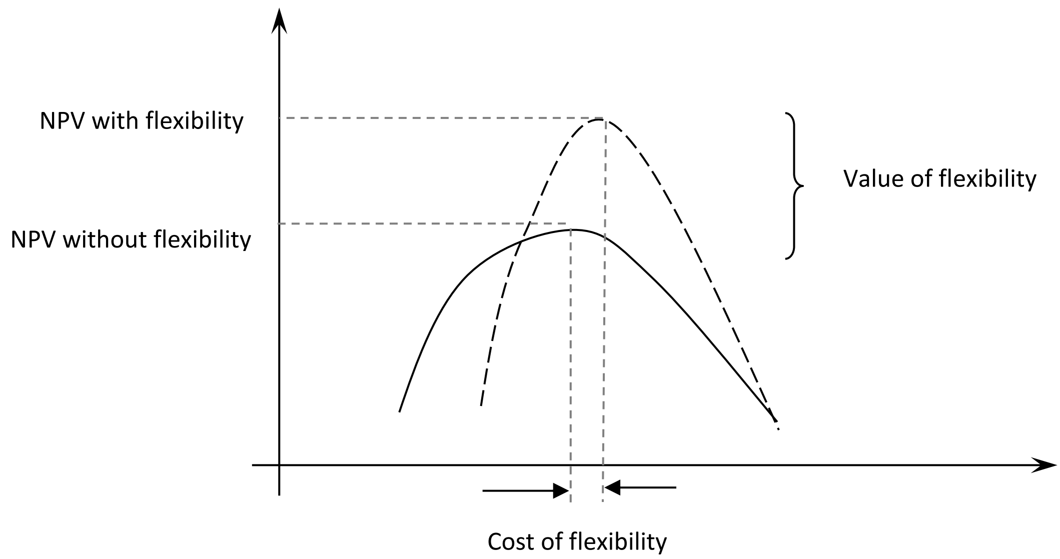


Figure 5: Valuing flexibility

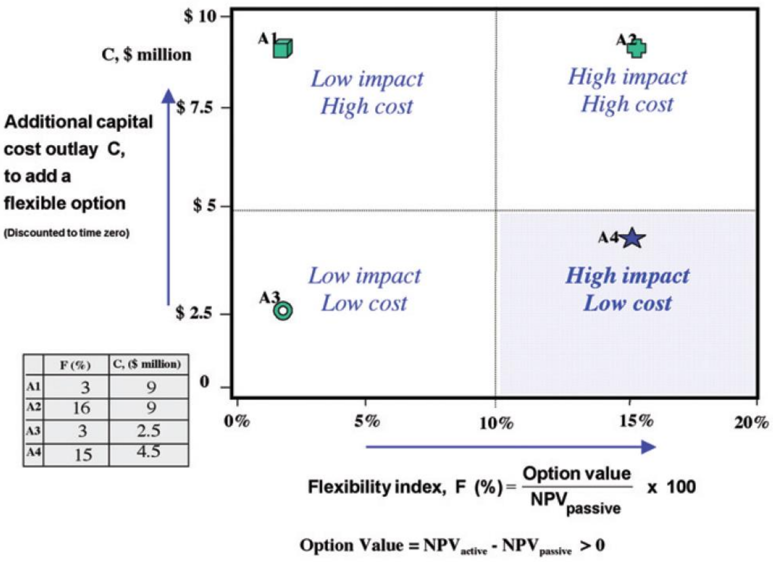
Figure 5 above is an example on valuing flexibility, with the horizontal axis representing the cost of flexibility and the vertical axis showing the project NPV. For the optimum flexibility value, the change in NPV should be zero at incremental changes in cost of flexibility. A suitable option for flexibility should have minimal costs, and on the other hand, should demonstrate a significant increase in the project NPV.

To assess value of flexibility alternatives, a flexibility index was proposed by Kazakidis *et al*.

Flexibility index, F(%) = {Option value, OV}/ {NPV passive} x 100%, OV>0

A flexibility index value of 25% would indicate that the introduced flexibility option would improve the NPV of the base case by 25%. A flexible alternative is often associated with capital and/or operating costs that have to occur for the particular alternative to be active throughout the project. (Kazakidis et al).

Decision making on which flexibility option to adopt takes into consideration the impact the particular option will have in terms of value, i.e. the flexibility index, and the costs of that option which may be constrained by the available budget. Kazakidis *et al* introduced a matrix that can be utilized in decision making on considering alternative flexibility options, and it is shown in figure 6 below.



	F (%)	C, ($ million)
A1	3	9
A2	16	9
A3	3	2.5
A4	15	4.5

Figure 6: Classification of flexibility options using the flexibility index (after Kazakidis and Scoble, 2003).

The ultimate option will be a low cost and high impact option.

2.2.4 Creating Flexibility in Project Financing and Scheduling

The project management concept of 'triple constraints' describes how limitations on costs, time or quality during project execution eventually impacts on the final project delivery. (Project Management Body of Knowledge Guide, 2000). In the case of a mining project, budgetary constraints can delay the project and compromise on the required operational capability which will directly impact the projected cash-flows. The constraints can be managed through inclusion of physical contingency in the schedule, and financial contingency in the capital and operating expenditures' budgets. Contingency accommodates impact of price changes and capital overruns. Progressive elaboration of project scope normally reveals some detail that requires additional work. Failure to cater for such requirements implies either settling for a substandard project that will not deliver on all the intended requirements, or protracted delays on the project which will affect the cash-flow. Lack of contingency in planning and financing the project capital will constrain the project scope making it vulnerable to price fluctuations and technical risks. Contingency is a form of flexibility in financing and scheduling a project.

2.2.5 Creating Flexibility to Deal With Resource Uncertainty

Resource aspects that can affect delivery of the mining operation or project are:

- The outline or delineation of the deposit.
- Grade variability and uniformity of mineralization. (Kazakidis, 2001)

The outline of the deposit relates to the continuity of the ore zones, geological structures and occurrence of distinct rock types constituting the ore body. Inadequate knowledge of the orebody can result in unforeseen geotechnical conditions that may impact on the delivery of the mine plan in terms of production levels and or costs.

Grade variability relates to spatial distribution of mineralization in the deposit. An ore body with high variability has no uniformity and presents difficulties in estimation. Methods of creating flexibility to deal with resource uncertainty can include:

- Designing a plant that can deal with treatability problems presented by major rock types existing in the ore body. Such problems may include ore hardness, excess generation of fines and high clay content that may cause slurry handling challenges.
- Prediction of dilution and oversize with measures and controls to mitigate against the problem. (Kazakidis, 2001).

- Ability to blend various rock types.
- Ability to maintain development ahead of stope production. (Kazakidis et al, 2003).
- Availability of alternative stopes ('back pocket stopes') or stockpiles in the case of open pit as alternative ore sources. (Kazakidis et al, 2003).
- Creation and use of alternative mine openings and faces. (Kazakidis et al, 2003). In the case of open pit, this implies multiple access ramps to deal with emergencies or failures.
- Ability to build stockpiles through mining slightly higher than the treatment capacity (surge capacity and control). This will enable the plant to run if there is inability to deliver ore from the mine.
- Ability to blast out of sequence. (Kazakidis et al, 2003).

2.2.6 Creating Flexibility to Deal With Production Uncertainty or Constraints

Production uncertainty in this context refers to downtime or delays that can be caused by equipment failures in the treatment plant or at the mine, constraints, as well as work stoppages that may arise from safety and or environmental incidences, job actions, permitting issues or directives from the state (in the case of high political risk countries). Such stoppages imply failure to supply the market and 'drying-up' of revenue streams during these periods. Flexibility implies ability to operate sections of the value delivery system to supply the market. Interventions that create flexibility to deal with production uncertainty include:

- Optimized maintenance systems to minimize delays.
- Building in redundancy in the plant, that is, optimized number of trucks by considering achievement of minimum cycle times, adopting modular designs in treatment plants and having multiple streams.
- Multiple mining faces and access ramps.
- Equipment spares availability, maintaining strategic inventories.

- Building strategic stockpiles to ensure ore availability in events of failure to mine. Metallurgical plant designs that have bins or stockpiles that will save a ‘surge-capacity’ function.
- Budgetary contingencies.
- An adequately skilled labour force.
- Optimized and realistic mine schedules or plans that are informed by process and orebody capabilities.

2.2.7 Creating Flexibility to Deal With Market Volatility

Fluctuation of commodity prices requires an operation to have a cut-off grade strategy, ability to reduce or increase throughput and to manage the mine mix. Interventions that create flexibility to deal with mineral price fluctuations include:

- Ability to conserve cash and shrink the cost base. This can be done through maintaining a flat organizational structure with a small proportion of fixed costs to variable costs. When a decision is made to reduce production, the benefits will also cascade to costs and margins are preserved. Organizations with a high proportion of fixed costs are likely to encounter reduction of margins and are at risk of making a loss when production is reduced.
- Consideration of modular designs in the treatment plant will enable sections of the plant to be operational whilst others are on care and maintenance. This gives flexibility to cut back on production when there is need thus reducing costs.
- Ability to ramp-up in time to benefit on upward trends in metal prices. This requires good maintenance practices and redundancy thus having a knock on benefit on equipment availability.
- Options should be available to process high grade or low grade ore. Cut-off grade strategy should be in place.

- There should be ability to manage mine mix, with ability to select particular ore types depending on access, treatability and grade.
- Managing reserves, ability to maintain development and avail reserves in times of increasing prices.
- Hedging strategies have been adopted by most companies as a strategy of safe guarding against the effect of commodity price fluctuations. The strategy of hedging, however, is not appropriate for all businesses. For instance, a robust and secure production plan will be required for a company with a hedging strategy to ensure security of supply, as compared to one selling all production to the spot market.

A commodity boom may disadvantage companies with hedging strategies or long-term trading contacts as was the case with iron ore. After the 2008-2009 global economic crisis, major iron ore producers have opted for selling on the spot market and negotiating on much shorter contracts, significantly changing the way iron ore has been traded for decades. (Blas; Financial Times, 2011). Shorter term contracts build in flexibility, providing market security when prevailing prices are unfavourable, as well as ability to benefit from rising spot prices.

In the event of resource risk or an inadequate cut-off grade strategy, shorter term contracts provide opportunities for recovery when market conditions improve. Major iron ore producers such as Vale and Rio Tinto are working with three month contracts, with BHP Billiton Iron Ore negotiating even shorter contracts to maximize on the boom in iron ore due to growing demand from China and India. (Blas; Financial Times, 2011).

2.2.8 Maintaining and Controlling Flexibility

Flexibility is a strategic issue as much as it is a tactical issue. Ability to maintain and control flexibility lies in the decisions that are made during planning. Planners’ decision making is often constrained by budgets and corporate directives from head office. To deal with corporate constraints, multi-disciplinary teams that include geology, mine planning and metallurgy should be involved in the planning process. This will assist in communicating capabilities of the operation so that they are fully understood at the corporate or strategic levels. This integrated planning approach will ensure alignment of long term goals to the short term and will discourage a volume or profit driven approach at the tactical level. At the same time, frequent demands for 'more' are minimized and realistic targets are set, and resources availed to maintain short and long term profitability. Adopting economic profit measures such as the economic value added (EVA) will align the short and long term measures of profit and NPV. This will ensure that the available flexibility is not 'eroded' for short term gain at the tactical level. (Macfarlane, 2006).

In most cases, reward systems and incentives 'drive bad behaviour' such as high grading, over feeding the plant or over production, foregoing maintenance and cost saving at the expense sustainability programs. Managers should be rewarded on measures that drive both short term profitability such as achieving set production targets or cost containment, and long term measures such as EVA and ability to mine according to plan.

The role of planning in flexibility cannot be over emphasized. A mine plan must have adequate flexibility to accommodate uncertainties mentioned above and still meet the goals of the project as defined by production planning, economic forecasting, manpower and equipment availability. Most methods mentioned earlier on creating flexibility can be included in the mine design and plans. Controls and governance structures can be put in place to ensure adherence to plan, and prohibit unsustainable profit driven interventions that erode flexibility.

Though most companies do have integrated planning in their organizational structures, its application is ineffective and cross disciplinary corporation remains inadequate. A geometallurgical function, however, will enable interaction of the various disciplines and enhance understanding of the ore body. Geometallurgy can maintain flexibility by informing the design and understanding the requisite controls necessary to deal with risks.

2.3 Geometallurgy

Geometallurgy is a rapidly evolving discipline, principally focused on enhancing business performance for mining projects through improved understanding of the mineral asset, better design and operational practices. It is an approach that aims to combine geology, metallurgy and mine planning, so as to improve value recovery that is usually lost through disparate functioning of these technical fields. If successfully implemented, the geometallurgical approach has the capability to match operational design, mining and mineral processing development options to be more specific to the resource and its characteristics, leading to lower operating costs and better recoveries.

2.3.1 Defining geometallurgy

The following definition of geometallurgy is stated in Wikipedia;

'Geometallurgy relates to the practice of combining geology or geostatistics with metallurgy, or, more specifically, extractive metallurgy, to create a spatially- or geologically-based predictive model for mineral processing plants. It is used in the hard rock mining industry for risk management and mitigation during mineral processing plant design. It is also used, to a lesser extent, for production planning in highly variable ore deposits.' (Wikipedia, 2012).

Though the above definition does capture the essence of geometallurgy, the concept is much broader as it is about improving the understanding of spatial nature of the pertinent rock properties that have an impact on value recovery. Once the spatial nature of such properties or variables is understood, the problem of predicting physical and financial outcomes in the treatment process is simplified. There is reduction in the number of ‘unknowns’ enabling operational designs that are more specific to the orebody, and there is improved decision making in the value generation process. (Coward, Dunham, Stewart and Vann, 2009).

The pertinent rock properties affecting value recovery can be referred to as geometallurgical variables and these include any rock property that has a positive or negative impact. Examples of some of the more critical geometallurgical variables include hardness, density, friability, mineralogy and content of deleterious materials. These variables determine the treatability of the orebody influencing project costs and revenues. Geometallurgy, in effect, has the capability to positively influence value-adding strategic and tactical decisions across the mining value chain.

The integration of geology, mining planning, operational design, mineral processing and metallurgy improves the fundamental understanding of resource economics, making the value proposition for geometallurgy undeniable. Geometallurgy does not aim to replace existing methods to design and optimisation of mining and mineral processing, but rather complement and synergise these approaches. The prime objective is to offer inputs that reveal inherent geological variability and subsequent impacts on metallurgical performance. The idea of incorporating geometallurgical variables or parameters into resource modelling supplements existing geology and tonnage/ grade-based attributes, enabling an all-inclusive approach to economic optimisation of mineral projects. (Walters, 2009).

2.3.2 Geometallurgy, geostatistics and resource models

The understanding of the resource is typically captured in the form of three dimensional or two dimensional models. Generally the resource model shows tonnes above cut-off, grade above cut-off and the spatial distribution of tonnes and or grade above cut-off.

Considerations of other factors affecting value recovery such as treatability, dilution, ore loss and metallurgical recovery are applied to the resource model as modifying factors for evaluation and determination of profitability of the project. Over and above depleting the resource at an optimum combination of grade and tonnes, serious consideration should be paid to throughput rates, mining and processing costs, concentration of deleterious elements and metallurgical recovery. (Dunham *et al*, 2007).

The accuracy of estimation of the mineral resource depends on the integrity of the sampling data such as density and volume, as well as the soundness of the geological interpretation. The geological data must be sufficient to define a geological model, and sufficiently show the lithological and mineralogical domains. Once the geological limitations are understood, a resource model is then generated. Various geostatistical methods exist for grade interpolation, and the techniques include nearest-neighbour approaches, area of influence and kriging. The resource model thus defines the spatial distribution of tonnes and grade for mineralization and is the 'template' for creation of the mining schedules and cash-flows.

Geometallurgical variables can be measured and be a constituent of the sampling database. The various approaches to estimation can be applied to create a spatial representation of the geometallurgical properties of a deposit in a similar approach to generating a resource model. Cautious selection of the suitable spatial modelling techniques is vital since some geometallurgical attributes behave quite differently in a spatial framework, compared to grade variables. Some variables or geometallurgical attributes may be non-additive thus requiring non-linear estimation methods. Other variables are rather interpolated from simulation rather than estimate. Resultantly, the obtained 3D model comprises additional variables related to the realized value of each block, over and above the traditional grade and tonnage variables. (Dunham *et al*, 2007).
The spatial geometallurgical models may be utilized in a variety of ways that include improved:

- mine and process design, resulting in more efficient capital allocation,
- mining project valuation,
- sequencing and scheduling,
- forecasting of revenues and costs,
- process optimisation, and
- Tactical improvements to mine planning (block selection) and more informed blending strategies.

The above examples illustrate that geometallurgical modelling or characterization of the resource with value determining attributes of the mineralization will have real positive influence on the business.

2.3.3 Geometallurgical attributes

Geometallurgical relevance to operational planning and plant design is based on recognizing attributes that contribute to value realization of the resource. These attributes comprise 'material characterization' in addition to traditional attributes such as the grade. The following are examples of geometallurgical attributes:

- concentration of deleterious elements (both those resulting in penalties for sold concentrates as well as those impacting on recovery or other processing responses),
- hardness and grindability,
- mineral liberation,
- metallurgical recovery,
- reagent consumption, and
- Smelting characteristics. (Dunham *et al*, 2007).

An appropriate sampling strategy is fundamental to the modelling and estimation process for both grade models and even the more complicated multivariate geometallurgical models. Sufficient samples of an appropriate size or geostatistical support ('volume variance effect') are necessary from the main lithological or mineralogical domains in the orebody in order to model both the average and variability of the geometallurgical variables to a known degree of uncertainty. The geometallurgical sampling program also needs to take into consideration the scale-up effects, that is, the relationship between bench-scale testing and plant-scale performance. (Coward *et al*, 2009).

- Another important fact to consider when handling geometallurgical variables is whether they can be linearly averaged, this is a property known as additivity. By ensuring that the attribute at hand is additive, we avoid bias during the estimation process. This applies when techniques such as arithmetic averaging, weighted-averaging or other linear combinations such as kriging are employed. An example of a non-additive variable is recovery, which has a polynomial relationship to grade. A successful way to implementing successful geometallurgical modelling is to consider additive variables that enable the prediction of non-additive variables. Coward *et al* (2009), proposed a primary response framework to classify the geometallurgical variables.

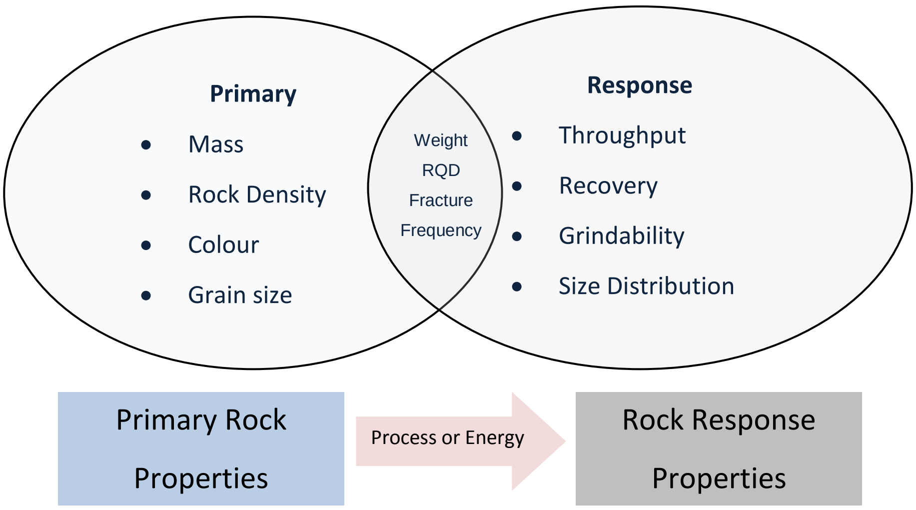


Figure 7: The primary response framework (after Coward et al, 2009)

Primary variables are rock attributes that can be measured directly for example grades, whereas *response variables* are those that describe the rock's responses to processes, for example throughput, or recovery.

Response variables are generally complex and are characteristically non-linear. Though not all primary variables are additive, it is much easier to assess their underlying properties. The possible bias resulting from linear averaging of non-additive variables is illustrated through the Jensen's Inequality shown in Figure 8 below.

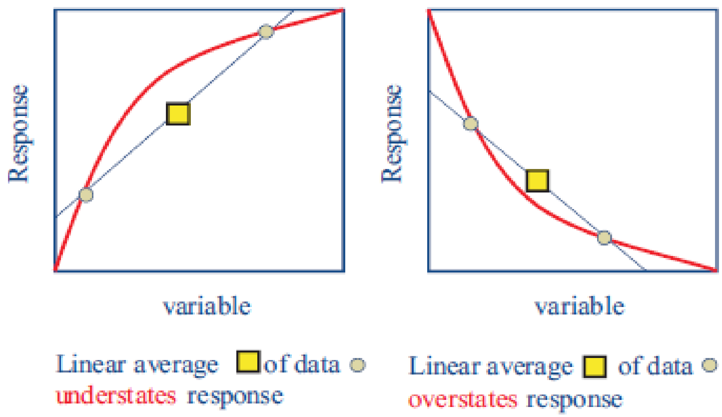


Figure 8: Jensen's Inequality (after Coward et al, 2009)

Figure 8 illustrates that when the two variables have a non-linear relationship, a simple linear average will overstate or understate the actual value. Whether the value is overstated or understated depends on the manner of the non-linear relationship. For more complex relationships, prediction of the errors would also be more complex. (Coward et al, 2009).

2.3.4 Selection of geometallurgical attributes

Selection of geometallurgical variables is much more complex for *greenfields* projects than *brownfields*. This is because a variety of processing options will still be under considerations and not yet clarified at the start of a project. By considering other projects, inferences about ore/ process impacts can be made based on the behaviour of certain deposits in similar processing plants. A suit of samples will have to be collected so that tests are performed corresponding to processing options available.

The selection and identification of geometallurgical variables in *brownfields* projects is aided by the existing processing plant data. The data can assess the effectiveness of the models themselves, as well as the effectiveness of the sampling. By establishing the correlation between specific rock properties and specific periods of metallurgical performance, appropriate geometallurgical variables can be selected. For this to be possible, however, both well managed, reliable mine depletions data and plant performance data should exist. This implies that there is need for a broader database that captures a wider array of variables than the conventional tonnes and grade. Different circuit types have some specific material characteristics requirements, some examples are illustrated in Table 2 below (after David, 2007);

Table 2: Circuit types and critical properties

Circuit type	Critical properties
SAG mill circuit	• Competence • Grindability
Stage crush and mill circuit	• Crusher work index (Bond work index) • Grindability
Iron ore (haematite)	• Lump/fine ratio • Impurity levels (S, P, etc.)
Copper sulphide ore	• Copper mineralogy • Alteration • Impurity levels (F, As, etc.)
Gold ore	• Gravity recoverable gold • Cyanide recoverable gold • Refractory gold • Copper grade • Preg-robbing properties
Nickel laterite	• Beneficiation characteristics • Acid consumption • Slurry viscosity
Water recovery and thickening	• Clay minerals content, presence of smectite • Flocculant consumption
Diamond recovery, kimberlitic ores	• Heavy minerals content • Competence, preferential liberation
Coal	• Impurity levels (S, P, etc.) • Lump/ Pea/ Duff ratio
Roasting processes	• Sulphur ratios
Bio-leaching processing	• Sulphur ratios

The samples tested for the above material properties should be of adequate size to represent the life of mine feed. Unsound assumptions on what constitutes a representative sample can be misleading on deciding on processing and plant design options. (David, 2007).

2.3.5 Process design and geometallurgy

Process design, from the owner's or business perspective, aims to achieve a number of objectives and these include:

- Minimizing capital expenditure.
- Minimizing the process operating cost.
- Minimizing the ramp-up or schedule time to production.
- Design a robust and effective process plant.
- The ultimate objective will be to have an as-short-as-possible payback period, and generate shareholder returns as soon as possible.
- Make a profit, (David, 2007).

An analysis of operational and design data sets (David, 2007), from both greenfields and brownfields sites, has revealed that the influence of the geological data on process design is largely limited (or missing), and often misapplied. Traditionally, no deliberate effort is made to connect the geological variables to a particular process flow-sheet when developing a resource block model. The block model aims to show information related to the resources and reserves. The geological data is generated by measurement of inherent rock properties such as assays, mineral composition and compressive strength, and is not dependent on the process method options on offer.

Most methods to conducting a metallurgical test program for process plant design follow a defined approach; this is usually referred to to Ore Dressing Studies: (David, 2007).

- Determine of the major facies or ore types, usually by geological definition such as lithology.
- Collect samples that are representative of the facies for metallurgical test-work. For metallurgical samples, large diameter drilling is preferred.
- Collect samples of a larger size, typically 20 – 100tonnes for pilot plant trials
- To conduct a comprehensive pilot test program, tonnage size samples are sourced for the major ore types.
- Conduct material characterization tests to outline the rock properties of the major geological ore types.
- If possible, perform tests on ore blends representative of the life of mine feed at various time periods. (David, 2007).

Critical design factors (such as crusher design parameters, mill specific energies, dense medium operating densities, dense medium yields, flotation recoveries and leach recoveries), are then derived from the analysis of data from the test program. The determined factors are used to set process design criteria and subsequent process equipment sizing. There are some significant flaws embedded in this approach, which can adversely impact on the project. In most instances, the ore types that are selected on the basis of geological classifications bear no metallurgical significance. Consequently, there is inadequate metallurgical definition between ore types, mainly due to high variability and overlapping of the metallurgical properties between the lithofacies. Some test results based on averaging non-additive parameters and the related domains often mislead designers and ultimately the metallurgists and operators who will operate the plant. (David, 2007).

Contrary to a conventional metallurgical test program or ore dressing study, a geometallurgical program is innately related to intended process flow-sheet and aims to provide a direct input to the ultimate selection of a process plant specific to the ore body.

A geometallurgical program needs to be preceded by the metallurgical test program designed to assist selection of the process flow-sheet. (David, 2007).

To develop a geometallurgical test program for process design, the first step is to review the geological domains for suitability as geometallurgical domains. Assignment of geometallurgical domains requires an understanding of the important process characteristics. An example of process characteristics is shown in table 2 above. Some of these characteristics are measured directly within the geological analysis, whilst a number of these can have the potential to be inferred by using one or more of the pieces of information within the geological database. (David, 2007).

The next step is to select samples representative of the orebody that characterize a reasonable spatial distribution of the appropriate lithological domains. The rock types that are likely to be treated early in the life of mine must be given priority to testing. Metallurgical testing is conducted to determine the design critical parameters. The critical properties are usually those that determine throughput, final product rates and saleability of product, and any other essential properties that relate to design criteria. Consideration of some environmental properties can also be important in some situations. (David, 2007).

2.3.6 Establishing a geometallurgical program

A geometallurgical program is a structured endeavour that aims to generate a reliable, practical and beneficial model of an ore deposit and mineral processing plant for resource exploitation. The program generally goes through the following steps, for application in either greenfields or brownfields projects (Lamberg, 2010):

- Compilation of geological information through drilling, drill core logging, chemical analyses, measurements, data analyses and other data collection processes.
- Utilize geological data to identify ideal locations for samples in an ore sampling program for metallurgical testing.

- Ore variability testing, laboratory testing of metallurgical samples to derive process model parameters.
- Assessment of the metallurgical validity of the geological ore-type definitions or geological domains and, where necessary, developing geometallurgical domains.
- Assess the primary geometallurgical variables and develop appropriate mathematical relationships across the geological database. Consideration of additivity or non-additivity of the variables is paramount for the estimation of important metallurgical parameters.
- Utilize metallurgical parameters defined above to developing a unit operations metallurgical model of the process.
- Plant simulation
- Calibration of the models via benchmarking for similar operations. (Lamberg, 2010).

There is growing focus into the field of geometallurgy and consequently research initiatives. In 2005, there was a realisation of a cross-disciplinary challenge to the mining industry. The global minerals industry saw need for a large-scale, integrated geometallurgical research, resulting in the commencement of a major initiative, the AMIRA International P843 'GeM^{III}' Project (Geometallurgical Mapping and Mine Modelling). The initiative is a collaboration of three major Australian research groups; CODES the ARC Centre of Excellence in Ore Deposits at the University of Tasmania, a global-leader in research related to economic geology; the Julius Kruttschnitt Mineral Research Centre (JKMRC) at the University of Queensland, a world-leader in mining and mineral processing research; and the WH Bryan Mining Geology Research Centre (BRC) at the University of Queensland, which aims to spear-head research in mining geostatistics, operations research and optimisation in mine design and planning. The AMIRA International P843 'GeM^{III}' Project is sponsored by twenty one companies that include Anglo Gold Ashanti, Anglo Platinum, Barrick, BHP Billiton, Codelco, Datamine, Golder Associates, GEOTEK, ioGlobal, Metso Minerals, Newcrest, Newmont, Oz Minerals, Pe_oles, Rio Tinto, Teck Cominco, Vale, Vale Inco and Xstrata Copper. The project is financed to an excess of AU$8 million, with a team of over 30 researchers and support staff.

2.3.7 Uncertainty, Integrated planning and geometallurgy

A widely accepted view is that the mining business is inherently risky. The delivery of mining projects has been unsatisfactory, with major problems being lower than expected ramp-up to production, geotechnical challenges, metallurgical under-performance and unbudgeted capital or operating costs. Most of the challenges can be attributed to decision making based on the interpretation of the geoscientific information and estimates. (Berry *et al*, 2006).

Berry *et al* (2006), highlighted results of a 2003 survey of 105 projects that identified common experiences and problems associated with the non-delivery mining projects. The results of the survey and categories of problems identified are summarized below:

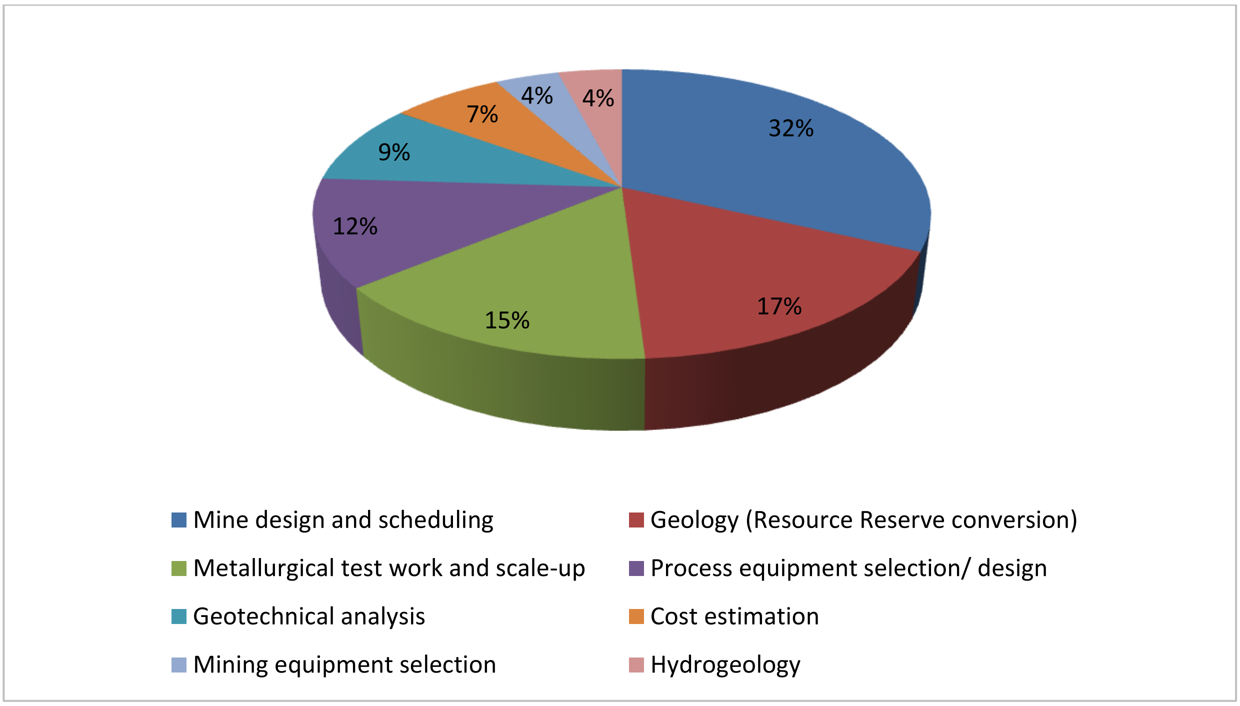


Figure 9: Percentage contribution of the cause of project failure (after Berry et al, 2006)

The study illustrates that the major cause of project failure is mine design and scheduling, with a contribution of 32%. Considering two of the causes in the study, metallurgical test-work and design, and combining them to represent metallurgical risk, the study shows that metallurgical risk is the second largest cause of project, responsible for 27% of the failures. The third contributor to project failure was resource/ reserve estimation, estimated at 17%. Though mine design and scheduling is rated as the biggest contributor, some elements within the geological estimation can cause design and scheduling challenges, implying that there can be an overlap in all the subclasses determined in the study.

Danilkewich, Mann and Wahl (2002) made reference to an independent earlier study of 1997 that demonstrated that resource and reserve estimation was biggest contributor to failure of projects, and that the second is related to metallurgical risk. Though Danilkewich *et al* (2009) did not state the relative percentages of the various classes of causes to project failure; there is some concurrence with the later study by Berry *et al* published in 2006.

According to the two studies above, inadequacies in geology, metallurgy, and mine planning functions are responsible for an estimated 76% of failures in mining projects. This strongly suggests insufficient determination of technical risks both at commencement and further into the future of the projects. The concept of geometallurgy aims to bridge the gap between these three functions (geology, metallurgy and mine planning), thus alleviating what can be referred to as geometallurgical risk.

2.3.7.1 Strategic geometallurgy

Integrated planning at the strategic level, aims to deliver economically optimal decisions concerning long term issues such as acquisitions, mining methods, extraction methods and processing rates. Integrated planning considers all technical aspects that affect depletion of the resource, i.e. geotechnical considerations, geological considerations as well as mineral processing and metallurgy. Strategic planning aims to identify more economic and practical alternatives to resource depletion.

Strategic geometallurgy influences the long term decisions. A geometallurgical approach during the strategic planning phase will enable identification of technical risk aspects and reducing the level of uncertainty. Spatial modelling of the critical physical attributes that affect process response provides an upgraded basis for operational design and mine planning.

Strategic geometallurgy influences the feasibility study outcomes and process design by reducing technical risk of greenfields operations, or informing expansion options for brownfields.

An example of strategic geometallurgy application, Canahuire Epithermal Au-Cu-Ag Deposit, Southern Peru:

An illustration of the application of a geometallurgical program at strategic level is the project development study for the Canahuire gold-copper-silver deposit in Southern Peru. A geometallurgical program has been initiated within the feasibility study phase that is to be completed in 2012. (Baumgartner, Brittan, Dusci, Gressier, Mayta, Poos, Trueman, 2011)

Four geological or mineralization domains have been developed for this deposit and these are:

i. Domain BX Au-Cu: Breccias (BXPS, BXM, and partly BXP) with Au and Cu mineralisation.
ii. Domain CAL Au: Limestones with Au mineralisation and small amounts of Cu.
iii. Domain SED Au: Non-calcareous sandstones with Au and small amounts of Cu (CSC and SMC, also partly SSQ).
iv. Domain BXP Ag: Diatreme breccia (BXP) with Ag mineralisation (sub-economic). (Baumgartner *et al,* 2011)

In line with the methodology of progressing a geometallurgical program, the four geological domains provided the basis for developing geometallurgical domains. Sampling was done within the domains in order to appreciate the variability within them, and for further refinement into geometallurgical domains. Based on the proportions of the domains within the deposit, the number of samples collected from each domain varied accordingly: Domain CAL Au: 25 samples, Domain BX Au-Cu: 19 samples, Domain SED Au: 6 samples and Domain BXP Ag: 1 sample. (Baumgartner *et al,* 2011).

A characterization and testing program has been developed consisting of a suit of metallurgical tests.

- Metallurgical variability test-work – consists of flotation, Carbon-In-Leach and conventional cyanidation tests. Optimisation work was also conducted to refine the process flow-sheet and improve gold recoveries. These tests revealed the high variability of the recoveries within the four geological domains.
- Mineralogical analysis – Gold extraction metallurgy is predominantly driven by mineralogical factors, and thus a mineralogical analysis is a critical geometallurgical aspect. The minerals in each in each sample are quantified by optical mineralogy, increasing knowledge of mineral distribution in the deposit. A mineral liberation analysis (MBGLA) was also conducted to ascertain metallurgical variability and quantitative mineralogy, and a follow-up x-ray diffraction (XRD) has been done to confirm the mineral species identified.
- Gold deportment – test work included analysis of mineralogical factors such as gold particle size, liberation, oxygen consumers and gold's association with other minerals.
- Comminution – Tests included the bond ball mill work index, angle of repose, abrasion bond test and SAG comminution tests
- Waste rock – Potential for acid rock drainage was detected due to presence of pyrites and other sulphides. (Baumgartner et al, 2011).

The result of the above work will be geometallurgical domains defined by both the metallurgy and geology, rather that the geology and mineralogy. After validation which is currently on-going, mathematical models will be developed across the database to reaffirm the geometallurgical domains and predict metallurgical responses such as recovery. (Baumgartner *et al,* 2011).

The resource model will contain not only the geological data, but metallurgical parameters modelled along with the economic elements. From the sampled areas of the deposit, variables of interest are estimated across the deposit by utilizing geostatistical characteristics of these variables. (Baumgartner *et al,* 2011).

The deliberate effort to build a geometallurgical model at Canahuire is aimed at predicting ore variability; waste rock behaviour and acid mine drainage as well as optimizing recoveries of the value elements specifically gold. The geometallurgical information on the resource will influence the decisions and strategies on plant design, operational planning and scheduling. The abundance of information will reduce geometallurgical risk that usually arises as a result of averaging data from limited populations of metallurgical test work. The challenges facing the geometallurgical program at Canahuire are the complexities associated with a greenfields project where there is no pre-existing production information to validate test results. Another challenge is the constraint of project timelines and budgeting that limits the amount of tests that can be performed. (Baumgartner *et al,* 2011).

An example of geometallurgical risk, Voorspoed ramp-up challenges:

An illustration of the importance of strategic geometallurgy is the ramp-up challenges that occurred at Voorspoed Mine. Located in the Orange Free State in South Africa, the mine has been under the ownership of De Beers since 1965. Since then, sampling campaigns that were done did not encourage re-opening the mine that had been closed earlier due to insufficient liberation caused by hard ore. In 2004, a rather lean operating model was proposed for the low-grade inferred resource to produce a marginal business case. Construction began in 2006, with the mine officially opening in November 2008. (Lindsell-Stewart, 2009).

On commissioning, only weathered kimberlite from the low-grade northern portion of the pipe was available for processing. This clay-rich kimberlite brought along treatability challenges. The compacted clay on the crushed ore stockpile blocked the opening of the clam shells feeding the conveyor below. Feed chutes blockages were encountered as the wet clay could not flow, sticking on chute walls causing build-ups. The clay contained ultra-fines that could not settle, leading to muddy overflows at thickeners. (Lindsell-Stewart, 2009).

Another challenge was ore quality control and distinguishing material types. The weathered, basalt rich kimberlite was difficult to distinguish from waste rock. The geological model that had been developed over the years could not show the contacts. Data collected in the first months of production was used to update the geological model to develop a functional model that could be used in mine planning. All these problems led to delays and a reduction in feed rates, the ramp-up was four times longer and only a third of ramp-up targets were achieved. A substantial amount of further work was identified, leading astronomical increases in the capital expenditure. (Lindsell-Stewart, 2009).

All these challenges with ore treatability showed the importance of strategic geometallurgy. The unreliability and poor predictability of the previous geological model led to realisation of unexpected dilution and frequent contact changes. Most of the challenges were mitigated by management of dilution and communication of contact deviations. The approach to develop a new model did consume a lot of time, but this led to the development of an appropriate model that could be used in production, and also provided a crucial knowledge base of the resource. (Lindsell-Stewart, 2009).

Increased knowledge of the ore body provides a more consistent and representative model that can be used as a basis of flow-sheet design. Major constraints and bottlenecks can be identified, and low-cost and high value opportunities can become apparent. This can result in better asset utilization, product recovery, decision making in terms of capital expenditure, and will lead to improved economics of the project.

2.3.7.2 Tactical geometallurgy

Tactical planning deals with the short term, and is concerned with implementation of strategic plans. Any form of risk alleviation is linked to an already defined design; it does not address strategic concerns of whether the design is the best option for the resource. The tactical environment is about achieving strategic plans, even if a suboptimal option has been implemented. (Kear, 2006).

Tactical geometallurgy is mainly applicable at the operational phase. The key benefit of a tactical geometallurgical approach is the improved vital communications between geologists, mine planners and metallurgists. By utilizing geometallurgical and resources models as well as production databases, the disciplines will come to comprehend the capabilities of the deposit and optimize the value extraction by scheduling the most appropriate combination and sequence of ore types. The uncertainty of grade control and the risk associated with ore treatability is reduced through appropriate blending options. (Dunham *et al*, 2007).

An example of tactical geometallurgy application, Kemi Chromite and Ferrochrome mine:

The Kemi Chromite and Ferrochrome Mine is located on the coast of the Gulf of Bothnia, in northern Finland, and was established in 1968. The mine was developed after comprehensive geological, ore dressing and metallurgical investigations confirmed the profitability of the deposit, initially discovered in 1959. The mine is operated by

Outokumpu, and in the 1990s a major expansion was planned leading to an underground development to replace the open pit. (Lamberg, 2010).

For the past 17 years, Kemi chromite mine has continuously collected mineralogical information that has systematically been used in production planning. The major challenge with the deposit is the high variability of the grain size of chromite in the deposit which adversely affects treatability. In order to meet production plans, there is routine and systematic measurements of the chromite grain from the drill cores. Grain size distribution information is obtained from polished sections of the cores from optical microscopy and image analysis. The data obtained is fed into the ore block model. The size distribution data is used to evaluate the treatment options of the ore types, determining suitability of fine concentration methods, and to forecast the expected recovery. The recovery estimation is based on the correlation of the grain size of chromite after grinding, to the grain size of the original chromite in the ore. Another consideration on the estimation is the recovery losses that occurs in the fine end (<80 microns) during the gravity separation process. (Lamberg, 2010).

Kemi Mine is a bench mark operation in terms of technology and data acquisition. Superior productivity levels have been achieved mainly due to;

- Cross functional information and data acquisition systems
- Cross disciplinary communication and information systems network for real-time monitoring and control
- Computerized information management, mine planning, and control and maintenance systems. (Lamberg, 2010).

In the tactical environment, geometallurgy can be considered to be an extension of the mine-to-mill practice. The tactical geometallurgical approach optimizes the business plan by ensuring operational delivery per shift or day at the lowest cost. Ensuring delivery of short term plans will ensure that the long term strategic plans are met, thus optimizing project economics. (Dunham *et al*, 2007).

2.3.8 Flexibility and geometallurgy

Risk on one hand can be considered a product of uncertainty, and flexibility on the other hand can be considered to be the inverse of uncertainty. Minnitt *et al* (2007), described flexibility as the ability to accommodate, and take tactical advantage of any changes in conditions.

Strategic geometallurgy can influence design decisions, and thus allow in-built flexibility to deal with risk that may arise as a result of unveiled uncertainty. A fully attributed geometallurgical model does a better job in identifying risks and opportunities during pit optimisation and operational design. The wealth of information that a geometallurgical model provides presents a platform for informed decision making.

At the tactical level, significant value-add can be made during the scheduling phase of mine planning. Various alternatives can be considered by assessing the costs of a particular flexibility option and the impact it will have in terms of sustainable value addition. Tactical geometallurgy enhances the mine-to-mill process, through increased communication between geology, metallurgy and mine planning. Reactive, uninformed short term planning can destroy value and flexibility, due to the desire to satisfy short term goals. Such situations, however, are avoidable in the presence of geometallurgical models as potential issues that can introduce constraints in future can be identified. (Dunham *et al*, 2007).

An illustration geometallurgical risk and importance of flexibility, Cawse Nickel Operation:

The Cawse nickel project was designed as a new generation pressure acid leach process utilizing solvent extraction and electro-winning for the production of LME grade nickel metal and a 40 percent cobalt sulphide concentrate. The operation located 50km from Kalgoorlie in Western Australia, treated nickel and cobalt oxide ores from a lateritic ultramafic profile. Owned and operated by Centaur Mining and Exploration (CME), Cawse Nickel Operations (CNO) had a successful technical 'due-diligence' and 'bankable feasibility study', confirming a large resource base that could be exploited by a relatively new technology at comparatively lower unit costs. (Bywater, 2003).

A major factor in the viability of the project was ore upgrading through a beneficiation circuit, considered to be a cost effective way to increase grade. The process involved crushing, then drum scrubbing to remove low grade >0.5mm material, containing barren silica and manganese. The ore had a head grade of 1.3% nickel, and was expected to upgrade by 47% to a grade 1.9% nickel with subsequent treatment in autoclaves, using sulphuric acid at 4500kPa and 250^{o}C. (Bywater, 2003).

The plant was commissioned in October 1998 reaching design throughput in May 2000, a major milestone for the project. A delayed start-up, however, had not helped the project in the early years which had resulted in a plant upgrade in 1999. The indications were that the upgrade achieved on the head grade was only 33%, with a final grade of 1.7% nickel. (Bywater, 2003).

On resource reserve estimation, there was an overestimation of about 20% due to lower than expected bulk density for limonite and cobalt ores as a result of inaccurate determination of moisture content. To compensate for the tonnage, the production rate was increased and this implied further drilling for resource definition. (Bywater, 2003).

Other parameters, as obtained from the feasibility study are summarized in table below, and compared to production results:

Table 3: Cawse Nickel Project feasibility study versus production results

Parameters	Feasibility (5yr)	Production (3yr)	%Variance
Capital costs $A (mil)	306.8	376.2	23
Dry bulk density	2.1	1.7	-20
Ore feed upgrade %	47	33	-30
Ni product grade	1.9	1.7	-10
Ni output (tpa)	8000	4551	-43
Co output (tpa)	900	667	-26
Ni recovery (%)	92	88	-4
Co recovery (%)	92	90	-2
Nickel price $US/lb	3.25	2.99	-8
Cobalt price $US/lb	10 -18	16	14
$A:$US exchange rate	0.65	0.6	-8
Cash operating costs $US/lb	1.34	1.97	47

A variety of factors can be attributed to reduction in cash-flow and operating viability. These included increased capital expenditure, slower ramp-up period, reduced reserves, reduced recoveries, increased maintenance and consumables costs, increased interest repayments due to fluctuating exchange rate as well as nickel and foreign exchange hedging. (Bywater, 2003).

The foremost cause of the failure of the project was that the feasibility study was dealing with new technology leading to underestimation of critical parameters, resulting in 43% less metal output. There was no flexibility built into the option, with all contingency being used up and capital expenditure being 23% above budget. (Bywater, 2003).

The capital budgeting option was suboptimal, at 100% debt. This led to high interest repayments, placing significant financial burden on the first few years of operation and during the ramp-up period. The project had no financial flexibility and this was exacerbated by the low metal output and the interest demands. (Bywater, 2003).

The failure to generate adequate cash-flow and service debt repayments ultimately led to liquidation of the asset, a debt of A$784 million remained after the sale of Centaur Mining and Exploration's gold and nickel divisions. The mine is currently operating under new ownership, a modified plant and further experiences in nickel laterite projects has led to the revitalization of the project. (Bywater, 2003).

Geometallurgical techniques can improve the outcome of a feasibility study and reduces the risk of prolonged ramp-up and failure to meet production targets. Knowledge of the ore characteristics can inform the design, and can improve scheduling and planning outcomes. Geometallurgy provides a wealth of information that can be utilized in responding to unwanted events, thus reducing the duration of such events and building in flexibility. (Dunham *et al*, 2007).

2.4 The influence of geometallurgy on risk, uncertainty and flexibility

Mining investments face many sources of uncertainty that increase the risk of financial returns. As illustrated in section 2.1.1, risk is a result of uncertainty; inadequate information (uncertainty) increases the "probability" element in the risk model (equation (1) on page 6 and 7). Uncertainty increases the risk in the rate, cost and efficiency of mining and treatment of the material extracted from the ore body (ore and waste). A key endogenous risk factor is ore body uncertainty as a direct result of inadequate information (arising from sparse sampling) and estimation processes. The processes of resource estimation, such as linear kriging, produce overly smoothed geological models that may offer the best linear unbiased estimate, but may not predict the variability that will be encountered when mining and processing the ore body.

Geometallurgy is an emerging integrated approach that aims to mitigate the risk on financial returns, by using robust spatial models of treatment characteristics in mine and plant design, mine planning and operation. This approach increases knowledge and information on the ore body, reducing uncertainty and the subsequent risk (geometallurgical risk). This approach, however, is often compromised by:

- the lack of spatial models containing geometallurgical variables that can be used to reliably forecast operational effectiveness; and
- Poor integration between the disciplines of mining, mine planning, metallurgy and geology.

The geometallurgical approach leads to understanding of the ore body and variability aspects that have material impact on extraction and processing of the ore. If this is not understood, mine planning and plant performance predictions will be biased and can lead to underestimation of the value of incorporating flexibility options in process design to deal with the impact or "consequence" element of the risk (equation (1) on page 6 and 7).

The relationship between the concepts of geometallurgy, risk and uncertainty, and operating flexibility can be summarised as follows in Table 4;

Table 4: The influence of geometallurgy on risk and uncertainty, and operating flexibility

Risk Elements	Effect of a geometallurgical program on risk and operating flexibility
Probability	More information on the resource and improved knowledge of the ore body, reduced uncertainty. This leads to better decision making, optimum designs and planning for flexibility. The "probability" element of risk is mainly influenced at the strategic phase of planning (strategic geometallurgy).
Consequence	A platform for functional collaboration of mining, metallurgy, planning and geology teams. This can improve decision making, quicker responses (flexibility) to downside or upside risk and optimal tactical planning. The "consequence" element of risk is mainly influenced during tactical planning (tactical geometallurgy).

A geometallurgical program can reduce uncertainty (thus the probability of occurrence), and can offer the enterprise flexibility options to deal with the impact of downside risk, and ability to capitalise on upside risk.

This research report aims to investigate and prove the following:

> The Orapa 2 plant is exposed to geometallurgical risk due to geological uncertainty. Some metallurgical design decisions have exacerbated the impact of the risk and limited the operation’s flexibility to react.
>
> Poor integration of functions, i.e. mining, geology, mine planning and metallurgy can increase geometallurgical risk worsen its impact. To effectively address geometallurgical risk in mining projects, the geometallurgical approach has to be embedded in the business structure.

The Orapa mine and operating methods are described in the following chapter.

3 ORAPA MINE OPERATIONAL DESCRIPTION

Orapa mine is the world's largest diamond mine, covering 1.18 square kilometres at ground level. Located in Botswana about 240 kilometres (150 mi) west of the city of Francistown, Orapa ("resting place for lions") is owned by Debswana, an enterprise culminating from the 50:50 venture between the De Beers Company and the Botswana government.

Orapa Mine is exploiting the Orapa A/K1 deposit, the largest within North-Central Botswana's Orapa Kimberlite Cluster. The cluster consist of approximately 80 known kimberlite pipes, some of which have either been previously mined (e.g. A/K2), are currently being mined (e.g. A/K1, B/K9, D/K1) or are being evaluated for economic potential (e.g. A/K20, B/K11).

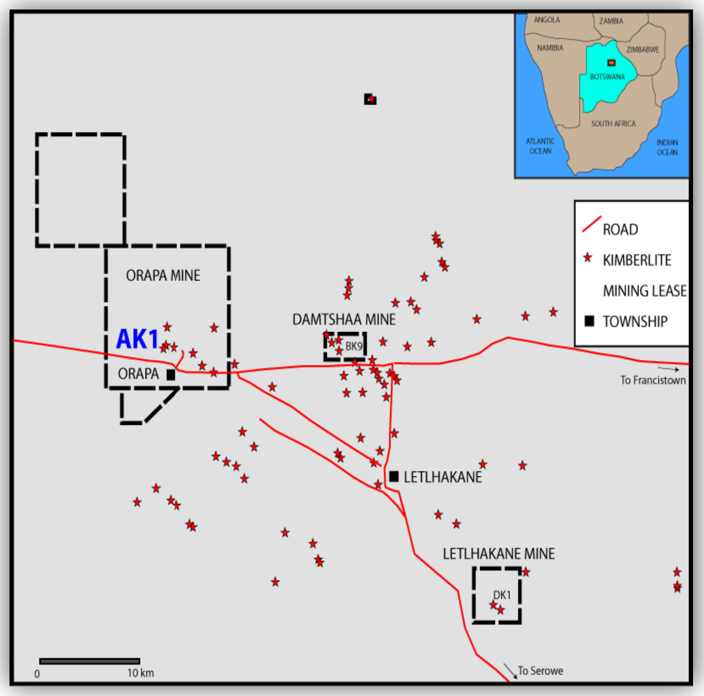


Figure 10: The location of the A/K1 deposit, Botswana

The Orapa A/K1 deposit has been exploited since its discovery in 1967, and the active operations commenced in 1971.

3.1 The Orapa A/K1 deposit description

Since discovery in 1967, the Orapa A/K1 deposit has gone through numerous evaluation programs, and because of this history, it provides an ideal case study to evaluate how the uncertainty in geological parameters varies as more information becomes available, proving how decisions made with limited geological information impact on the long term profitability of the operation. The deposit is unique in the sense that it is one of the very few deposits that have more than 10 rock types.

3.1.1 Historical Studies

The A/K1 resource is a north-south elongate pipe, resembling a peanut-shell shape from the plan view, wider in the south and narrower in the north. At depth the pipe separates into two distinct pipes with the classic carrot shaped morphology typical of kimberlite pipes. (Tait, 2009).

A considerable amount of geological work has been done on the A/K1 deposit, most of it documented in internal company reports.

Table 5: Geological studies done at A/K1 deposit

Period	Activity	Depth	Result
1967	Discovery		
1967-68	Pitting	~37m	
1967-68	Delineation core drilling	100m	
1975-80	Pitting	30m	Proven reserve to 30m
1982-93	LDD Phase 1	200m	Proven reserve to 200m
1987-93	LDD Phase 2	250m	Probable reserve to 250m
1994-97	Deep delineation core drilling	660m	Inferred resource to 660m

As the upper part of the A/K1 deposit is dominated by crater in-fill type deposits, much of this early work focused on the internal sedimentology of this part of the deposit, however, key to the bulk of these early geological studies was identification of the strong relationship between lithofacies type and diamond grade. (Tait, 2009).

The first geological model for A/K1 was developed in 1991, showing that the ore body consisted of two lobes (north and south), coalescing at near surface at the upper crater. It was noted that there is a clear distinction between the north and south lobes, as well as the upper crater material. A special project was commissioned in 1995 to review and refine the 1991 model of based on updated after extensive reviews of the geology. The result was the first digital 3D geological model of A/K1 resource, with associated block model, referred to as the 1996/97 A/K1 geological model (or Matthew Field 1996/97 Geological model). The geological model was adopted by Orapa Mine, and was in use for mine planning until 2008, the associated block model is still being used for mine planning. The Matthew Field work was influential in developing the kimberlite nomenclature for the various lithofacies in the A/K1 deposit. The model was later updated in 2007, but there was no redefinition of the internal geology or conceptual emplacement of the model. (Tait, 2009).

Figure 11 shows the updated models as of 2007 and 2009, showing the clearly distinct north and south lobes, and the upper crater.

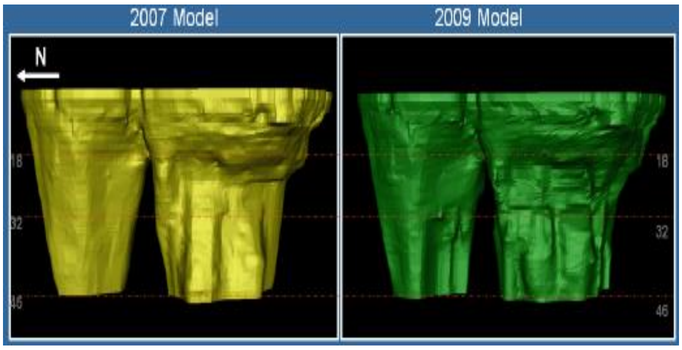


Figure 11: A/K1 north lobe and south lobe

Another special project was commissioned in 2006 to address short comings in the Mathew Field model, update A/K1 geological model with the latest information and create a block model. The results of this project were as the second 3D geological model of A/K1 resource, referred to as the 2007 A/K1 Geological model (or Darrel Farrow and Michelle Maccelari Model). The 3D geological model is still currently being used for mine planning but the associated block model is not yet officially accepted for use in mine planning. (Tait, 2009).

3.1.2 More Recent and Current Work – The Orapa Resource Evaluation Program

The Orapa Resource Evaluation Program (OREP) was initiated in 2005 to address inadequate understanding and sampling of the Orapa A/K1 resource at depths below 265mbgl. Though some previous programs did reach depths of 600mbgl, the previous grade sampling campaigns have not extended beneath 265mbgl. The OREP programme was instigated with the aim of increasing the level of confidence of the A/K1 resource from inferred to indicated resource (between 265mbgl to 600mbgl) in terms of geology, volume, density, grade and revenue. The original delivery plan for OREP envisaged two phases of the project, the first phase focusing on geology, volume and density; and the second focusing on grade and revenue. (Tait, 2009).

The first phase of the OREP program (OREP I), was intended to define the volume and internal geology of A/K1 to a depth of ~600mbgl. This was completed in 2009. The second phase (OREP II) commenced in 2010 and is expected to complete in 2015. OREP II is mainly a grade and revenue sampling program, intended to supplement the geological data from OREP 1. Further geological, density, geotechnical and geometallurgical information is also expected to be collected from OREP II. Completion of the full OREP program (a third phase is planned for 2017-2019) will deliver a local block estimate of the diamond content of the A/K1 resource to a depth of 685mbgl. (Tait, 2009).

The OREP I project contributed in the updating of data used in construction of previous models to develop the 2009 Geological model. The model has maintained the general shape of the AK1 pipe as depicted in the first conceptual model and subsequent models.

The main concern from the ‘new’ information, however, is the realization of major changes in AK1 kimberlite lithofacies, noted in both the North pipe and the South pipe. The significance of this realization is that the process plants were designed for the previously observed lithofacies, and the design does not fully address treatability requirements for the new rock types.

3.1.3 The geology - Key lithofacies within A/K1

The 1991 model noted that the north and south lobes were quite distinct. Later information (Tait, 2009) showed that the two lobes represented individual volcanic events that formed the North and South Pipes. The upper crater was a result of sedimentary in filling deposits or emplacements that followed initial pipe formation.

3.1.3.1 A/K1 Upper Sedimentary Deposits

The upper part of A/K1 consists of sedimentary deposits filling the crater. Much of this material has been depleted and information for these lithofacies is no longer available. According to the 2009 geological model, 5 major sedimentary units in the upper crater existed. The A/K1 South is the latter of the two pipes (Tait, 2009), and the majority of these emplacements lie within the crater created by eruption of the south pipe. In the geological model, these are defined by the rock codes:

- A3T -Talus deposits
- BB -Basalt Breccia
- EGSST -Epiclastic Grits, Shales and Sandstones
- EBB -Epiclastic Boulder Beds
- DFBB -Debris Flow Basalt Breccias

3.1.3.2 A/K1 South Pipe

The following are the rock codes existing in the A/K1 South Pipe, more detail on the rock types is offered in table below:

- SVK Southern Volcaniclastic Kimberlite (Major variations in internal structural composition and texture, thus SVK is sub-divided into further stratigraphic domains and these are also highlighted separately.)
- SVK_U Southern Volcaniclastic Kimberlite – Upper
- SVK_M Southern Volcaniclastic Kimberlite – Middle
- SVK_L Southern Volcaniclastic Kimberlite – Lower
- SDVK Southern Dark Volcaniclastic Kimberlite
- BB Basalt Breccias
- A3T Talus flow deposits
- EGGST Epiclastic Grits and Shales
- EBB Epiclastic Boulder Beds
- DFBB Debris Flow Basalt Breccia

3.1.3.3 A/K1 North Pipe

The 2009 model shows three major units in the A/K1 North Pipe, specifically NPK, MVK2, NPK_GG. This signifies a significant departure from the previous model that showed a single rock unit (NPK) projected down the depth of the pipe. A summary of the rock codes is shown below, and further explained in table below:

- MVK2 Massive Volcaniclastic Kimberlite
- NPK Northern Pyroclastic Kimberlite
- NPK_GG Northern Pyroclastic Kimberlite (Granite Gneiss Rich)

Table 6 : The A/K1 deposit geological description

		Rock Type		Properties
South Pipe	Upper crater	A3T- Bedded talus flow deposits		A planar bedded (mm-cm scale), well-sorted, clast-supported lithic and crystal-bearing kimberlitic sandstone
		EGSST - Epiclastic shales, sandstones and gritstones		A thick sequence of domains of well laminated, fine grained shales inter-bedded with coarser-grained olivine and basalt bearing gritstones.
		DFBB - Debris flow basalt breccia		Coarse to very coarse, poorly-sorted, matrix-supported, basalt-rich breccias.
		BB - Basalt breccia		A coarse-grained, crudely-bedded, poorly-sorted, basalt-rich breccia. The rock matrix is predominately carbonate cement with localised, heavily altered kimberlitic material.
		EBB - Epiclastic boulder beds		Basalt-rich breccias, set within muddy matrix of oxidised kimberlite and sediment
		SVK_U - Volcaniclastic kimberlite – upper		A dark greenish grey, medium to coarse-grained, poorly sorted, olivine and lithic bearing, massive, matrix supported volcaniclastic kimberlite
		SVK_M - Volcaniclastic kimberlite – middle		Greenish grey, medium-grained, olivine and lithic bearing, matrix-supported, volcaniclastic kimberlite, dominated by clasts of basalt with a clay rich matrix
		SVK_L - Volcaniclastic kimberlite – lower		Bedded sequence, of grey to green, olivine and lithic-bearing, matrix-supported, bedded volcaniclastic kimberlite. Comprises abundant olivine macrocrysts and fragments supported by a matrix of serpentine and clay.
		SDVK - Dark volcanistic deposit		A dark green to grey to black medium to coarse-grained, moderately-sorted, olivine-rich, matrix-supported massive volcaniclastic kimberlite, set in a dark matrix of serpentine.
North Pipe		NPK	Pyroclastic kimberlite	Pale grey, fine-medium grained and olivine-bearing, matrix-supported, crudely layered pyroclastic kimberlite. An obvious feature of the rock is the abundance of fine (sub-cm) clasts of basement granite gneiss that gives the rock an almost speckled appearance.
		NPK_GG	Pyroclastic Kimberlite (Granite Gneiss Rich)	Similar in composition and general appearance to NPK, but contains significantly more lithic content, a notable increase in the occurrence of granite-gneiss.
		MVK2	Massive Volcaniclastic Kimberlite	A dull greenish grey, fine to medium-grained, poorly-sorted, lithic and olivine-bearing, matrix-supported, massive volcaniclastic kimberlite.

3.2 Extraction methods

3.2.1 The mine plan & mining method

The A/K1 pit is designed to be mined in three mining Cuts; Cut 1, Cut 2 and Cut 3, utilizing the split shell method, as illustrated in Appendix 1.

The open pit mine design parameters are shown in the Table below:

Table 7: The A/K1 pit design parameters

Ramp width	35m
Ramp Gradient	8%
Minimum Mining Width	80m
Bench Height	15m
Safety Berm Height	2.0m
Surface Elevation	960mbgl
Cut 1 Pit Bottom Elevation	710mbgl
Cut 2 Pit Bottom Elevation	530mbgl
Cut 3 Pit Bottom Elevation	485mbgl
Benches Sunk per Annum	3
Plant Throughput	19.2Mtpa
Haul Truck Size	190t
Haul Fleet Utilization	83%

Figure 12 illustrates the scheduled production levels for A/K1, for ore and waste, for the life of mine up to 2026. The ore tonnes mined fed into the two main treatment plants, Orapa 1 and Orapa 2.

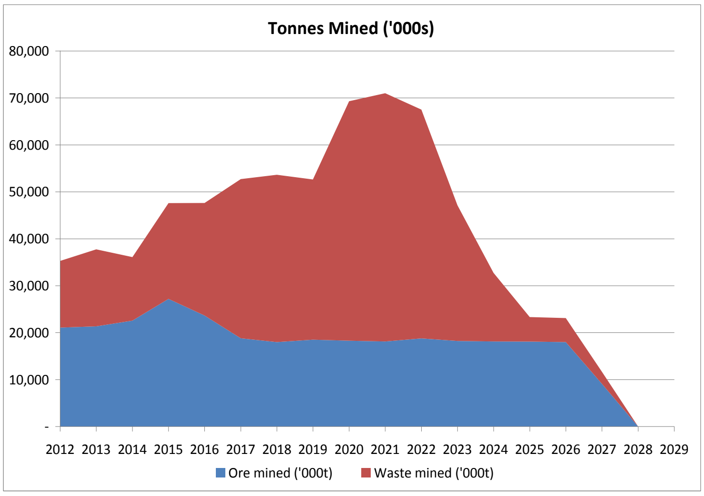


Figure 12: Scheduled production for A/K1

The corresponding recovered revenue from both No.1 Plant and No.2 Plant and the recycle from the recovery plant tailings or old recovery tailings (ORT) is shown in figure 13.

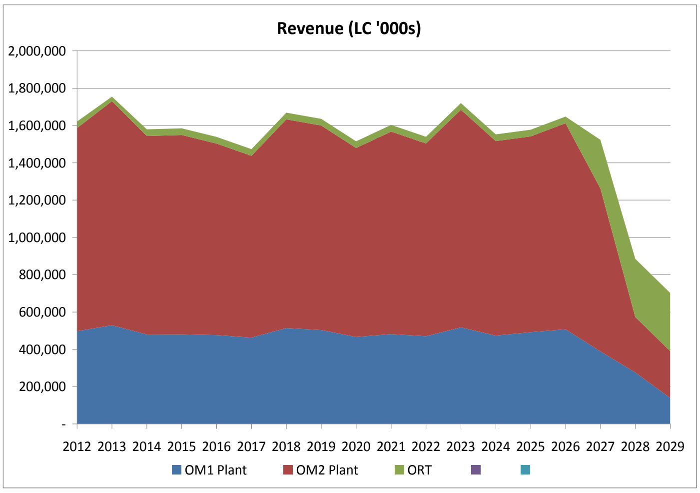


Figure 13: Revenue recovery from A/K1 across life of mine

3.2.2 The treatment method

The tonnes mined from A/K1 pit feed two processing plants, Orapa 1 and Orapa 2. The concept of diamond processing, entails stages of comminution to progressively reduce the particle size of the ore, and screen it into a feed size envelope desirable for the concentration process. A major outcome of the comminution process is *diamond liberation*; a process whereby the locked-up stones are released from the host rocks. Once the diamonds are liberated, they can now be separated from the gangue material into a concentrate by the Dense Medium Separation process.

Liberation is the fundamental activity of any diamond processing operation. The state of liberation in a plant, while not measurable, is diagnosed by the level of grind (fracture) completed by the plant at any given sub-stage of comminution. The total grind conducted by the plant is achieved by a combination of comminution equipment and recycle such that all material is reduced to below a desired size.

The combination of the comminution stages is also known as feed preparation. Subsequently, diamond concentration is achieved through a process known as dense medium separation (DMS). A concentrate is formed by utilizing the density differentials between the gangue or host rock and the diamonds. The final recovery stage is done by using x-ray machines in a separate plant, known as the 'recovery plant'.

3.2.2.1 Comminution stages

- *Primary crushing*

The run-of-mine ore (straight from the pit, ungraded or unprocessed in any way), is hauled to the primary crushers. Orapa has two units, Crusher 2 and Crusher, both with ability to feed either plant. The primary crushers reduce the run-of-mine to less than 150mm.

- *Scrubbing and secondary crushing*

The main function of this stage is to prepare correctly sized feed to the Dense Medium Separation (DMS), and feed envelope for the Orapa plants is 1.65mm to 25mm. Material is reduced to less than the top cut-off size (25mm), and all greater than 25mm material is kept in a recycle loop until repetitive trials eventually reduce the size to less than 25mm. The grits and slimes, material less than the bottom cut-off size (-1.7mm), are screen out for the subsequent thickening.

- *Recrush or tertiary crushing*

Unlike Orapa 1, the Orapa 2 DMS is configured in such a way that it is split into Coarse DMS and Fine DMS. The Fine DMS floats exit the plant as tailings, and the Coarse DMS tailings are transferred to the recrush circuit. The recrush circuit handles the recycling material from the Coarse DMS that is greater than 8mm but less than 25mm, with the intention of crushing further to a product of less than 8mm. The recrush circuit utilizes high pressure roll crushers (HPRC).

3.2.2.2 Dense Medium Separation

The product from scrubbing, screening and crushing is the prepared feed for the DMS. The concept of the DMS entails separating material according to their density using cyclones. Diamonds, together with some heavy minerals present in the feed will sink forming the concentrate. The lighter gangue will float forming the tailings. There is a difference between the Orapa 1 and Orapa 2 DMS plants. In the case of the Orapa 2 plant, the DMS is split into Coarse DMS and Fine DMS. The Fine DMS treats material of size range between 1.65mm and 8mm, and the Coarse DMS treats material of size range between 8mm and 25mm. Orapa 1 has a combined DMS, implying that the plant treats material of size range 1.65mm to 25mm. All the DMS floats at Orapa 1 are discarded as tailings, whilst for Orapa 2 only the floats from the Fine DMS are discarded as tailings. The DMS floats on the coarse side is retained in a closed circuit with the recrush circuit until the material is reduced to less that 8mm and eventually exits the plant.

3.2.2.3 Slurry handling and disposal

The slurry handling and disposal circuit's primary functions are to recover process water from the slurry for reuse, as well as dispose of the grits and slimes (slurry) i.e. material less than 1.65mm.

The primary units of the circuit are the thickeners, where slurry settling occurs and clarified water is recovered for reuse in the plant. The thickening process entails the solids in suspension (slurry) settling under the influence of gravity, forming higher density slurry as the underflow and recovering clarified water as overflow. The underflow is continuously removed by use of underflow pumps, and the clear water at the top of the thickener overflows into a launder gravitating to the clarified water storage tanks. The thickening process is aided by the use of flocculants, which make fine particles agglomerate to former larger and easier to settle particles or flocs. The flocculant is one of the key consumables in the treatment process making it one of the major cost drivers. Grits and slimes form a major component of the tailings that exit the plant, making slurry handling a key component of the diamond processing plant in terms of magnitude and costs.

The difference between Orapa 1 and Orapa 2 in this circuit is the fact that Orapa 2 uses a special type of thickeners known as 'ultraseps', whereas Orapa 1 has conventional thickeners and an additional ultrasep for additional capacity and redundancy.

3.2.2.4 Process flowsheets

The Orapa 1 and Orapa 2 simplified process flowsheets are shown in Figure 14 and Figure 15 below:

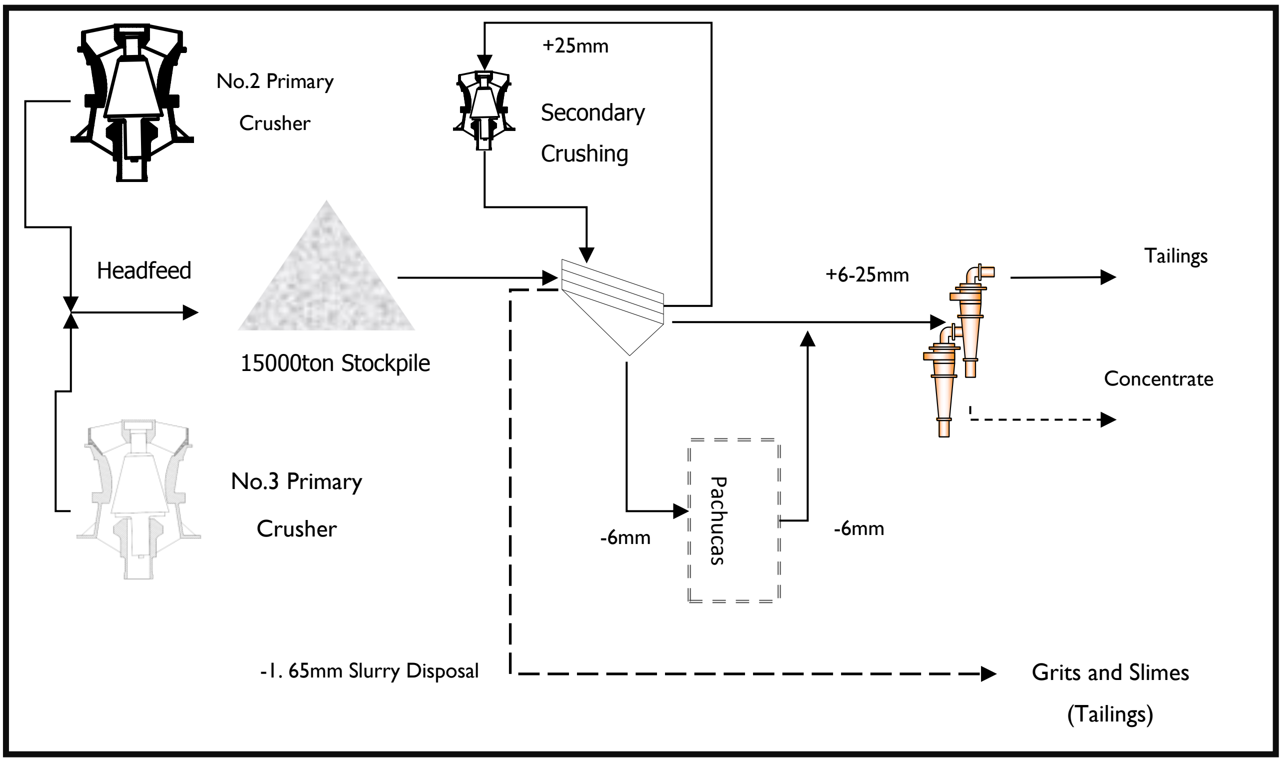


Figure 14: Orapa 1 main treatment plant flow-sheet

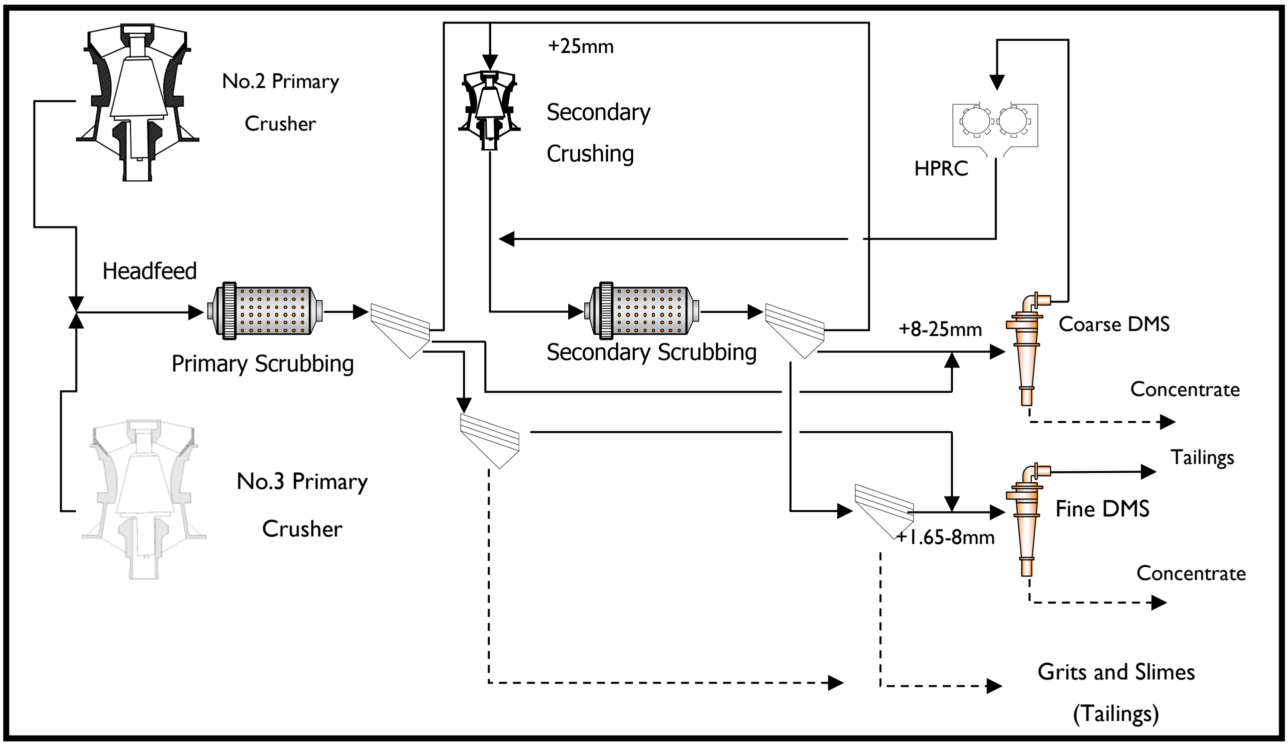


Figure 15: Orapa 2 main treatment flow-sheet

3.2.3 Ore Dressing studies (ODS) – characterization of key lithofacies within A/K1

The first conceptual ore dressing study was completed in 2002, and was done by DebTech, the DeBeers technical division based in Johannesburg. The study aimed at understanding the metallurgical properties of the North and South lobe of the A/K1 ore body between the surface and 400mbgl level.

As the Orapa Resource Extension Project (OREP) was initiated in 2006, a more comprehensive ODS was needed in order to develop an integrated AK1 resource model that would offer further understanding of the geological, geo-technical and to some extent 'geometallurgical' characteristics of the ore bodies in both the North and South lobes to levels below 600 mbgl. A total of 158 drill core samples for ore characterization where taken to DebTech. An approximate, further 15 tonnes of four different ore types were taken to Paterson & Cooke Consulting Engineers (PCCE) in Cape Town for slimes thickening and deposition pilot scale tests. Three ore types were taken for crushing testwork at Kawasaki Heavy Industries (KHI) in Japan, the original equipment manufacturers for the primary and secondary crusher models currently being used at Orapa Mine.

The results generated from both the internal (DeBeers Family of Companies and DebTech) and external laboratories including pilot scale tests were collated, and interpreted to produce a full data pack. The results are summarized to illustrate the expected behaviour of the main lithofacies at A/K1 within the treatment plants.

3.2.3.1 Communition studies

Comminution characterisation aims at determining ore properties and how they relate to the comminution unit processes, with varying depth, spatial location and or how they relate to a particular rock type. This form of characterisation is used to indicate potential changes in the comminution behaviours of the material within the ore body.

The testwork consists of rock mechanics tests, the geo-mechanical tests that assess the impact breakage characteristics of the material. This particular test essentially measures the ore-specific energy per size reduction behaviour. To measure the material's abrasion resistance, an abrasion test is also conducted. The tests are performed to determine the responsiveness of the different facies or geological zones to comminution by both impact and abrasion forces.

In summary, communition characterisation objectives are:

- Characterization of geo-mechanical behaviour of the ore
- Spatial profiling of the comminution characteristics within the ore body

The hardness and breakage properties that determine comminution behaviours of the key lithofacies at A/K1 are summarized in the table below:

Table 8: Hardness and breakage properties for A/K1 rock types

	Rock Mechanics	Drop Weight	
	Yield Strength	Impact Breakage	Abrasion Breakage
North Lobe			
MVK1	Low	Soft	Moderate Soft to Soft
MVK2	Low-Medium	Hard	Medium
NPK	Low	Hard to Moderate Hard	Moderate Hard to Medium
NPK_GG	Low-Medium	Hard to Moderate Hard	Moderate Soft to Soft
BB	Low to High	Hard to Soft	Moderate Soft
South Lobe			
BB	Medium to very High	Hard	Hard
SVK_M	Low to very Low	Very Soft	Soft
SVK_U	Very Low to Medium	Soft to Very Soft	Moderately Soft

In general the North lobe ore facies should be relatively easy to treat compared to the South lobe where the much softer rock types are to be blended with the much harder basalt breccia. Basalt breccia is the hardest rock type and experience has shown than if it is unblended, it leads to higher liner wear rates on crushers, increased amounts of recirculation in the comminution circuits and much lower throughput.

3.2.3.2 DMS studies

The Dense Medium Separation (DMS) process is the primary diamond concentration stage, prior to the subsequent final concentration using the x-ray technology. The DMS is a critical stage where high end efficiencies are required to avoid diamond losses to tailings. Inefficiencies can also lead to excess concentrate generation that may result in overloading of the subsequent x-ray recovery process.

The DMS characterization is based on 'densimetric analyses' and provides an indication of the theoretical mass percentage of feed that will report to the concentrate stream of a DMS process. This is known as the theoretical yield. Screening analyses provide information in terms of expected yields per size fraction.

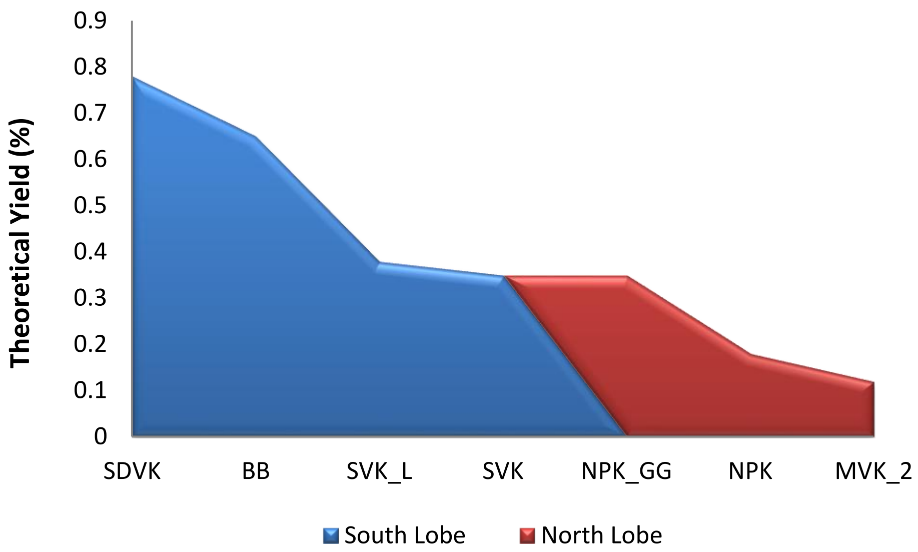


Figure 16: Theoretical yield generation for A/K1 rock types

The general design criteria for diamond processing DMS plant yields are around 1% or less, with yields of 0.5% being considered common. From the OREP ODS results above, the yields achievable with the A/K1 can be considered to be acceptable.

3.2.3.3 Slurry disposal and water recovery studies

The aim of the slurry disposal and water recovery studies is to characterize the slimes and determine treatability behaviour. Efficient treatment of the slimes is particularly crucial for Orapa to satisfy the operation's strong water conservation strategy. The semi-arid climate requires the treatment plants to recycle as much process water as possible. Thus behaviour of the water recovery circuit is of utmost importance not only for operational efficiencies, but for the conservation of water as well.

The 2002 ODS was comprehensive in the mineralogy analysis, slurry cation content determination as well as determining the reagent consumption rates. The subsequent 2009 ODS slurry and water recovery tests confirmed the 2002 study results, but was not as comprehensive as the later.

The mineralogy analysis was conducted on the Orapa slurries to determine the amount of minerals present. Results obtained between the 2002 study and the 2009 study were quite similar. The total mineral analysis results indicated that the samples were composed of a wide range of minerals that includes serpentine, smectite, feldspar, goethite, calcite, dolomite, mica, and quartz.

The mineralogy analysis results for the North Lobe and South Lobe are summarized in figure 17 and figure 18 below.

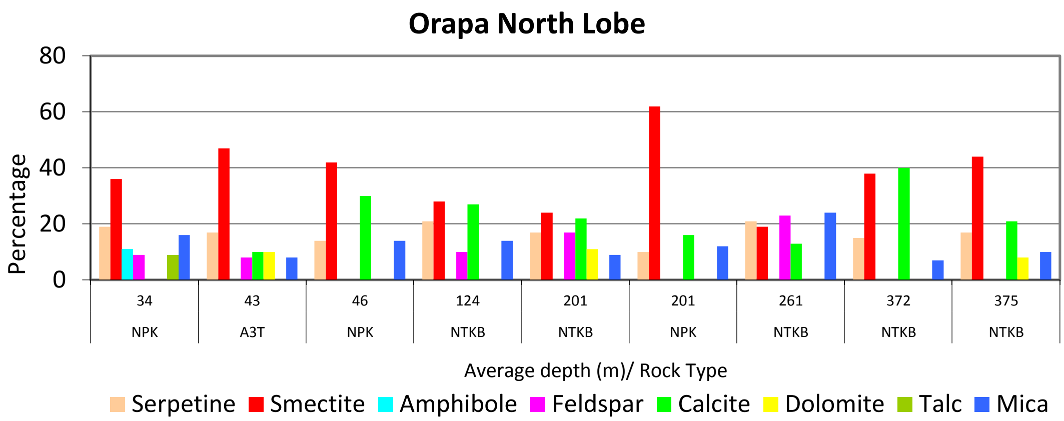


Figure 17: A/K1 North Lobe total mineral analysis

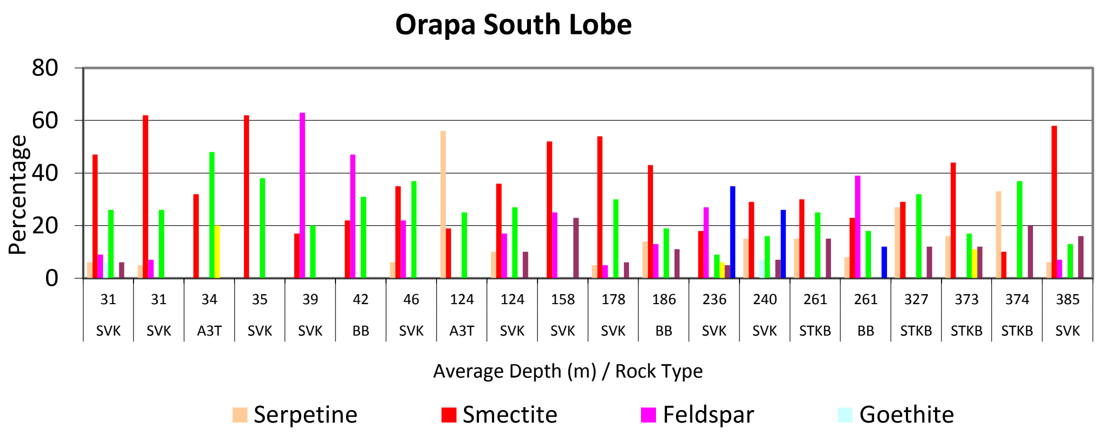


Figure 18: A/K1 South Lobe total mineral analysis

Smectite is an important indicator of slimes settling behaviour, and its presence in the slurries can indicate potential solids settling difficulties during the solid-liquid separation or the thickening process. Some smectite clays swell when they are in contact with water and may result in generation of further ultra-fines.

The other tests performed to determine the slurry settling behaviour were the exchangeable cation tests. Exchangeable cations are often adsorbed on the clay crystal lattice structure and can be exchanged by other ions in solution when in contact with water. The exchangeable sodium percentage (ESP) is an indication of the cation-exchanged state of the clays of the ore in the dry or in situ state and is determined from cation extraction test.

Exchangeable sodium percentage in excess of 15 percent indicates a potential of generating non-settling settling slurry. Partial settling slurry is expected when the ESP values are between 10 and 15 percent, while easy settling slurry is expected for ESP values below 10 percent. The ESP results obtained per rock type and depth for the North Lobe and South Lobe are summarized in figure 19 and figure 20 below.

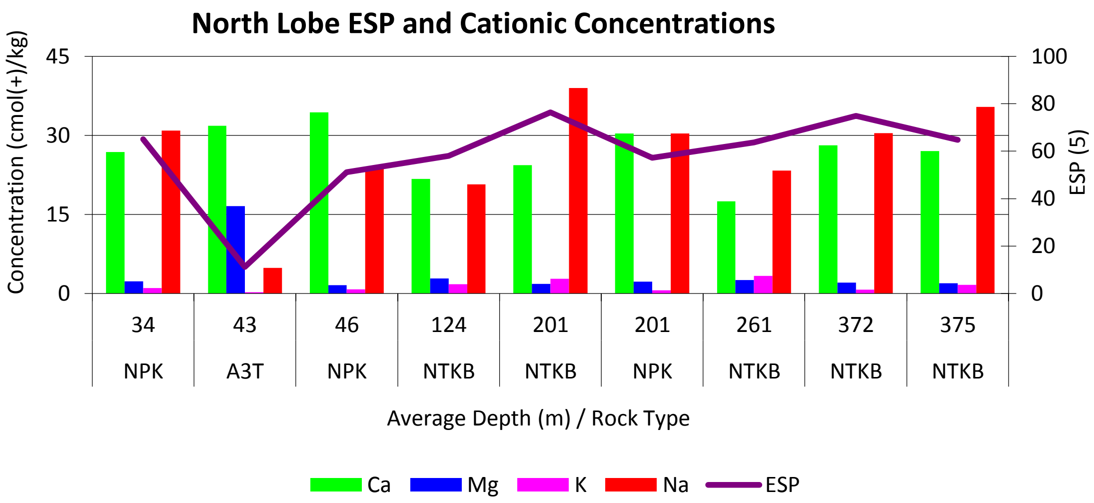


Figure 19: North Lobe exchangeable sodium percentage and cationic concentration

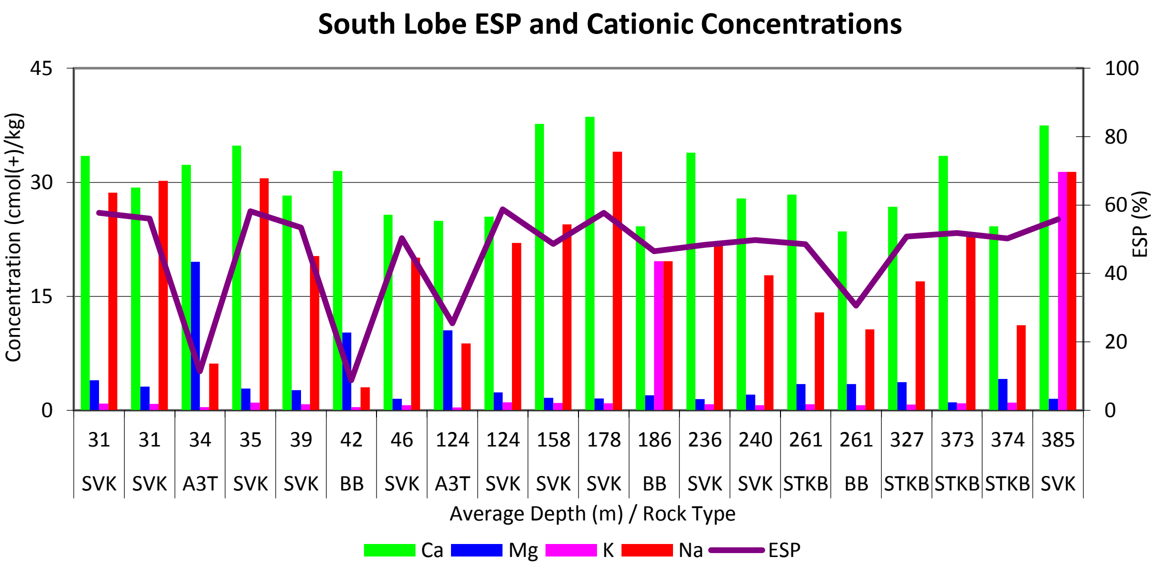


Figure 20: South Lobe exchangeable sodium percentage and cationic concentration

The measured ESP values range between 9 and 76 percent with sample A3T recording the lowest value for both the North Lobe and South Lobe. Except for A3T ore, all the other ore types show potential to generate non-settling slurries when in contact with water.

The flocculant consumption rates were estimated to range from 20g/ton to 80g/ton in the 2002 study. The study also showed that the NPK and SVK rock types do have potential to have a consumption rate of above 60g/ton. The most effective flocculant type selected universally for the AK1 rock types was Magnafloc 5250.

The 2009 study, however, estimated consumption rates of 10g/ton -35g/ton for the North Lobe slurries and 10g/ton – 30g/ton for the South Lobe slurries. The most effective flocculant type selected universally for the AK1 rock types was Magnafloc 5250, consistent with the 2002 study.

The slurry handling and water recovery tests results for both the 2002 and the 2009 Ore Dressing Studies show the potential for settling problems with the AK1 rock types.

The 2002 study does emphasize the risk in its conclusion.

"*Total mineral analyses indicate that Orapa samples are composed of a wide range of minerals that have a potential to cause slime settling problems. The ESP values of Orapa material are high and with the exception of A3T ore, may result in low settling rates. The ESP values were close to that of Jagersfontein where the highest ESP values in the De Beers group were recorded. Particle size distribution indicates that Orapa samples have a medium to high quantity of ultra-fine -20 micron size fraction and require high flocculant dosage."*

The 2009 study does not emphasize the risk, an abrupt assumption is made stating that the raw water (Orapa water) is of high conductivity and will thus aid with coagulation and settling of the slurries.

"Analysis of the dry in-situ ore characterisation with respect to ESP and Cation Exchange Capacity indicate a potential to generate stable slurry suspensions. However, when this material is in contact with the high conductivity water (≥ 3 mS/cm), as the case with Orapa water, it is expected to generate slurry that is classified as natural coagulating and easy settling."

Both the 2002 and the 2009 Ore Dressing Studies do not show any other potential treatment related challenges with the AK1 ore types, except for the slurry handling and water recovery circuit.

3.2.4 Orapa 2 thickening design decisions (selection of ultraseps)

Orapa Number 2 plant was commissioned in the year 2000, and the goal of the project was to increase the A/K1 output from 6 million carats per annum to 12 million carats per annum.

Due to water scarcity in Botswana, thickening is an important unit process in ore treatment plants. Water conservation has always been one of Debswana Diamond Company's major strategic areas. As such, a lot of research had been done over the decades to identify the most appropriate thickening technology. After extensive test campaigns, Debswana installed the Tasster at the Orapa 1 plant in 1991. This is a French designed raked paste thickener, and it has worked quite well for the application.

Following this success at Orapa 1, further test work was conducted and additional thickening technologies were tested to identify suitable and more efficient applications in diamond kimberlite slurries. The ULTRASEP, designed and installed by Bateman South Africa, was marketed as a cheap option for thickening in both capital and operating costs. The ultraseps thickeners were expected to offer low flocculant consumption, low maintenance costs as well as having an advantage of a small foot print which would minimize space requirements in the new plant. Before the Orapa 2 installation, non-raked thickeners had not been used in large tonnage applications. After some test work, Debswana proceeded to install eight 10 m diameter Ultraseps at Orapa 2, making it the first large tonnage project to use thickeners with no moving parts for slurry handling and water recovery.

As a result of this installation, many other projects followed suit such as Exxaro's Hillendale Mineral Sands Mine in Natal. Soon after, De Beers also adopted the technology for the Kimberly Combined Treatment Plant (CTP).

3.2.5 Thickening constraint description

The ultraseps are a simplified approach to thickening but highly sensitive to variability in the ore quality and clays content. Though the ultraseps performed reasonably well in the early years of the Orapa 2 project, signs of emerging constraints were being noted. An additional paste thickener was installed on justification of better water retention but was never adequate to alleviate the emerging constraint. The paste thickener installation project was badly managed and the thickener never produced the desired high density slurries and is dogged by frequent mechanical failures. The positive displacement pumps meant to pump high density slurry (or paste) to the tailings dump has never worked, and the central gear box for the rotating rake has low reliability. The operation of the ultraseps has been worsened by the appearance SVK_M and NPK_GG. These rock types produce non-settling slurries, making the operation of the slurry handling section extremely challenging.

The non-settling slurries would imply muddy clarified water (or overflows) from the ultraseps (referred to as 'sliming' of thickeners in metallurgical terms). The muddy water would cause high viscosity of the medium in the DMS plant due to contamination, leading excessive concentrate generation and filling up concentrate bins. This would force slowing down of the head-feed rate or even complete stoppages. The cause of such delays is referred to as sliming ultraseps, and occurs as a result of poor blending and high clay content in the ore blend.

4 METHODOLOGY (FOR ORAPA GEOMETALLURGICAL RISK DEMONSTRATION)

The 2009 model realised that some rock types, not previously identified in previous geological studies, existed in the A/K1 deposit. This realization led to a sub-classification of the volcanistic (SVK) and pyroclastic (NPK) kimberlites, into SVK_Upper, SVK_Medium and SVK_Lower, as well as NPK and granite rich NPK_GG. When the mining level was at 210mbgl, the SVK_M and NPK_GG rock types were encountered, this coincided with slurry settling challenges in the plant and subsequent excess concentrate generation that led to head feed delays and increased flocculant consumption. This has led to suspicions that the rock types are linked to slurry settling challenges. The author has thus assumed existence of geometallurgical risk at the operation, which is causing a considerable loss in revenue due to lack of flexibility to deal with the risk.

To investigate existence of such risk at Orapa Number 2 Plant, the following methodology was followed:

- An analysis of historic data;
- Regression analysis to determine the relationships between ore mix and process performance ; and
- Financial evaluation of the impact of ore body variability on process efficiency.

The investigation demonstrates an introduction of a constraint in the slurry handling and water recovery circuits and how this adversely impacts throughput and operating costs. The slurry settling challenges are linked to the two rock types that were previously unidentified in geological models prior to 2009. The data used in the research was extracted from the mine production databases and monthly reports.

4.1 Ore mix analysis on head feed

From the daily mining reports, the ore mix data was compiled into monthly and yearly summaries. By plotting a historical time series graph, an analysis was made to identify changes in rock types or mine mix into the plant with time.

4.2 Analysis into the impact of settling challenges on plant throughput

(i) Ore settling delays were extracted from the plant delays database, illustrated in Appendix 2.

(ii) Historical time series graphs were plotted to analyse trend in head-feed delays caused by slurry settling problems.

(iii) A statistical analysis was done to compare the slurry settling delays before SVK_M and NPK_GG were encountered in the mine mix.

4.3 Analysis of flocculant consumption

(i) Flocculant consumption data was extracted from the monthly production reports from 2003 up to the year 2011.

(ii) Historical time series graphs were plotted to analyse trend in the flocculant consumption.

(iii) A statistical analysis was done to compare the flocculant consumption before SVK_M and NPK_GG were encountered in the mine mix.

4.4 Investigation of correlation between flocculant consumption and settling delays

An appropriate time series graph was plotted to analyse the variation of flocculant consumption and delays due to slurry settling challenges in the thickener.

Using a scatter plot, the correlation between flocculant consumption and plant delays resulting from thickener settling challenges was investigated. The aim of the analysis was to establish a relationship that will confirm that settling challenges resulted in excessive use of flocculant, and to determine whether they are emanating from a common problem.

4.5 Investigation of the rock types responsible for the thickening constraint

Flocculant consumption was used as a proxy to thickening challenges. In order to identify the rock types responsible for the settling changes, a multi-linear regression analysis was done. Taking the rock types as independent variables, the multi linear regression technique predicts the outcome of one dependent variable, in this case the rate of flocculant consumption. By making such a prediction, the relative influence or relationship of each rock type with flocculant consumption is determined. The analysis thus enables us to identify the rock types that greatly impact flocculant consumption. The major rock types that have been consistently present in significant quantities in the head feed are used as independent variables in the analysis.

4.6 Financial impact estimation

To estimate the financial impacts of the ore settling treatability challenge, an ore processing model is assumed. The model, in the form of an equation, is based on two major aspects of the overall recovery, liberation and efficiency. To estimate the recovered revenue, the head grade and the diamond price per carat are factored in, and this is shown in equation (2). The treatment cost element is derived by consideration of sectional costs, and this is shown in equation (3).

The revenue per tonne recovered is given by:

$$\text{Revenue/tonne} = \{\$/C \times [G \times LF] \times EF\} \quad (2)$$

And the cost per tonne per section in the plant is given by

$$\text{Cost/tonne} = \sum_{1}^{n} C_k \quad (3)$$

Where:

$/C = Dollar per carat

G = Insitu grade above plant bottom size, expressed in carats/tonne

LF = Liberation factor

EF = Efficiency factor, process dependent

C_k = The cost incurred per section per head feed tonne processed

To approximate revenue losses due to the settling delays, the *revenue per hour* version is obtained by the multiplication of equations (2) and (3) with the expected hourly throughput. This is the plant name plate capacity of 1 400 tonnes per hour.

5 RESULTS AND DISCUSSION

5.1 Ore mix analysis on head feed

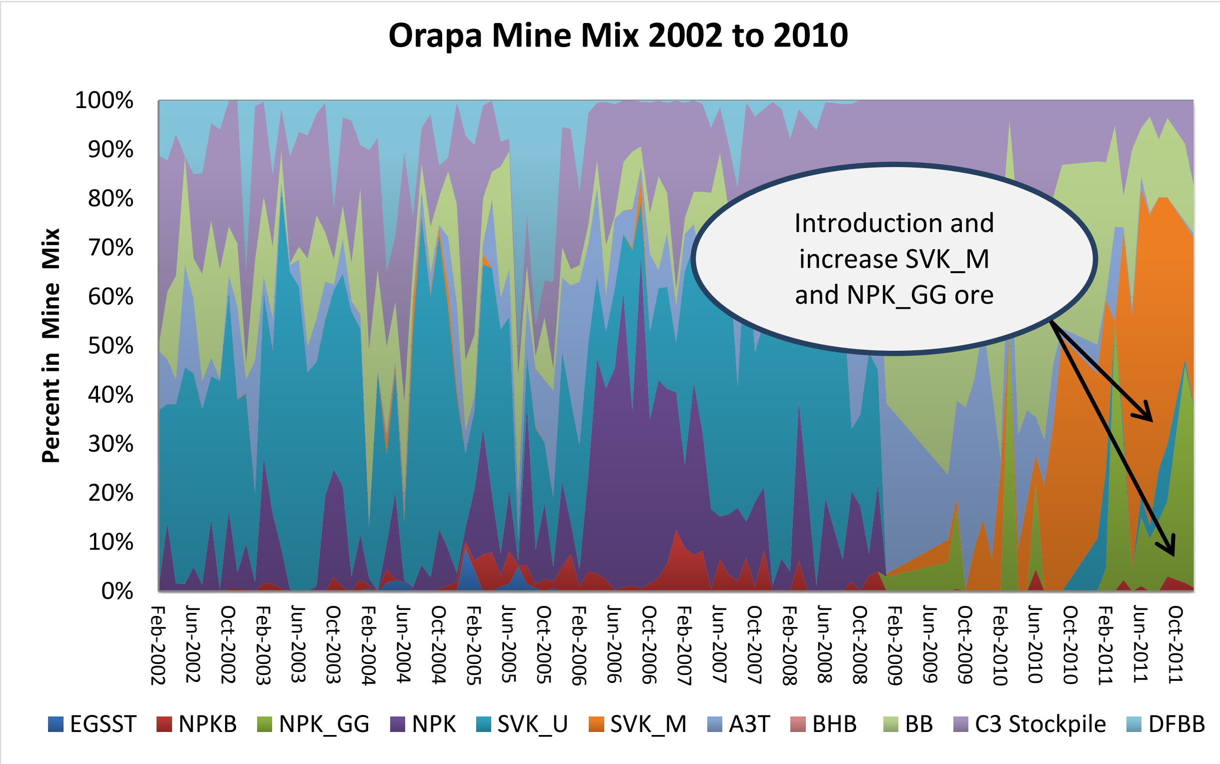


Figure 21: Orapa 2 mine mix January 2002 to December 2011

Figure 21 above shows that before February 2009, the SVK_M and NPK_GG were not present in the mine mix, or the blend that was fed into the plant. The two rock types have been in the ore blend since February 2009 with noted increase in prevalence.

5.2 Delays due to slurry settling challenges

From the delays database, head feed stoppages that were caused by challenges in slurry thickening problems (‘sliming’ ultraseps), were extracted. A time series staged individual chart below demonstrates the statistical differences in delays due to slurry settling between two stages, that is, before SVK_M and NPK_GG were encountered as part of the reserve and fed into the plant, and after the rock types were fed into the plant.

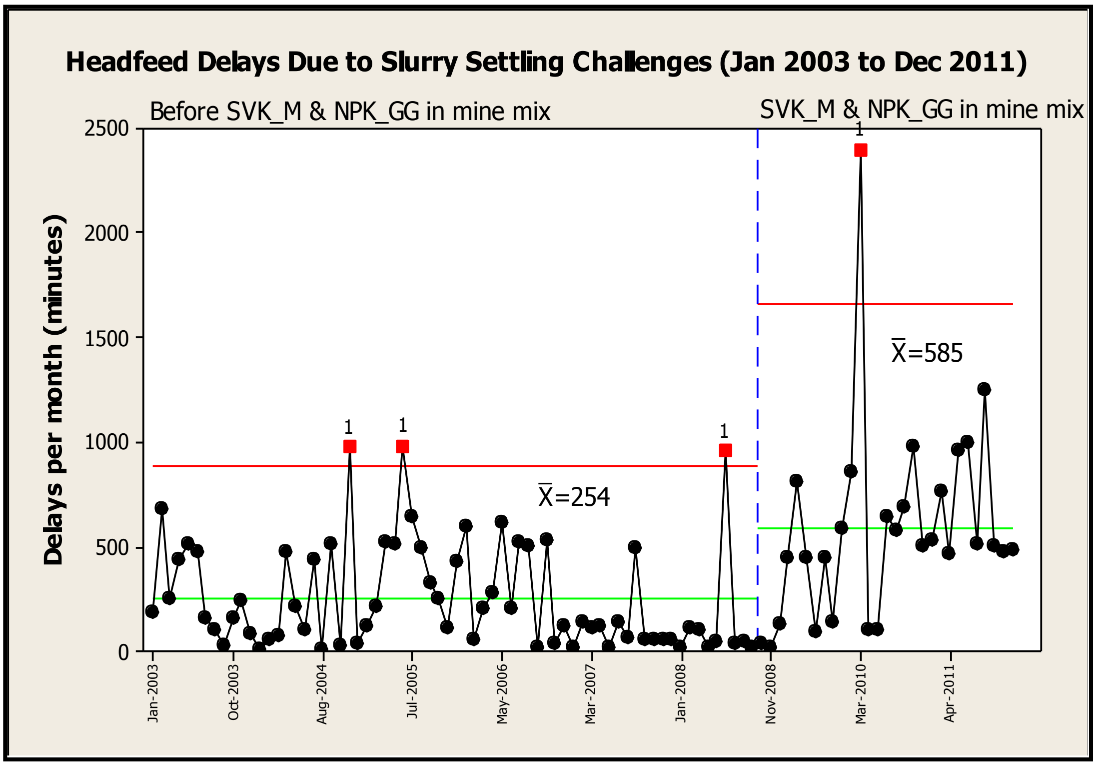


Figure 22: Head feed delays due to slurry settling challenges

Figure 22 shows an increase in slurry settling delays after introduction of SVK_M and NPK_GG into the ore mix. The monthly average increased from 254 minutes before the rock types were encountered, to 585 minutes after the rock types were realized in the reserve and in the plant feed. (With outliers removed, the averages are 221 minutes and 521 minutes, before and after introduction of SVK_M and NPK_GG. A time series staged individual chart with outliers removed is shown in Appendix 7). There is noted increase in variability in the slurry settling delay parameter, as represented by the change in the standard deviation from 253.35 before the rocks were encountered, to 470.71 after the rocks were encountered. The increase in both the average delays and the associated standard deviation indicates a decrease in process control capability after introduction of SVK_M and NPK_GG into the plant feed.

Figure 23 shows the statistical summary of the slurry settling delays before SVK_M and NPK_GG were introduced in the mine mix.

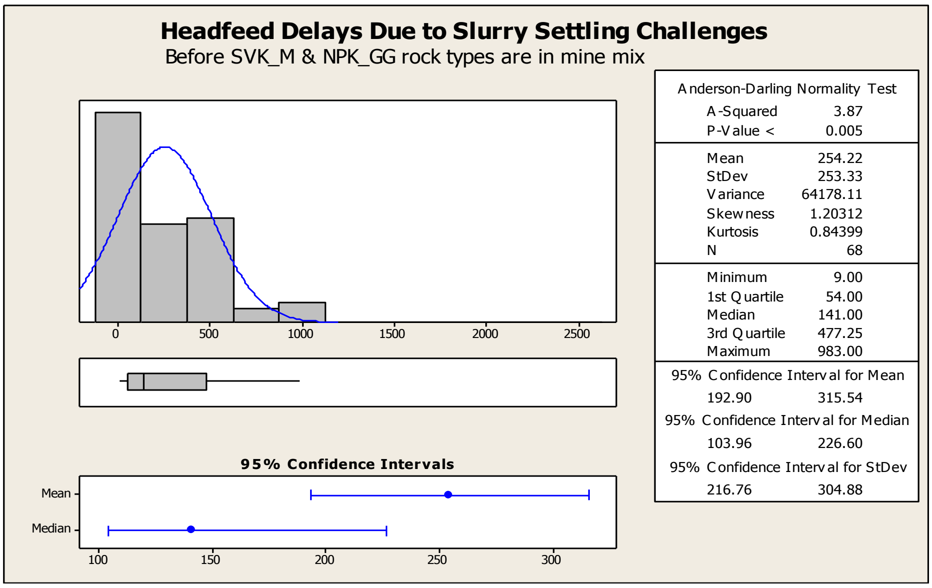


Figure 23: Head-feed delays due to slurry settling before SVK_M and NPK_GG are added to mine mix

Figure 24 below shows the statistical summary of the slurry settling delays after SVK_M and NPK_GG were introduced in the mine mix.

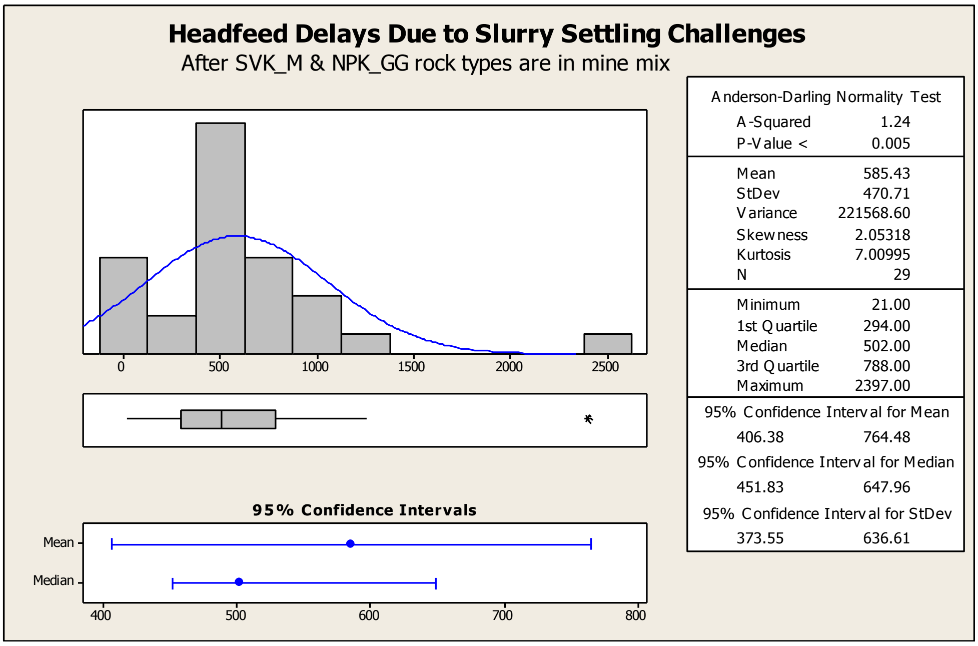


Figure 24: Head-feed delays due to slurry settling after SVK_M and NPK_GG are added to mine mix

The significance of the quartiles in the statistical summaries above is to show that 25% of the values in a distribution are below the 1st Quartile, and 25% are above the 3rd Quartile. The statistical summaries above show that before introduction of the two rock types, 50% of the monthly delays were between 54minutes and 477minutes. After introduction of the rock types, 50% the delays were between 294 minutes and 788 minutes. The 95% confidence intervals show that without SVK_M and NPK_GG in the mine mix, the delays due to ultraseps sliming would likely be between 193 minutes and 316 minutes, with the two rock types in the mine mix the delays are likely to be 406 minutes and 765 minutes. The analysis shows the two rock types have increased the occurrence of delays due to settling challenges. The statistical test was conducted to confirm the difference in the mean of the two sets of data. A hypothesis test was done in the form of a Two Sample T-Test using Minitab statistical software. The hypotheses were set as follows;

Null hypothesis (Ho): The mean of the Delays before SVK_M & NPK_GG are in the plant feed is statistically similar to the mean of the Delays after SVK_M & NPK_GG GG are in the plant feed.

Alternate hypothesis (Ha): The mean of the Delays before SVK_M & NPK_GG are in the plant feed is significantly less than the mean of the Delays after SVK_M & NPK_GG are in the plant feed.

The test results show a p-value of less than 0.05. This rejects the null hypothesis confirming there is a statistical difference between the two data sets, and that there were fewer delays due to settling challenges before the two rock types were introduced into the plant feed. Appendix 3 is a summary of results of the hypothesis test.

5.3 Impact of the new rock types on flocculant consumption

The monthly flocculant consumption rates, in grams per head feed ton processed, were compiled from January 2003 to December 2011 and illustrated in the individual chart below.

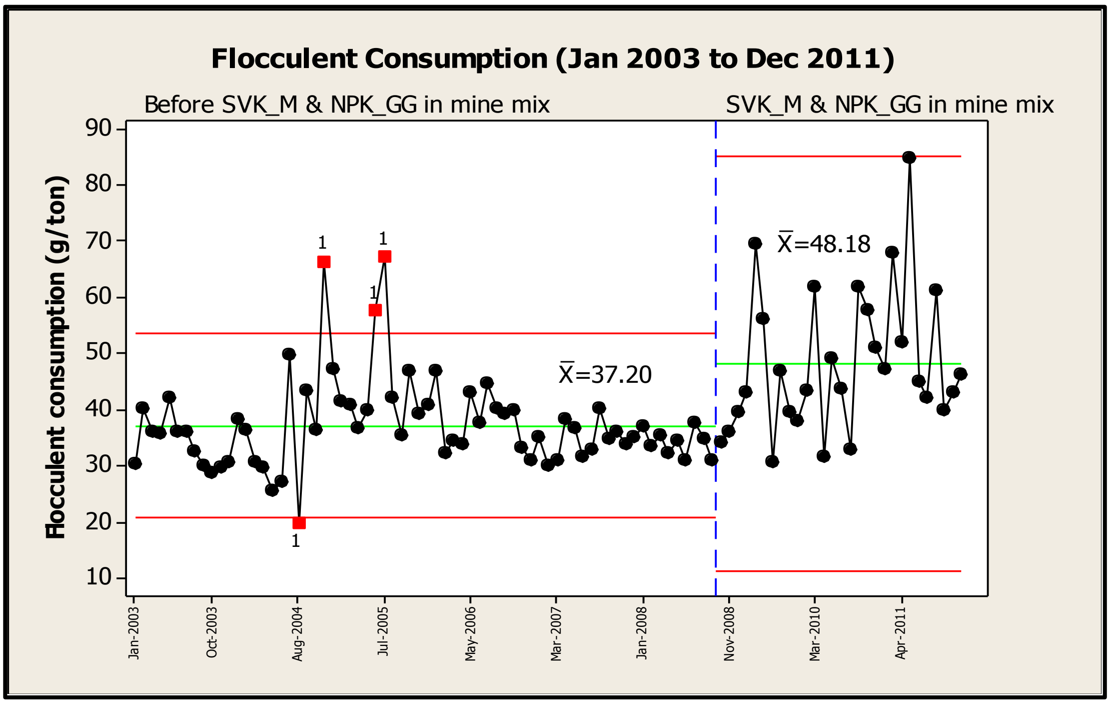


Figure 25: Individual chart for flocculant consumption

For the period from January 2003 to December 2011, the monthly flocculant consumption averaged 40.49 g/ton of head feed of ore processed. The standard deviation for the variable during this period was 10.79.

Figure 26 shows the statistical summary of flocculant consumption before SVK_M and NPK_GG were introduced in the mine mix.

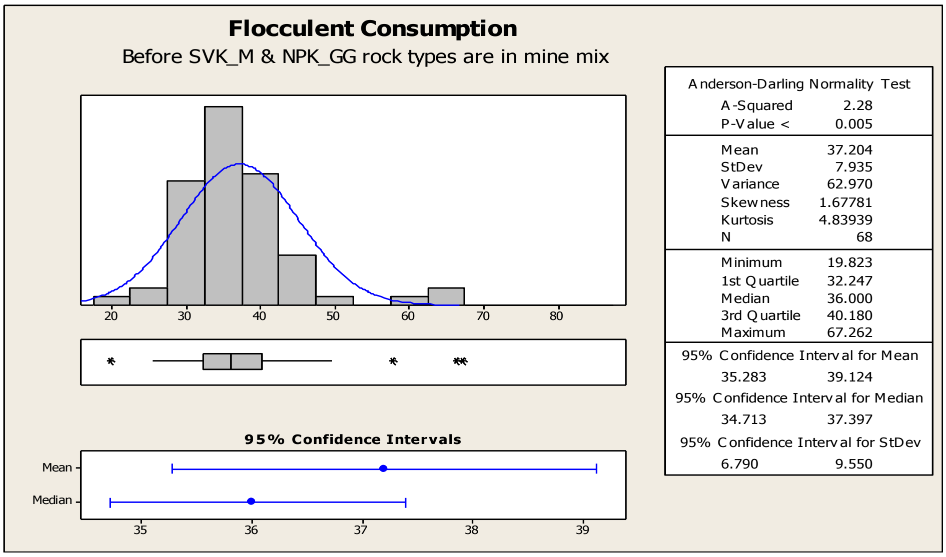


Figure 26: Flocculant consumption before SVK_M and NPK_GG are in mine mix

Figure 27 shows the statistical summary of flocculant consumption after SVK_M and NPK_GG were introduced in the mine mix.

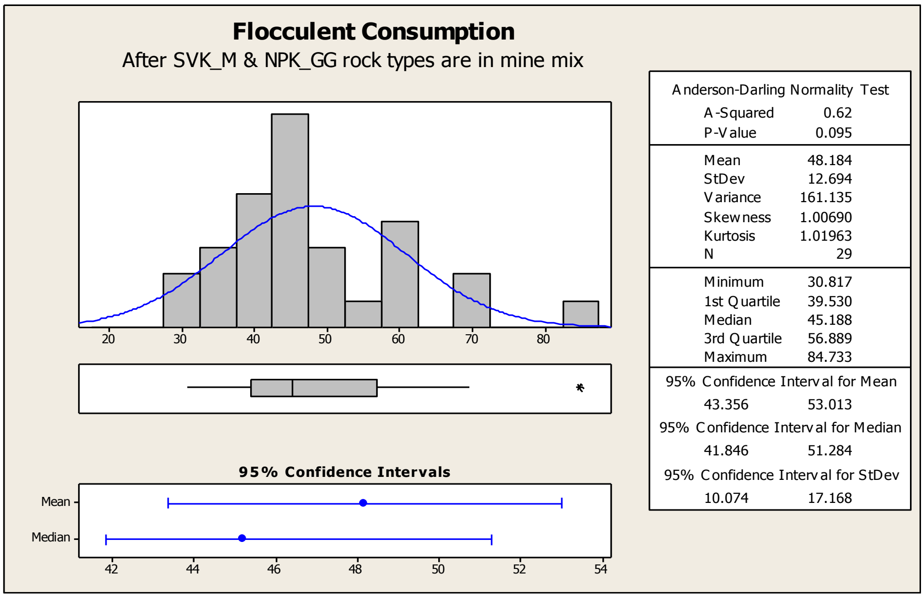


Figure 27: Flocculant consumption after SVK_M and NPK_GG are in mine mix

The statistical summaries show an increase in flocculant consumption after introduction of SVK_M and NPK_GG into the ore mix. The monthly average increased from 37.20g/ton before the rock types were encountered, to 48.18 after the rock types were realized in the reserve and in the plant feed. There is a noted increase in variability in the slurry settling delay parameter, as represented by the change in the standard deviation from 7.94 before the rocks were encountered, to 12.69 after the rocks were encountered. The increase in both the average delays and the associated standard deviation indicates a decrease in process control capability after introduction of SVK_M and NPK_GG into the plant feed.

The significance of the quartiles in the statistical summaries above is to show that 25% of the values in a distribution are below the 1st Quartile, and 25% are above the 3rd Quartile. The statistical summaries above show that before introduction of the two rock types, 50% of the monthly consumption values were between 32.2g/ton and 40.2g/ton. After introduction of the rock types, 50% of the consumption values were between 39.5 g/ton and 56.9 g/ton. The 95% confidence intervals show that without SVK_M and NPK_GG in the mine mix, the average consumption rate would likely be between 35.3g/ton and 39.1 g/ton, with the two rock types in the mine mix, average consumption is likely to be between 43.4g/ton and 53.0g/ton.

The analysis shows the two rock types have increased the flocculant consumption as a result of settling challenges. A statistical test was conducted to confirm the difference in the mean of the two sets of data. A hypothesis test was done in the form of a Two Sample T- Test using Minitab statistical software. The hypotheses were set as follows;

Null hypothesis (Ho): The mean for the Flocculent Consumption before SVK_M & NPK_GG GG are in the plant feed is statistically similar to the mean for the Flocculent Consumption after SVK_M & NPK_GG GG are in the plant feed.

Alternate hypothesis (Ha): The mean for the Flocculent Consumption before SVK_M & NPK_GG is less than the mean for the Flocculent Consumption after SVK_M & NPK_GG GG are in the plant feed.

The test results show a p-value of less than 0.05. This rejects the null hypothesis confirming there is a statistical difference between the two data sets, and that the flocculent consumption was significantly less before the two rock types were introduced than after. Appendix 4 is a summary of results of the hypothesis test.

5.4 Correlation of flocculant consumption and slurry settling delays

A correlation can be deduced by a simple regression that is carried out to determine the relationship between a dependent variable and an independent variable. (Orlov, 1996). The scatter plot in Figure 28 shows the relationship between the slurry settling delays and flocculant consumption. With an R-squared value of 0.50 (slightly increases to 0.54 with outlier removed), the two variables exhibit a strong correlation, and in this case, this demonstrates a common source of variation in the two variables.

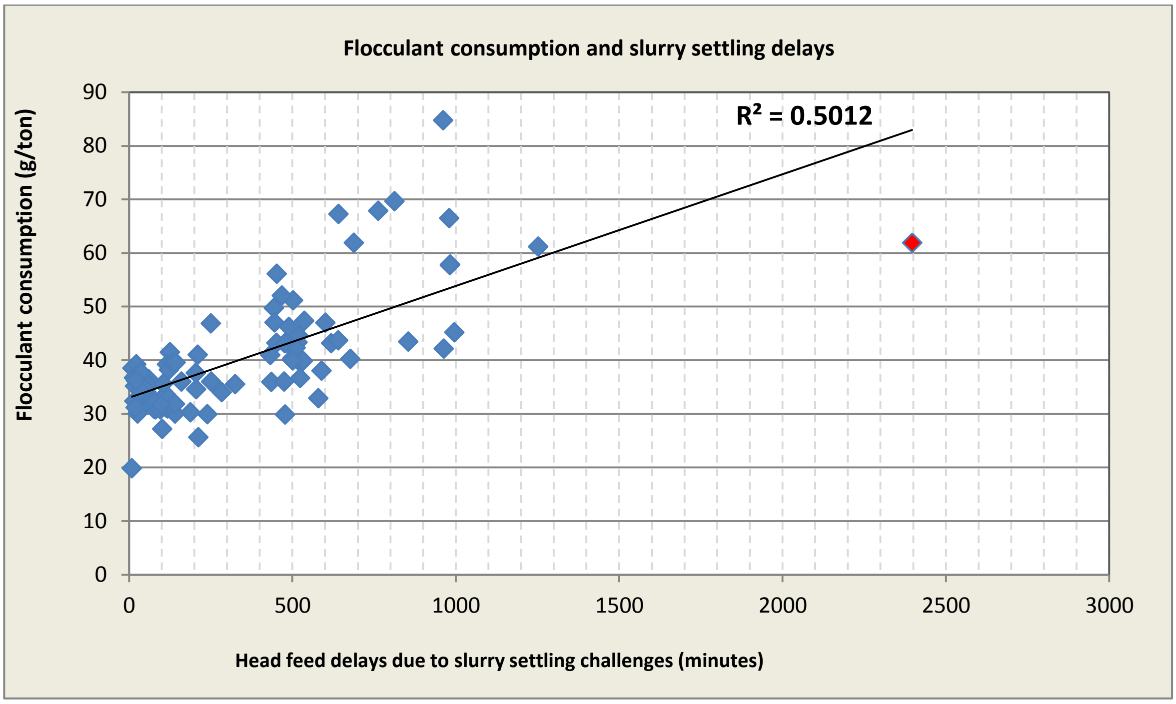


Figure 28: Correlation of flocculant consumption and delays due to slurry settling challenges

Figure 29 is time series trends for flocculant consumption and percentage contribution of slurry settling delays to total plant delays. The increase in slurry settling delays coincides with the sudden increase in flocculant consumption.

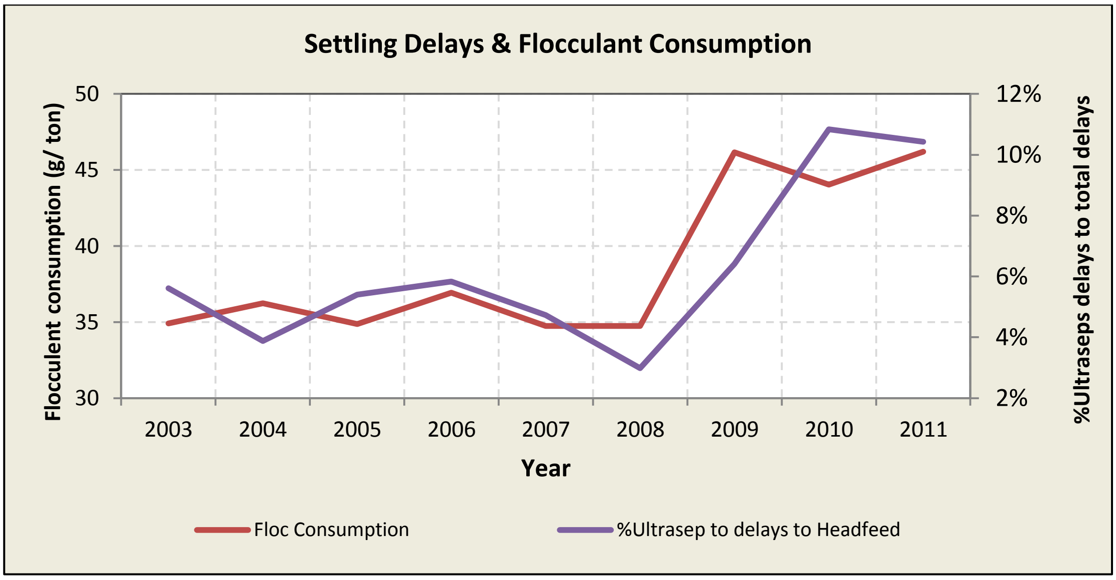


Figure 29: Delays due to slurry settling challenges and flocculant consumption

5.5 Process response behaviour to mine mix

Slurry settling challenges (or 'sliming' ultraseps) can be considered as a process response to high clay content slurries, resulting in head feed delays and increased flocculant consumption. The previous analysis showed a strong correlation between the slurry settling delays and flocculant consumption, an indication that the two are linked to a common cause. Either of the two variables can be used as a proxy to investigate rock types responsible for the settling challenges. To determine the impact of the rock types on flocculent consumption, a multiple linear regression technique was used.

Multiple linear regression can be considered to be a form of broadened simple regression, where a set of multiple (or more than one) dependent variables are assumed to be directly related to the output variable. (Orlov, 1996). A multiple regression model can predict a dependent variable Y, from multiple X-variables. A regression model assumes that the relationship between Y and X_1, X_2, … X_n has the following form as shown in Model (0) below; (Winston, 2007).

Y = Constant + $B_1X_1 + B_2X_2 + \ldots B_nX_n$ ***Model (0)***

Using Microsoft Office Excel 2010 Data Analysis Tools, the constant and coefficients (B_1, B_2, … B_n) are calculated to give the most accurate prediction of Y from the X-variables. Orlov (1996), gives a detailed description on how to utilise the Microsoft Excel regression tool, and this was used as a guide in the statistical work.

In the practical sense, there is a general acceptance that the ore blend has a direct impact on thickening performance. Two models were fitted to investigate the variation of flocculent consumption with rock types in the plant feed blend. In the first model (Model (1)); the assumption made is that flocculent consumption can be predicted from the percentage of ore types in the plant feed. Model (1) investigated the relationship between flocculent consumption (dependent Y-variable), and the ore mix.

Flocculent Consumption = Constant + B_1(%Rock type)$_1$ + B_2(%Rock type)$_2$ + … B_n(%Rock type)$_n$

Model (1)

In the second model (Model (2)); the assumption made is that the rock types influence variation of flocculent consumption from the design consumption rate of 28g/tonne. Model (2) investigated the relationship between variance of flocculent consumption (dependent Y-variable), and the ore mix. The constant in the model is set to zero.

%Variance from design consumption rate = B_1(%Rock type)$_1$ + B_2(%Rock type)$_2$ + … B_n(%Rock type)$_n$

Model (2)

The data for the investigation is tabulated in Appendix 3. Line fit plots were determined for seven major rock types, and stockpile material implying a total of eight X-variables. The line fit results were consistent for flocculent consumption and variance from design consumption rate, showing ores types NPK_GG and SVK_M to have greater impact on flocculent consumption. The line fit plots are shown below.

Figure 30 is a line fit plot showing correlation between NPK_GG and flocculent consumption. The R-squared value confirms a correlation between %NPK_GG in mine mix and flocculent consumption.

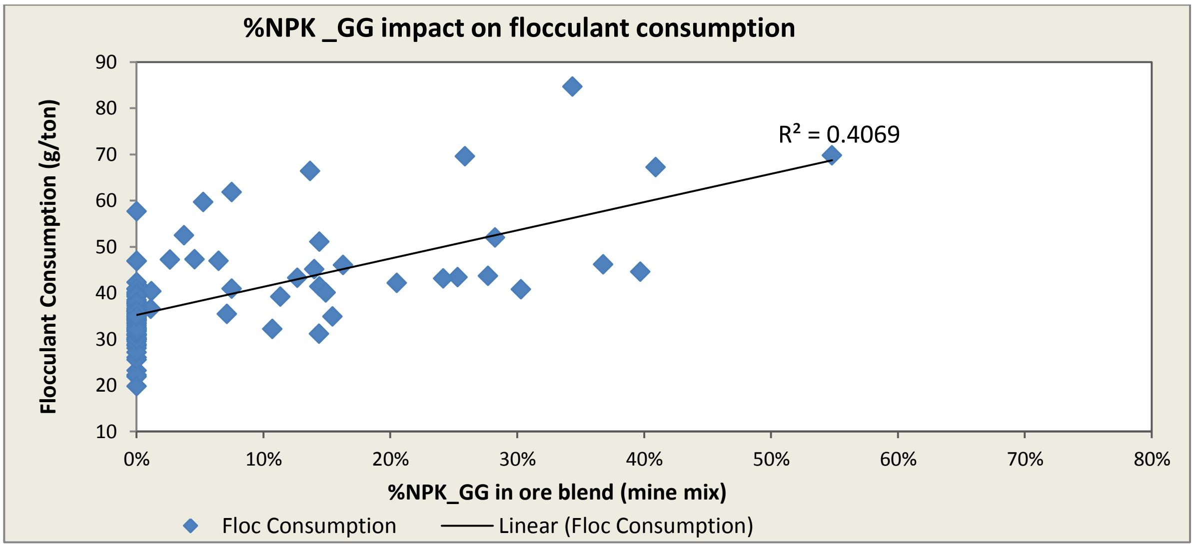


Figure 30: Impact of NPK_GG on flocculant consumption (settling challenges)

Figure 31 is a line fit plot showing correlation between SVK_M and flocculent consumption. The R-squared value is 0.24, indicating a weak but positive correlation. The correlation, however, is much stronger than the remaining rock types in comparison. The line fit plots for the remaining rock types are shown in Appendix 7.

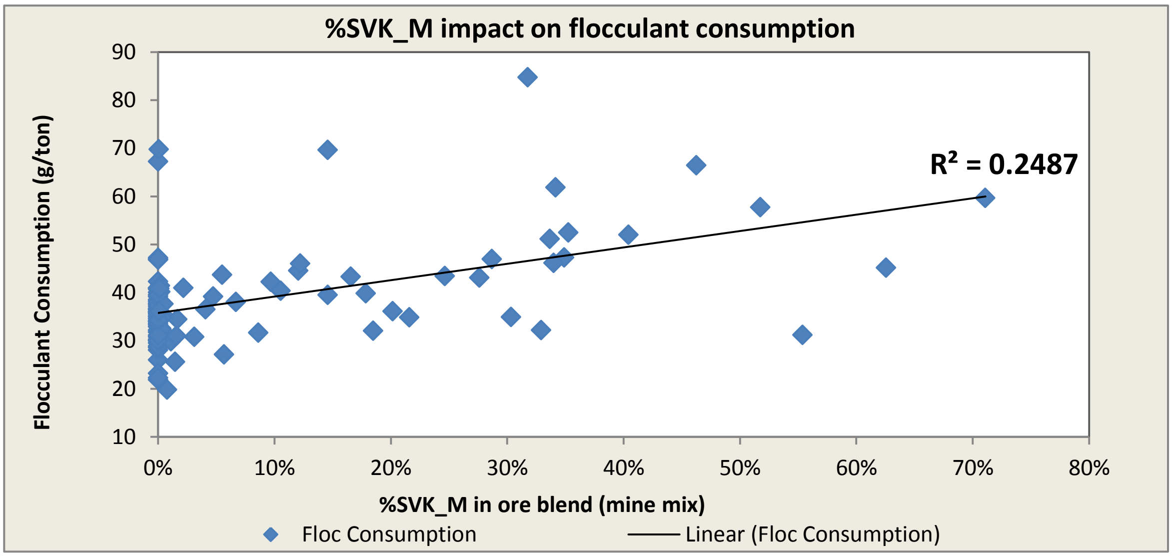


Figure 31: Impact of SVK_M on flocculant consumption (settling challenges)

The summary output results for the regression tests are shown in Appendix 6. Model 1 was deduced to be as follows:

Flocculent Consumption = 682 - 639·%NPKB - 596·%NPK_GG - 648·%NPK - 652·SVK_U - 628·%SVK_M-638·%A3T- 646 - %BB-650 - %C3 Stockpile

To investigate the impact of each rock-type, percent rock-types are individually set to contribute 100% of the plant feed, with the rest set at 0%. Model (1) predicts the following flocculent consumptions if each rock type is to be 100% of plant feed.

Table 9: Predicted flocculent consumption from model (1) at 100% of each rock type

Rock type @ 100%	NPKB	NPK _GG	NPK	SVK_U	SVK_M	A3T	BB	C3 Stockpile
Predicted consumption (g/ton)	43	86	34	30	54	44	36	32

Table 9 results show that the highest flocculent consumption is experienced with NPK_GG (86g/ton when 100%NPK_GG is processed), followed by SVK_M (54g/ton when 100% SVK_M is processed). This proves NPK_GG and SVK_M to be the most challenging rock types when it comes to settling slurries.

Model 2 coefficients were deduced to be as follows:

%Variance from design rate = 0.67%NPKB + 1.92%NPK_GG + 0.18%NPK + 0.06SVK_U + 0.94%SVK_M + 0.61%A3T + 0.031 %BB + 0.15 %C3 Stockpile

Table 10 shows the regression coefficients from each rock type.

Table 10: Regression coefficients (with rock types representing x-variables)

y-variable	*x-variables*							
Flocculant consumption	NPKB	NPK_GG	NPK	SVK_U	SVK_M	A3T	BB	Stockpile
Coefficient	0.67	1.92	0.18	0.06	0.94	0.61	0.31	0.15

The summary output tables give the p-value for each of the explanatory variables (rock types). The statistical significance of each rock type is represented by the corresponding p-value. P-values are the probabilities that the coefficients are *not* statistically significant. (Winston, 2007). The usual significance level is 5%, and if the p-value is less than 0.05 then it is more statistically significant. (Orlov (1996), Winston (2007)). The results in Appendix 7 show that the three most statistically significant rock types to the outcome of the model are:

NPK_GG **p-value of 1.94E-11**

SVK_M **p-value of 7.30E-07**

A3T **p-value of 0.03**

Orlov (1996), and Winston (2007) explain that the lower the p-value, the more relevant is the corresponding variable to the outcome of the model and practical relevance inferences can be drawn from the results.

The practical conclusion is that NPK_GG and SVK_M have the greatest impact on the variance of actual flocculent consumption from the design consumption rate, followed by A3T.

To investigate the impact of each rock-type, percent rock-types are individually set to contribute 100% of the plant feed, with the rest set at 0% (Orlov, 1996). Model (2) predicts the following results if each rock type is to be 100% of plant feed.

Table 11: Predicted flocculent consumption from model (2) at 100% of each rock type

	x-variables							
	NPKB	NPK_GG	NPK	SVK_U	SVK_M	A3T	BB	Stockpile
%Variance from design rate	67%	192%	18%	6%	94%	61%	31%	15%
Variance from design rate (g/ton)	19	54	5	2	26	17	9	4
Predicted consumption rate (Variance + 28)g/ton	47	82	33	30	54	45	37	32

Table 11 and Model (2) results show NPK_GG and SVK_M to be the most challenging rock types on settling.

The results show the strength of each rock type to influence slurry settling performance and flocculent consumption. The above results can be a basis for development of a new blending matrix to counter the impact of NPK_GG and SVK_M, which are set to be more abundant rock types in the ore mix in the future.

5.6 Financial implications and flexibility

The average notional price for a carat for the Orapa 2 stream was about USD74.54 according to the 2010 diamond price book for Debswana. Assuming this is the prevailing price, equation (2) can be used to estimate the recovered revenue in dollar per ton processed.

At an overall grade of 84 carats per hundred tonnes, liberation factor of 86% and a recovery efficiency of 99%, equation (3) estimates a recovery of $53.31 for every ton of ore treated. To determine the revenue per hour, the result is multiplied by the rate of treatment, 1400tonnes per hour. Thus, the revenue recovery rate for Orapa 2 is approximately USD74.6K$/hour.

From the analysis in section 5.5, SVK_M and NPK_GG introduced a constraint at the slurry handling section that leads to delays that range from 406minutes to 765minutes. In terms of the financial impact, this translates to USD6.06 million to USD11.4 million per annum, about 6 – 10% of annual planned revenue.

The prevailing price of flocculant is approximately $3.00/ kg (free on board). Based on this price, increases in flocculant consumption translate to $80 000 - $100 000 per annum.

Considering the life of mine plan, the percentage of NPK_GG and SVK_M are expected to increase in the mine mix as shown in figure 33.

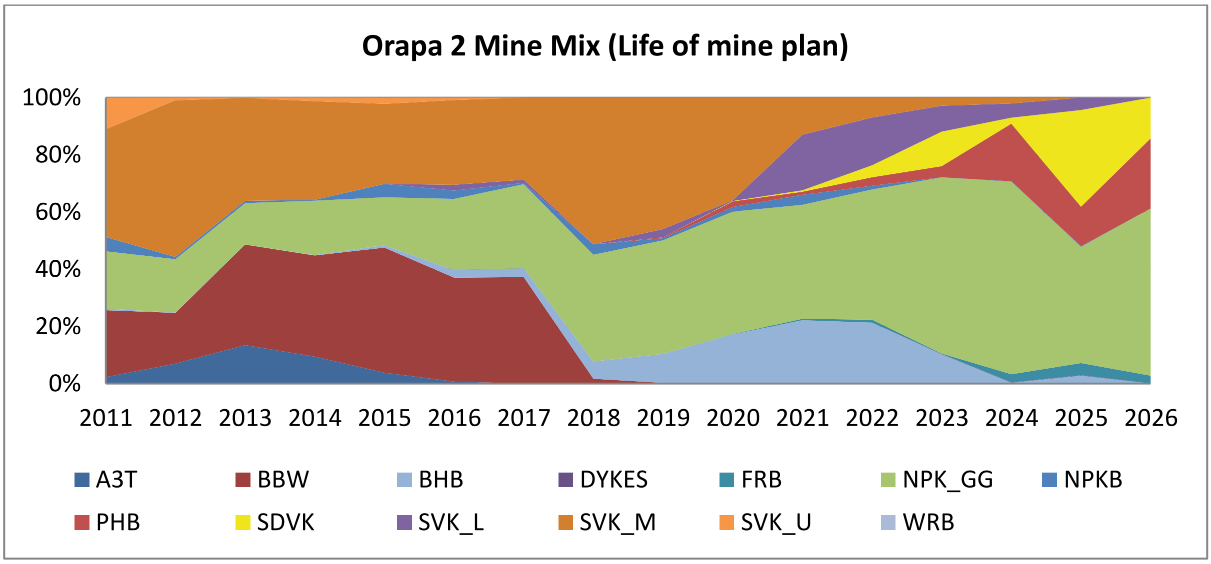


Figure 32: Anticipated mine mix in life of mine plan

The two rock types are set to be dominant in the mine mix in the coming years, and this will exacerbate the slurry settling challenges and lead to further losses in revenue and operating costs.

Beyond 2014, the basalt breccias will be exhausted limiting blending capabilities as a flexibility option. From 2017 onwards, the mine mix will consists of more than 80% combined SVK_M and NPK_GG. This is set to further worsen the current challenges unless an intervention is made to build in flexibility.

The extent of this risk can be illustrated by forecasting the operational delays and flocculent consumption. From the historical plant performance, projections can be made to predict the impact on throughput and operating costs based on the blend in the life of mine plan. Figure 34 illustrates the projections of settling delays as a percentage of the total delays, as well as flocculent consumption.

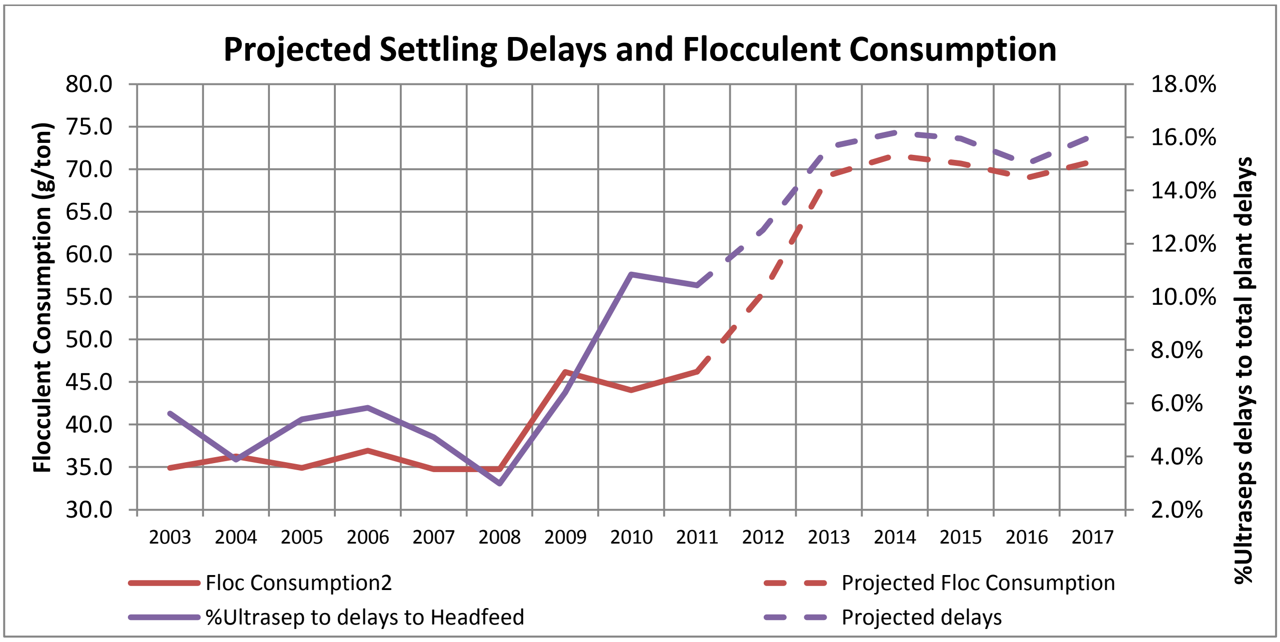


Figure 33: Forecast of the impact of the 5year plan blend on settling delays and flocculent consumption

From the calculations above based on equation (2) and equation (3), the blend performance has shown a revenue loss of 6 – 10% against the plan since 2008. The forecasting in Figure 34 above shows increases in SVK_M and NPK_GG in the 5 year plan are likely increase the ultraseps' delays contribution to the overall plant delays from the 2011/ 2012 levels of just above 10% to average about 15.5% from 2013. This is likely to proportionally increase the revenue losses due to ultraseps delays to 9% - 15%.
There is a risk of approximately 9% - 15% reduction on the recovered revenue on the 5 year and life-of-mine plans due to the throughput risk resulting from the planned ore blend. The flocculent consumption is expected to double, using to the 2008 levels as a baseline, (from 35g/ton to 70g/ton) resulting in 100% increase on costs of the consumable.

5.7 Risk Considerations

The Orapa 2 plant design was based on the Matthew Field 1996/97 Geological model. There was a general acceptance that this geological model had some shortcomings, that is why the OREP programme was initiated in 2005 with the aim of increasing the level of confidence of the A/K1 resource from inferred to indicated (between 265mbgl to 600mbgl) in terms of geology, volume, density, grade and revenue. Orapa is mining an indicated resource, where the grade or geological continuity are assumed and not confirmed (The SAMREC Code, 2007). The limited confidence in the geological continuity implies higher levels of uncertainty, increasing the probability of resource risk.

The unexpected facies in the blend had an impact on the plant throughput, by introducing a constraint in the slurry handling section of the plant. The consequence is the financial impact in terms of recovered revenue as discussed in section 5.6 above. Factors that influenced or exacerbated the impact are that the thickeners used were an untested technology, limited blending capability and unavailability of tactical plans to deal with the risk.

Taking into consideration the risk elements (probability and impact), the factors that contributed to the geometallurgical risk of the Orapa A/K1 deposit are shown in Table 12 below.

Table 12: Geometallurgical risk factors for Orapa A/K1 deposit

Probability	Consequence/ Impact
Limited confidence on the resource due to sparse sampling, estimation methods (kriging). This led to higher uncertainty on the resource increasing the probability of risk as metallurgical design decisions were based on the available ore body information.	Exacerbated by lack of flexibility as some technology that is still new in the industry was used, with little known capability There was no feedback process of the production information into the planning process, to better plan and mitigate against the impact of the previously unknown geology

5.8 Discussion

The Orapa A/K 1 deposit makes a viable case for implementation of a geometallurgical program. The results above demonstrate existence of geometallurgical risk due to a thickening constraint as a result of previously untreated rock types in the ore blend that form non-settling slurries.

The analysis in sections 5.2, 5.3, and 5.5 shows that SVK_M and NPK_GG were encountered in late 2008, and there was a subsequent increase in head feed delays showing an increase in stoppages caused by sliming ultraseps, or non-settling slurries. Before encountering SVK_M and NPK_GG rock types in the deposit, slurry settling was responsible for about 4% of the total delays per year on average. After introduction of these rock types into the mine mix, slurry settling was now responsible for over 10% of the total head-feed delays.

The ore dressing studies done for the Orapa A/K1 deposit confirmed potential challenges with slurry settling. The two studies showed high smectite content in SVK and NPK, but could not identify sub-classification of these facies into domains with different behaviour in the treatment plant. In section 3.2.3, it was demonstrated that ESP in excess of 15 percent indicates a potential of generating non-settling settling slurries. Both studies showed high ESP values for the major rock types in the A/K1 deposit, an indication of potential slurry settling challenges. The assumption that this could be dealt with by blending it with other rock types was made without a geometallurgical model. The 2002 ODS clearly stated the potential risk to slurry settling and high flocculant consumption in its conclusion. No action was taken to alleviate this risk. The second study, 2009 ODS, down played this risk, making an erroneous assumption that the process water quality had enough cations to aid settling of the high smectite ores. The availability of ex-stockpile lower grade material and some basalt breccias made it possible to blend the SVK_M and NPK_GG to acceptable mixes that would reduce the impact of the rock types. This could have resulted in the inability to fully appreciate the limitations of the ultraseps thickeners to deal with ore variability. An alternative approach might have been an investment in pH control capabilities for the process, to mitigate the impact of the ESP, as well as incorporating the ESP variable in the block model estimates used in mine planning.

The presence of SVK_M and NPK_GG, as sub groups of the major geological zones i.e. volcanoclastic and pyroclastic kimberlites, is a clear indication of the existence of geometallurgical domains. Though these rock types are similar to other sub-classifications in the major geological zones, they exhibit different behaviour in the process plant. The settling challenges of these kimberlites are similar to the challenges encountered at Voorspoed, an operation treating similar rock types and discussed earlier as an illustration of strategic geometallurgy.

Orapa exhibits a case where a plant was designed based on geological domains, without anticipation of sub-classification that occurred with depth to reveal rock types with different geometallurgical properties. A design decision to install ultraseps instead of conventional thickeners has proved detrimental as no flexibility was built in at this stage. Ultraseps were a new untested technology, and Orapa 2 was the first ever high tonnage application of such thickeners. Since the ultraseps thickeners have no moving parts, there is no shearing force on the settling slurries, making the units highly sensitive and unable to cope with high clay content slurries. Selection of ultraseps as thickeners of choice was a risky decision. This was a new technology that had not been tested, though test-work had proved that the ultraseps would work for "generic" Orapa kimberlites, no thoughts were put into building flexibility in case the characteristics of the material mined from the orebody were to be different to that that was expected. The risk of the non-settling slurries was hinted by the 2002 ODS, but this risk was poorly communicated to the business due to inadequate structures that would support such requirements. A geometallurgical function would have explored this risk and adequate control or mitigation measures would have been instituted at the time.

Since the operation is failing to deal with the demonstrated risk, this validates a need for a flexibility option that can accommodate this risk and remove the constraint. This is despite the redundancy built in the process at design, eight ultraseps and an additional paste thickener that was later installed in 2006 on the justification of better water recovery efficiency.

Beyond 2017, the situation is bound to worsen. The two rock types will make up more than 60% of the mine mix. In terms of blending, the capability has been exhausted since the volcanoclastic kimberlites and pyroclastic kimberlites were being blended with basalt breccias. The increases in the occurrence of these rock types indicates diminishing blending ability, a technique that has enabled controlled amounts of the two rock types to be depleted and processed in the plant in the previous few years. The mine mix will consist of about 10% breccias, which is not adequate for effective blending ratios.

A geometallurgical program could have identified varying metallurgical responses in the geological domains and existence of geological sub-classifications and their impacts would have been noted in earlier studies of the resource. The geological domains could have been split into geometallurgical domains and contingency would have been built in to deal with any expected changes in the resource. Appreciation of the risk would have signalled commencement of a planning process to explore the best possible options available to alleviate the risk.

6 CONCLUSION AND RECOMMENDATIONS

The Orapa operation clearly demonstrates the existence of geometallurgical risk. This study highlights the need for adoption of geometallurgy as a method of improving resource understanding and identifying resource parameters that drive value. Orapa 2 is losing over 10% of its annual revenues due to the presence of rock types that were previously not modelled in the resource and future cash-flows are at an even greater risk due to an indicated increase of these rock types in the plant feed mix.

A logical next step is the development of a geometallurgical program and a geometallurgical model for the A/K1 kimberlite orebody, however, this can be a time consuming process. A tactical approach is needed for immediate action. Important lessons can be drawn from the Voorspoed project on how to deal with geometallurgical risk on a brownfield operation. Most of the Voorspoed challenges were mitigated by management of the observed changes in the deposit in terms of levels of dilution and communication of the contact deviations. There is need for Orapa to develop an urgent geometallurgical task team comprising competent mine planners, geologists and metallurgists. The task team will work within the scope of short term planning to develop tactical solutions that include blend optimisations. This will improve cross disciplinary communication and minimize impact of delays. As a brownfield operation, a lot of data exists that can be used to update the existing geological model with geometallurgical attributes. This approach can be adopted by the geometallurgical task team, and problematic areas in the pit can be identified as well as those that are not as challenging, and this can inform tactical decisions on blending and prioritization of processing. This is similar to the approach that was employed at Voorspoed where data was collected in the first months of production and was used to update the geological model to develop a functional model that could be used in mine planning. The Kemi Ferrochrome Mine employs this strategy where geometallurgical information attained during production is continually updated in the model.

The strategic solution to develop a geometallurgical model takes some time and may not immediately address the already on-going problem. However, this is critical for the development of an appropriate model that can be used in production, strategic planning and also provide a crucial knowledge of the resource. Work on the model can commence whilst the geometallurgical task team is engaged on the tactical management of the risk. A separate team, with experience and competence in ore dressing studies, can be engaged for the development of a geometallurgical model. Significant amounts of information already exist from previous ore dressing studies and some may be extracted for use in a properly structured geometallurgical program. Since geological models already exist showing the ore types and lithology, a geometallurgical program can be instituted. The following actions should be carried out to develop a successful geometallurgical program:

- Assessment of the metallurgical validity of the geological ore-type definitions or geological domains and, where necessary, developing geometallurgical domains. Data from previous ore dressing studies can be utilized in this step;
- Conduct large diameter drilling to collect samples that are representative of the facies for metallurgical test-work.
- Collect large size samples for pilot plant trials, with a bias towards thickening trials as this has shown to be the major challenge. For a comprehensive test program, tonnage size samples are to be sourced for the major ore types especially SVK_M and NPK_GG, and the recently modelled MVK that is yet to be treated in the plant;
- Assess the primary geometallurgical variables and develop appropriate mathematical relationships across the geological database. Consideration of additivity or non-additivity of the variables is paramount for the estimation of important metallurgical parameters;
- Perform tests on ore blends representative of the life of mine feed at various time periods;
- Plant simulation;
- Develop a 3D geometallurgical model; and
- Calibration of the models by benchmarking with similar operations such as Voorspoed Diamond Mine.

After the development of the geometallurgical model, The Kemi Chromite and Ferrochrome Mine approach of continually updating the model needs to be adopted to improve its reliability. Kemi Mine is a bench mark operation in terms of functional data acquisition, cross disciplinary communication and resultant superior productivity levels.

To minimize expenditure associated with the development of a geometallurgical program, the work can be coopted into the on-going Orapa Resource Extension Project. Sampling requirements can be redefined so that appropriated sampling is conducted to meet the geometallurgical requirements.

In order to remove the plant throughput constraint, capital injection is required. Any proposals or interventions to optimize performance of thickeners such as retrofitting the ultraseps and the paste thickener with shear enhancement mechanisms should be rigorously tested. The Cawse Nickel Project has also proven the risks associated with adopting new technology with challenging ores. Orapa 2 has this far selected two thickening mechanisms, ultraseps and paste thickening, both of which were emerging technologies that have performed well below expectation. Any proposal should thus incorporate a tried and tested option, such as the conventional thickeners with a moving rake. An option that includes a conventional thickener will allow for flexibility and avoid prolonged capital expenditure due to implementation of untried technologies.

There is geometallurgical risk at Orapa's A/K1 deposit that is likely to impact future cash-flows by a minimum of 10%. There is currently no flexibility to deal with the risk and capital injection is required to optimize the current thickening process and remove the constraint. The operation is mining an indicated resource which is quite common with diamond deposits, but has significant levels of geometric and compositional uncertainty and variability. The indicated resource is likely to be depleted by 2016. Developing a geometallurgical program will enhance the understanding of the resource, and lead to tactical and strategic plans to identify and deal with the risk. A geometallurgical program will provide requisite information about the resource and assist in decision making on any interventions to deal with geological uncertainty or future metallurgical improvement initiatives.

The risk also needs to be managed at the tactical level by improving interactions between mine planning, geology and metallurgy. An informed approach is required for a metallurgical solution to alleviate or ideally remove the constraint (i.e. thickeners). This could potentially be achieved through better short term scheduling that is based not only on maximizing grade, but by incorporating rock properties in the plan and blending to achieve an optimal process performance, which in turn will improve financial returns..

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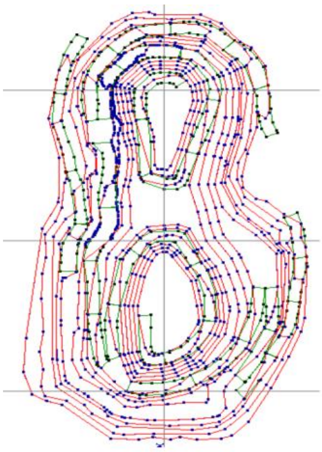
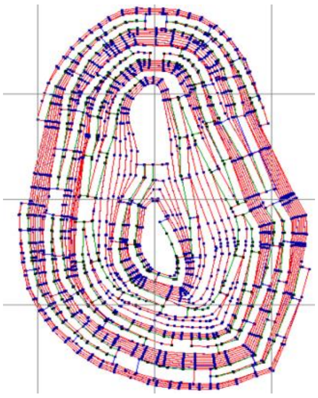
APPENDIX 1: A/K1 Mine Design



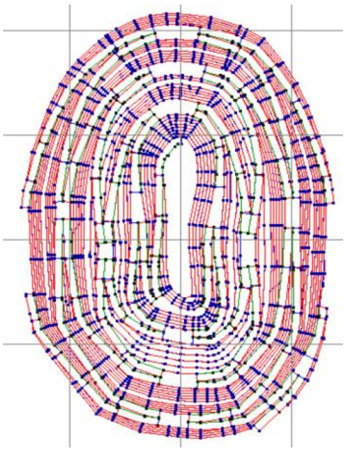
Orapa Mine A/K1 CUT 1 DESIGN	
Designed by: A Ditlhakeng	Approved by: Geotech
Date: 1970s	Version 1
A/K1 CUT 1	
Ramp Width – 35m	Gradient – 8%
Two lane Ramp	Berm height – 15m
Berm width – variable	Pit bottom – 710mbgl

Figure 34: A/K1 Cut 1 design



Orapa Mine A/K1 CUT 2 DESIGN	
Designed by: A Ditlhakeng	Approved by: Geotech
Date: 1970s	Version 1
A/K1 CUT 2	
Ramp Width – 35m	Gradient – 8%
Two lane Ramp	Berm height – 15m
Berm width – variable	Pit bottom – 530mbgl

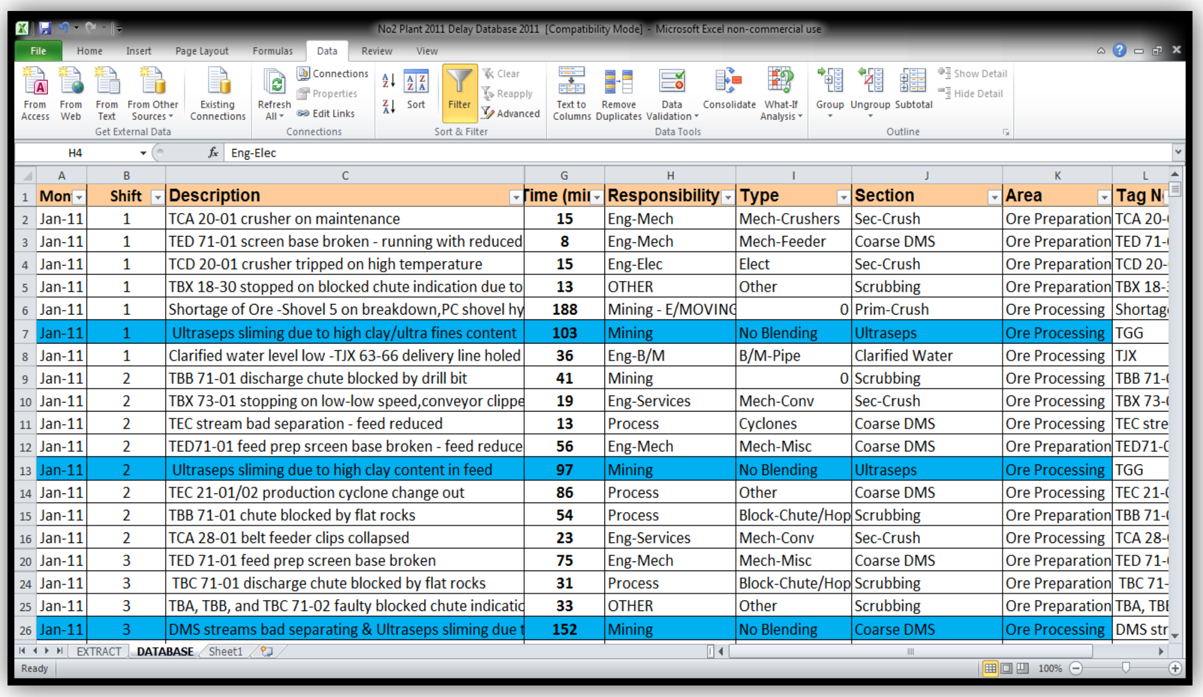
Figure 35: A/K1 Cut 2 design



Orapa Mine A/K1 CUT 3 DESIGN	
Designed by: A Ditlhakeng	Approved by: Geotech
Date: 1990s	Version 1
A/K1 CUT 3	
Ramp Width – 35m	Gradient – 8%
Two lane Ramp	Berm height – 15m
Berm width – variable	Pit bottom – 485mbgl

Figure 36: A/K1 Cut 3 design

APPENDIX 2: Illustration of the Orapa 2 delays database



	Mon	Shift	Description	Time (mi	Responsibility	Type	Section	Area	Tag N
2	Jan-11	1	TCA 20-01 crusher on maintenance	15	Eng-Mech	Mech-Crushers	Sec-Crush	Ore Preparation	TCA 20-
3	Jan-11	1	TED 71-01 screen base broken - running with reduced	8	Eng-Mech	Mech-Feeder	Coarse DMS	Ore Preparation	TED 71-
4	Jan-11	1	TCD 20-01 crusher tripped on high temperature	15	Eng-Elec	Elect	Sec-Crush	Ore Preparation	TCD 20-
5	Jan-11	1	TBX 18-30 stopped on blocked chute indication due to	13	OTHER	Other	Scrubbing	Ore Preparation	TBX 18-
6	Jan-11	1	Shortage of Ore -Shovel 5 on breakdown,PC shovel hy	188	Mining - E/MOVING	0	Prim-Crush	Ore Processing	Shortag
7	Jan-11	1	Ultraseps sliming due to high clay/ultra fines content	103	Mining	No Blending	Ultraseps	Ore Processing	TGG
8	Jan-11	1	Clarified water level low -TJX 63-66 delivery line holed	36	Eng-B/M	B/M-Pipe	Clarified Water	Ore Processing	TJX
9	Jan-11	2	TBB 71-01 discharge chute blocked by drill bit	41	Mining	0	Scrubbing	Ore Processing	TBB 71-
10	Jan-11	2	TBX 73-01 stopping on low-low speed,conveyor clippe	19	Eng-Services	Mech-Conv	Sec-Crush	Ore Processing	TBX 73-
11	Jan-11	2	TEC stream bad separation - feed reduced	13	Process	Cyclones	Coarse DMS	Ore Processing	TEC stre
12	Jan-11	2	TED71-01 feed prep srceen base broken - feed reduce	56	Eng-Mech	Mech-Misc	Coarse DMS	Ore Preparation	TED71-
13	Jan-11	2	Ultraseps sliming due to high clay content in feed	97	Mining	No Blending	Ultraseps	Ore Processing	TGG
14	Jan-11	2	TEC 21-01/02 production cyclone change out	86	Process	Other	Coarse DMS	Ore Processing	TEC 21-
15	Jan-11	2	TBB 71-01 chute blocked by flat rocks	54	Process	Block-Chute/Hop	Scrubbing	Ore Preparation	TBB 71-
16	Jan-11	2	TCA 28-01 belt feeder clips collapsed	23	Eng-Services	Mech-Conv	Sec-Crush	Ore Processing	TCA 28-
20	Jan-11	3	TED 71-01 feed prep screen base broken	75	Eng-Mech	Mech-Misc	Coarse DMS	Ore Preparation	TED 71-
24	Jan-11	3	TBC 71-01 discharge chute blocked by flat rocks	31	Process	Block-Chute/Hop	Scrubbing	Ore Preparation	TBC 71-
25	Jan-11	3	TBA, TBB, and TBC 71-02 faulty blocked chute indicati	33	OTHER	Other	Scrubbing	Ore Preparation	TBA, TB
26	Jan-11	3	DMS streams bad separating & Ultraseps sliming due t	152	Mining	No Blending	Coarse DMS	Ore Processing	DMS str

Figure 37: Illustration of the delays database layout

APPENDIX 3: Mine Mix and Flocculent Consumption (2002 to 2011)

Date	Floc Consumption (g/ton)	NPKB	NPK _GG	NPK	SVK_U	SVK_M	A3T	BB	C3 Stockpile
Mar-02	30.25	0.00%	0.00%	0.68%	36.19%	0.00%	12.07%	12.13%	38.94%
Apr-02	26.02	0.03%	0.00%	13.94%	24.16%	0.00%	9.10%	26.04%	26.73%
May-02	21.88	0.00%	0.00%	1.58%	36.50%	0.00%	5.06%	28.06%	28.80%
Jun-02	23.20	0.00%	0.00%	1.53%	44.12%	0.00%	20.85%	33.50%	0.00%
Jul-02	22.27	0.00%	0.00%	4.94%	39.47%	0.00%	14.97%	23.47%	17.16%
Aug-02	28.06	0.01%	0.00%	1.34%	35.74%	0.00%	5.67%	36.81%	20.44%
Sep-02	28.61	0.00%	0.00%	14.67%	29.15%	0.00%	3.76%	32.63%	19.80%
Oct-02	29.51	0.02%	0.00%	0.17%	42.60%	0.00%	0.44%	28.26%	28.50%
Nov-02	40.78	0.42%	0.00%	16.05%	44.47%	0.00%	3.49%	9.94%	25.63%
Dec-02	52.51	0.00%	3.75%	0.00%	0.24%	35.25%	18.68%	12.97%	29.12%
Jan-03	40.67	0.00%	0.44%	9.22%	30.51%	0.03%	2.91%	38.20%	18.70%
Feb-03	40.96	0.04%	0.00%	1.72%	18.22%	0.00%	27.07%	21.69%	31.27%
Mar-03	30.25	1.73%	0.00%	25.63%	33.52%	0.06%	3.16%	16.67%	19.24%
Apr-03	40.20	1.45%	0.00%	14.38%	33.19%	0.20%	6.38%	29.83%	14.57%
May-03	36.00	0.61%	0.00%	8.11%	73.03%	0.00%	1.46%	8.16%	8.63%
Jun-03	35.91	0.04%	0.00%	0.31%	64.65%	0.00%	1.39%	11.82%	21.78%
Jul-03	42.29	0.00%	0.00%	0.10%	61.93%	0.00%	5.50%	9.10%	23.36%
Aug-03	36.00	0.00%	0.00%	0.00%	44.62%	0.00%	5.19%	25.12%	25.06%
Sep-03	36.00	0.26%	0.00%	0.71%	45.75%	0.04%	8.88%	23.70%	20.66%
Oct-03	32.52	0.00%	0.00%	19.33%	35.98%	0.29%	7.41%	10.66%	26.33%
Nov-03	30.00	2.90%	0.00%	21.90%	36.54%	0.00%	1.11%	27.29%	10.26%
Dec-03	28.89	0.79%	0.00%	20.47%	43.45%	0.00%	7.07%	10.36%	17.85%
Jan-04	29.90	0.06%	0.00%	2.91%	54.01%	0.06%	2.17%	11.89%	28.90%
Feb-04	30.81	2.31%	0.00%	9.08%	41.88%	0.00%	2.87%	35.19%	8.67%
Mar-04	38.51	0.61%	0.00%	1.56%	10.34%	0.00%	1.40%	45.61%	40.48%
Apr-04	36.37	0.00%	0.00%	0.00%	44.57%	0.00%	0.41%	28.16%	26.86%
May-04	30.83	2.74%	0.00%	5.20%	18.28%	3.11%	1.24%	54.10%	15.34%
Jun-04	29.84	0.34%	0.00%	17.54%	25.04%	1.12%	0.83%	41.69%	13.44%
Jul-04	25.59	0.00%	0.00%	0.32%	12.12%	1.47%	3.69%	31.73%	50.68%
Aug-04	27.13	0.00%	0.00%	0.51%	52.40%	5.67%	0.44%	31.01%	9.97%
Sep-04	59.69	0.00%	5.24%	0.00%	0.56%	71.09%	4.50%	11.37%	7.24%
Oct-04	19.82	0.00%	0.00%	2.83%	57.41%	0.79%	3.77%	12.39%	22.82%
Nov-04	43.30	0.00%	12.66%	0.00%	44.98%	16.56%	0.43%	19.07%	6.31%
Dec-04	36.55	0.00%	1.12%	7.39%	49.26%	4.09%	10.32%	25.08%	2.74%
Jan-05	66.44	0.00%	13.67%	0.00%	1.61%	46.25%	17.10%	10.47%	10.91%
Feb-05	47.21	1.54%	2.62%	0.00%	15.17%	0.00%	4.72%	30.26%	45.70%
Mar-05	41.45	1.83%	14.40%	0.00%	16.18%	0.19%	2.75%	26.08%	38.57%
Apr-05	40.97	0.00%	7.50%	25.66%	33.41%	2.18%	1.80%	10.57%	18.87%
May-05	36.61	7.92%	0.00%	11.76%	46.10%	0.00%	13.73%	5.96%	14.52%
Jun-05	40.04	2.33%	0.00%	4.42%	45.64%	0.00%	6.40%	36.07%	5.14%
Jul-05	37.82	6.66%	0.00%	12.45%	35.32%	0.00%	9.74%	33.46%	2.37%
Aug-05	67.26	0.18%	40.90%	0.00%	4.49%	0.00%	9.33%	40.34%	4.76%
Sep-05	42.22	3.76%	20.50%	12.54%	0.00%	9.68%	11.17%	31.58%	10.76%
Oct-05	35.50	0.73%	7.12%	0.00%	24.77%	0.38%	11.45%	48.99%	6.56%
Nov-05	46.87	2.49%	0.00%	15.49%	12.28%	0.00%	12.84%	49.45%	7.45%
Dec-05	39.18	1.31%	0.00%	3.11%	14.11%	0.00%	21.85%	42.06%	17.55%
Jan-06	40.93	4.88%	0.00%	17.38%	26.59%	0.00%	14.94%	11.79%	24.41%
Feb-06	46.97	7.50%	6.46%	0.00%	24.99%	0.00%	23.33%	9.13%	28.59%
Mar-06	32.21	1.01%	0.00%	3.50%	25.13%	0.00%	33.38%	22.01%	14.96%
Apr-06	34.57	4.02%	0.00%	20.09%	26.78%	0.00%	19.85%	6.60%	22.66%
May-06	33.99	3.59%	0.00%	43.86%	16.59%	0.00%	18.19%	5.94%	11.83%
Jun-06	34.12	2.34%	0.00%	39.02%	11.55%	0.00%	7.28%	10.63%	29.19%
Jul-06	37.58	0.24%	0.00%	45.19%	16.14%	0.00%	14.37%	1.21%	22.85%
Aug-06	44.59	0.82%	39.70%	22.20%	0.00%	12.04%	4.81%	9.85%	12.57%
Sep-06	40.41	0.00%	1.18%	35.66%	22.61%	10.55%	7.63%	12.01%	10.36%
Oct-06	39.21	0.59%	11.33%	56.29%	11.20%	4.74%	1.88%	4.89%	9.08%

Nov-06	39.88	1.59%	0.00%	33.34%	0.00%	17.87%	15.84%	8.80%	22.55%
Dec-06	33.31	2.82%	0.00%	40.18%	18.67%	0.00%	3.82%	19.22%	15.29%
Jan-07	30.99	5.59%	0.00%	35.59%	20.84%	0.05%	10.99%	8.57%	18.37%
Feb-07	35.12	12.47%	0.00%	28.05%	10.09%	0.00%	7.49%	3.59%	38.31%
Mar-07	30.00	8.91%	0.00%	16.86%	39.54%	0.00%	7.47%	3.76%	23.46%
Apr-07	31.11	7.33%	0.00%	35.21%	27.80%	0.00%	4.39%	6.68%	18.59%
May-07	38.20	8.25%	0.00%	24.39%	29.21%	0.00%	3.98%	16.21%	17.96%
Jun-07	36.72	0.43%	0.00%	16.27%	50.41%	0.00%	0.86%	18.78%	13.25%
Jul-07	31.81	6.63%	0.00%	8.58%	53.57%	0.00%	2.60%	19.38%	9.24%
Aug-07	32.94	3.42%	0.00%	12.24%	53.98%	0.00%	6.24%	10.88%	13.26%
Sep-07	40.11	2.07%	14.92%	0.00%	24.62%	0.06%	14.33%	30.35%	13.66%
Oct-07	34.97	6.78%	0.00%	7.38%	58.96%	0.00%	1.20%	2.40%	23.28%
Nov-07	36.02	0.91%	0.00%	17.27%	30.37%	0.00%	8.86%	25.72%	16.87%
Dec-07	34.02	8.39%	0.00%	12.85%	33.82%	0.01%	7.70%	3.81%	33.41%
Jan-08	35.31	0.03%	0.00%	1.15%	60.14%	0.04%	1.10%	9.19%	28.36%
Feb-08	37.20	0.37%	0.00%	6.22%	49.67%	0.00%	7.23%	3.89%	32.61%
Mar-08	33.62	0.00%	0.00%	4.00%	63.85%	0.00%	8.14%	8.65%	15.36%
Apr-08	35.57	6.28%	0.00%	32.47%	27.96%	0.01%	6.28%	8.94%	18.07%
May-08	32.36	0.03%	0.00%	20.16%	38.63%	0.31%	11.45%	7.02%	22.40%
Jun-08	34.45	0.00%	0.00%	1.04%	58.18%	1.65%	6.92%	13.66%	18.55%
Jul-08	31.00	0.03%	0.00%	18.92%	37.76%	1.58%	7.97%	11.93%	21.80%
Aug-08	37.64	0.21%	0.00%	12.47%	50.57%	0.47%	2.92%	15.15%	18.22%
Sep-08	34.87	0.00%	0.00%	6.47%	55.12%	0.38%	8.75%	14.47%	14.81%
Oct-08	31.12	1.97%	0.00%	18.40%	12.74%	0.04%	22.33%	25.02%	19.49%
Nov-08	34.32	0.06%	0.00%	17.28%	18.61%	0.04%	22.48%	19.40%	22.11%
Dec-08	36.14	3.07%	0.00%	4.56%	21.15%	20.17%	17.50%	12.38%	21.16%
Jan-09	39.52	4.02%	0.00%	17.93%	23.15%	0.00%	25.53%	9.08%	20.29%
Aug-09	69.63	0.00%	25.87%	0.01%	0.00%	14.58%	13.22%	22.54%	23.77%
Sep-09	46.06	0.48%	16.26%	0.07%	0.00%	12.21%	19.79%	24.67%	26.52%
Oct-09	40.82	0.00%	30.28%	0.00%	0.00%	0.11%	37.12%	18.29%	14.21%
Nov-09	47.00	0.00%	0.00%	0.00%	0.00%	28.69%	34.46%	19.05%	17.80%
Dec-09	39.54	0.00%	0.00%	0.00%	0.00%	14.59%	40.25%	26.74%	18.42%
Jan-10	38.03	0.00%	0.00%	0.00%	0.00%	6.70%	34.00%	40.04%	19.25%
Feb-10	43.45	0.00%	25.30%	0.00%	0.00%	24.65%	2.32%	20.05%	27.68%
Mar-10	36.19	0.00%	0.00%	53.76%	0.00%	0.09%	22.20%	19.98%	3.97%
Apr-10	31.69	0.00%	0.00%	0.00%	0.00%	8.62%	23.30%	43.99%	24.09%
May-10	32.07	0.00%	0.00%	0.00%	0.00%	18.49%	18.34%	35.07%	28.09%
Jun-10	43.69	4.46%	27.68%	0.00%	10.10%	5.52%	7.86%	11.44%	32.95%
Jul-10	34.88	0.00%	0.00%	0.00%	0.00%	21.60%	9.27%	34.40%	34.73%
Aug-10	61.84	0.00%	7.50%	0.00%	0.00%	34.15%	12.40%	33.92%	12.04%
Sep-10	57.72	0.00%	0.00%	0.00%	0.00%	51.77%	1.75%	33.34%	13.14%
Jan-11	51.14	0.00%	14.40%	0.00%	10.66%	33.65%	5.87%	23.03%	12.39%
Feb-11	47.33	0.00%	4.57%	0.00%	20.04%	34.91%	0.51%	27.35%	12.62%
Mar-11	69.81	0.00%	54.80%	0.00%	0.00%	0.07%	19.37%	20.68%	5.08%
Apr-11	52.00	2.29%	28.25%	0.00%	2.27%	40.42%	0.90%	6.53%	19.35%
May-11	84.73	0.00%	34.34%	0.00%	0.00%	31.77%	1.75%	22.25%	9.90%
Jun-11	45.19	1.07%	14.00%	0.00%	4.44%	62.56%	2.37%	9.93%	5.64%
Jul-11	32.23	0.00%	10.69%	0.00%	32.73%	32.92%	0.04%	20.36%	3.26%
Aug-11	31.17	0.00%	14.37%	0.00%	10.43%	55.39%	0.11%	11.78%	7.91%
Sep-11	34.91	3.02%	15.43%	0.00%	31.46%	30.33%	0.02%	16.25%	3.50%
Nov-11	43.13	1.73%	24.16%	0.00%	1.35%	27.62%	0.47%	15.84%	28.82%
Dec-11	46.18	0.77%	36.76%	0.00%	0.50%	34.00%	0.90%	9.81%	17.27%

APPENDIX 4: Statistical test for settling delays (Hypothesis testing)

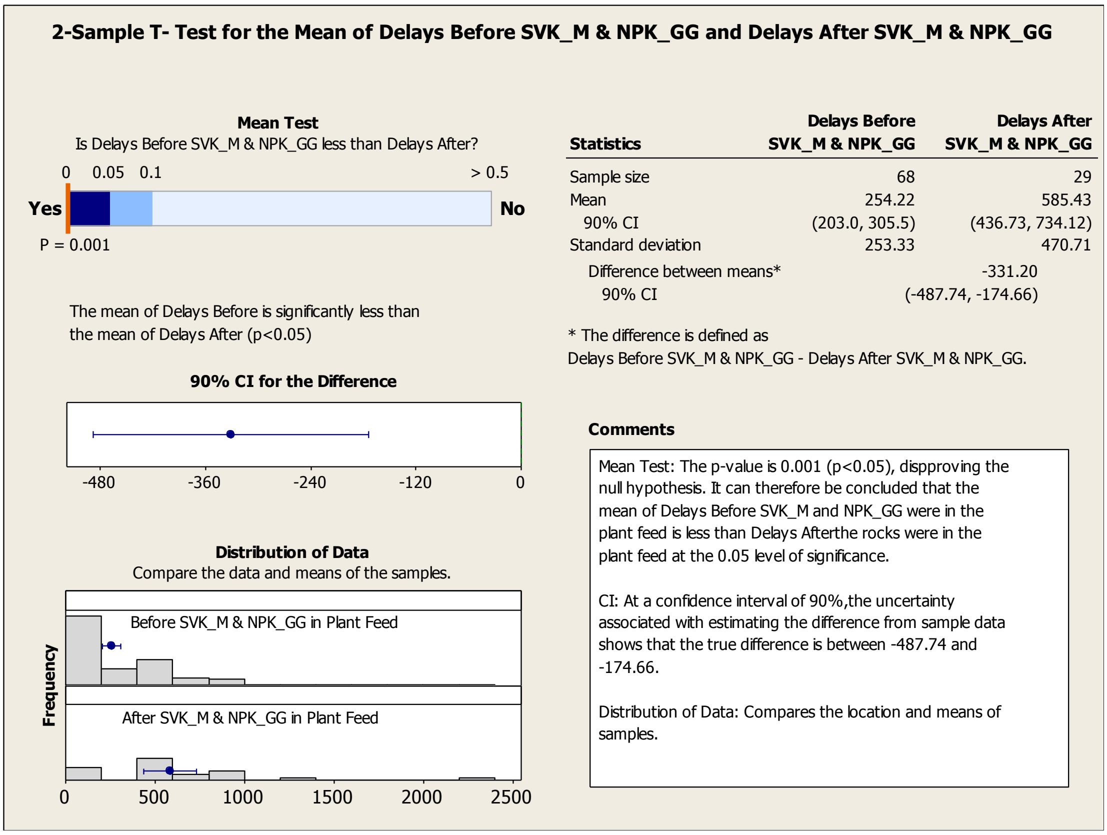


Figure 32: Hypothesis Test (2 Sample t-test) results on the slurry settling delays

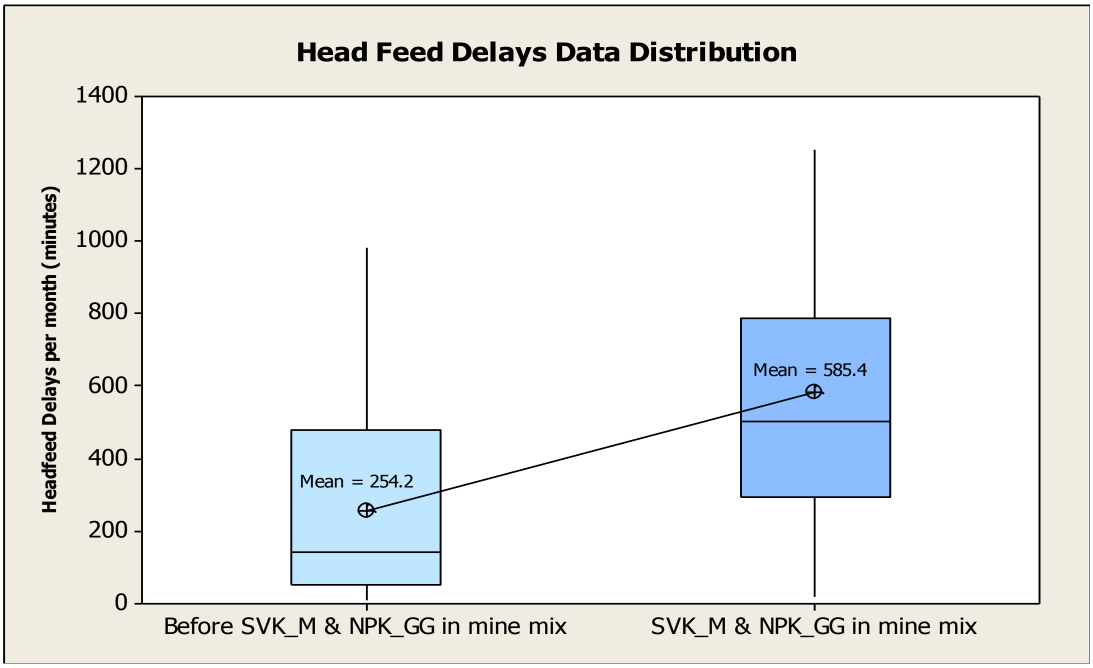


Figure 39: Head-feed delays distribution comparison (Before SVK_M & NPK_GG are in plant feed and after SVK_M & NPK_GG are in plant feed)

APPENDIX 5: Statistical test for Flocculant consumption (Hypothesis testing)

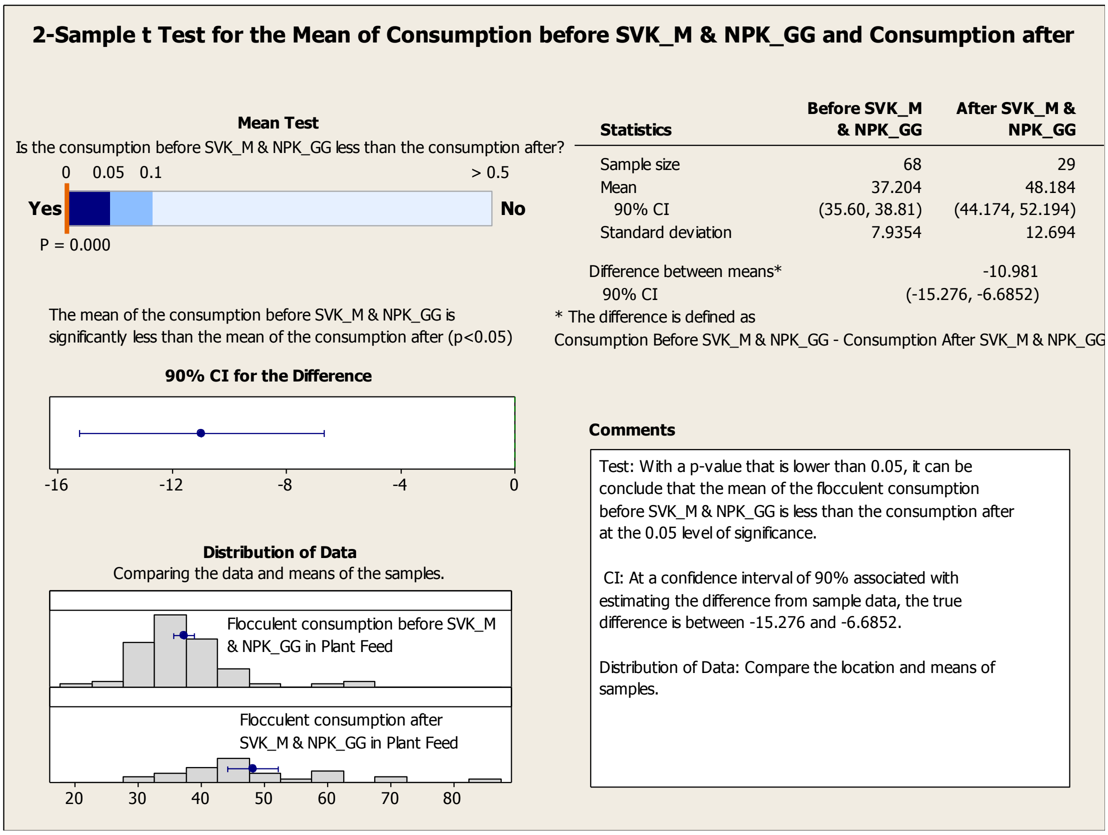


Figure 40: Flocculant consumption statistical summary (2003 to 2011)

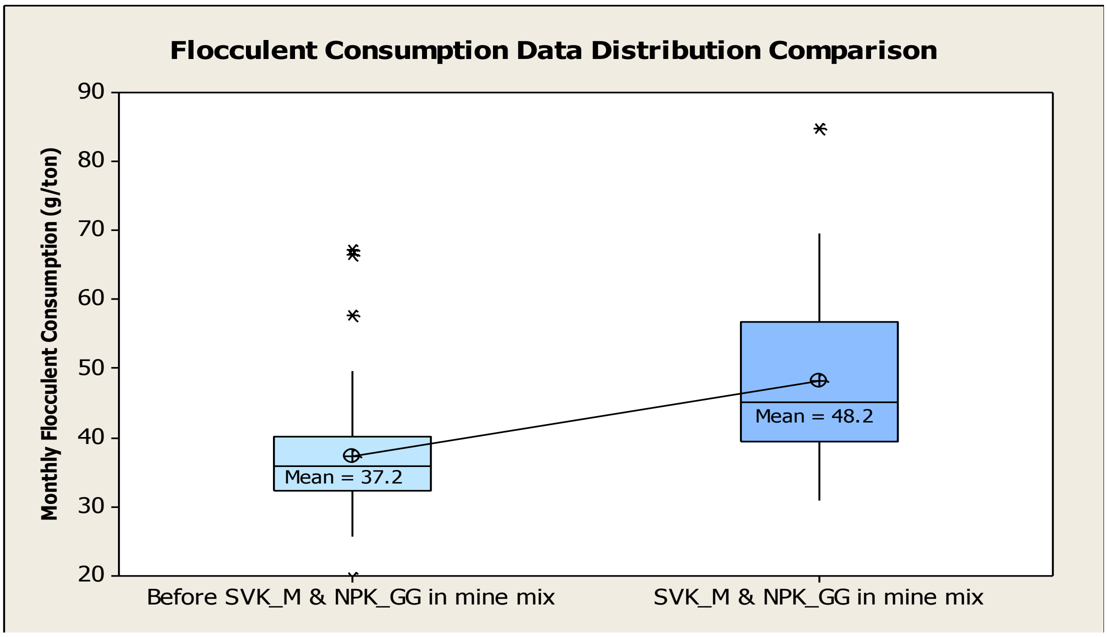


Figure 41: Flocculent consumption distribution comparison (Before SVK_M & NPK_GG are in plant feed and after SVK_M & NPK_GG are in plant feed)

APPENDIX 6: Regression Results Summary

SUMMARY OUTPUT FOR MODEL (1)

Regression Statistics	
Multiple R	0.73
R Square	0.53
Adjusted R Square	0.49
Standard Error	7.46
Observations	108

ANOVA

	df	*SS*	*MS*	*F*	*Significance F*
Regression	8	6236	780	14	1.94E-13
Residual	99	5513	56		
Total	107	11749			

	Coefficients	*Standard Error*	*t Stat*	*P-value*	*Lower 95%*	*Upper 95%*
Intercept	682	401	1.70	0.093	-115	1479
NPKB	-639	405	-1.58	0.118	-1441	164
NPK _GG	-596	399	-1.49	0.138	-1388	196
NPK	-648	401	-1.62	0.109	-1442	147
SVK_U	-652	402	-1.62	0.107	-1449	144
SVK_M)	-628	402	-1.56	0.121	-1426	169
A3T	-638	402	-1.59	0.116	-1435	160
BB	-646	402	-1.61	0.111	-1443	152
C3 Stockpile	-650	401	-1.62	0.109	-1446	147

SUMMARY OUTPUT FOR MODEL (2)

Regression Statistics	
Multiple R	0.87
R Square	0.76
Adjusted R Square	0.74
Standard Error	26.87
Observations	108

ANOVA

	df	*SS*	*MS*	*F*	*Significance F*
Regression	8	231675	28959	40	9.01029E-28
Residual	100	72203	722		
Total	108	303878			

	Coefficients	*Standard Error*	*t Stat*	*P-value*	*Lower 95%*	*Upper 95%*
Intercept	0	#N/A	#N/A	#N/A	#N/A	#N/A
NPKB	0.67	1.10	0.61	0.55	-1.52	2.85
NPK _GG	1.92	0.25	7.56	1.94E-11	1.42	2.42
NPK	0.18	0.20	0.88	0.38	-0.22	0.57
SVK_U	0.06	0.12	0.51	0.61	-0.18	0.30
SVK_M	0.94	0.18	5.29	7.30E-07	0.59	1.30
A3T	0.61	0.28	2.15	0.03	0.05	1.17
BB	0.31	0.19	1.67	0.10	-0.06	0.68
C3 Stockpile	0.15	0.23	0.68	0.50	-0.29	0.60

APPENDIX 7: Line Fit Plots (Influence of rock types on flocculant consumption)

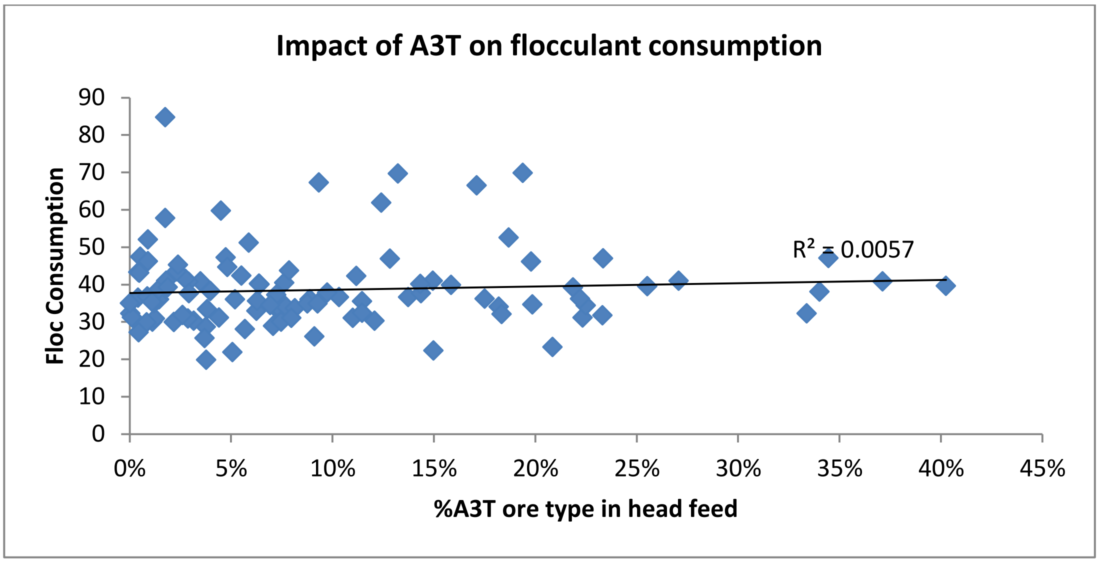


Figure 42: Line fit plot of A3T rock type on flocculant consumption

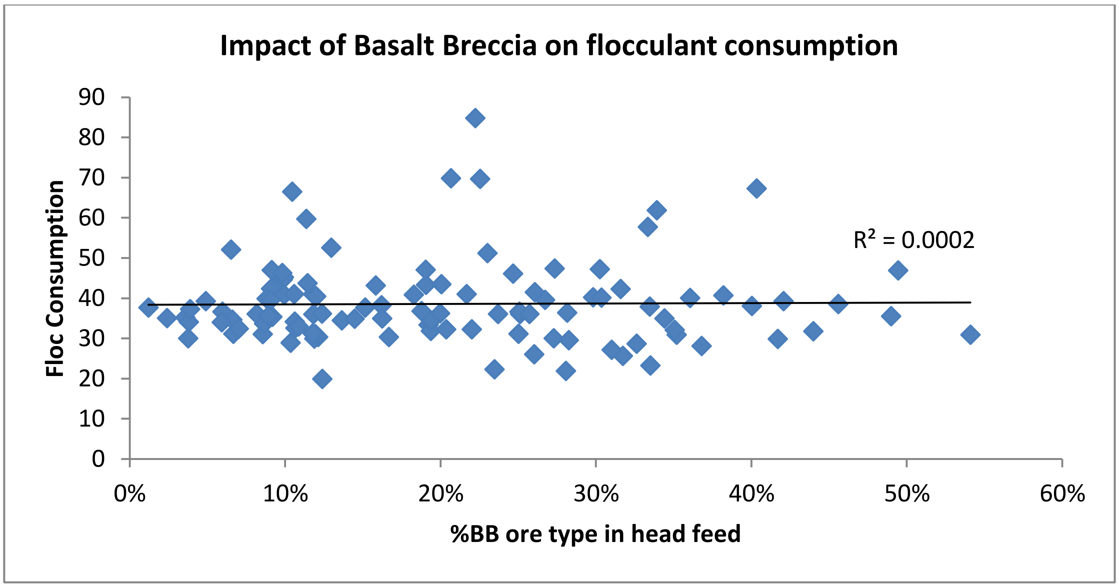


Figure 43: Line fit plot of Basalt Breccia rock type on flocculant consumption

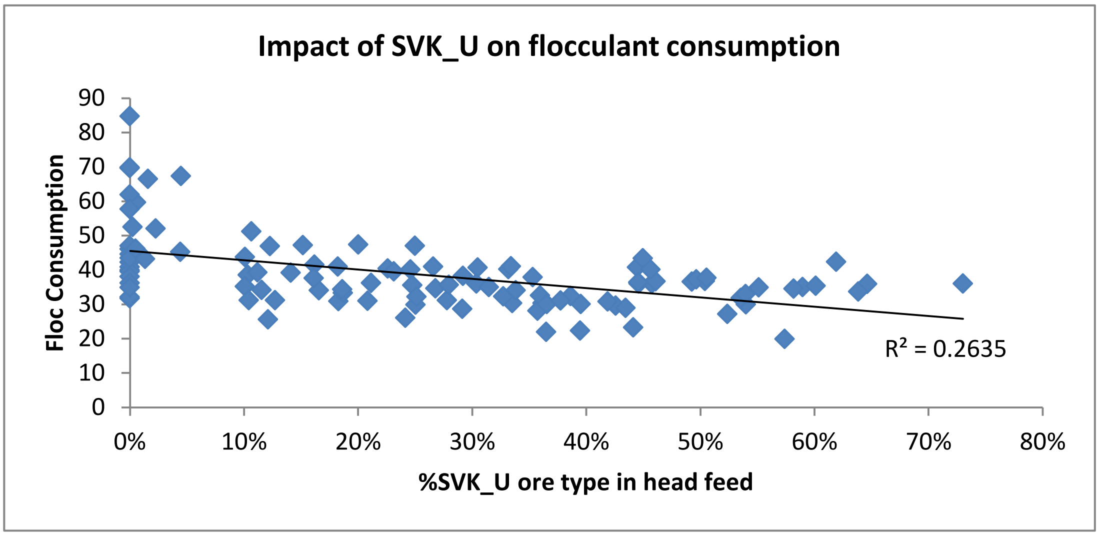


Figure 44: Line fit plot of SVK_U rock type on flocculant consumption

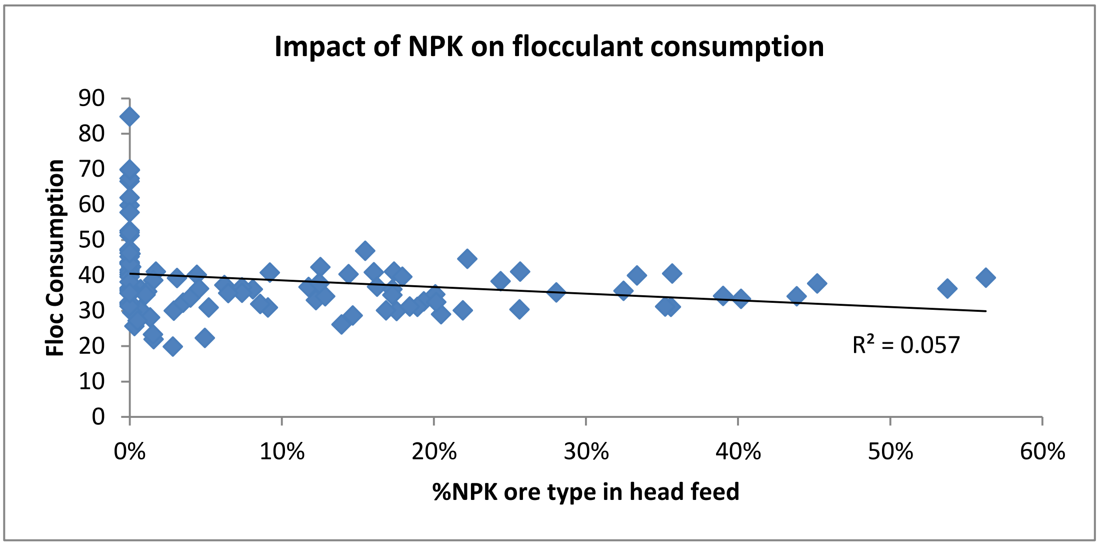


Figure 45: Line fit plot of NPK rock type on flocculant consumption

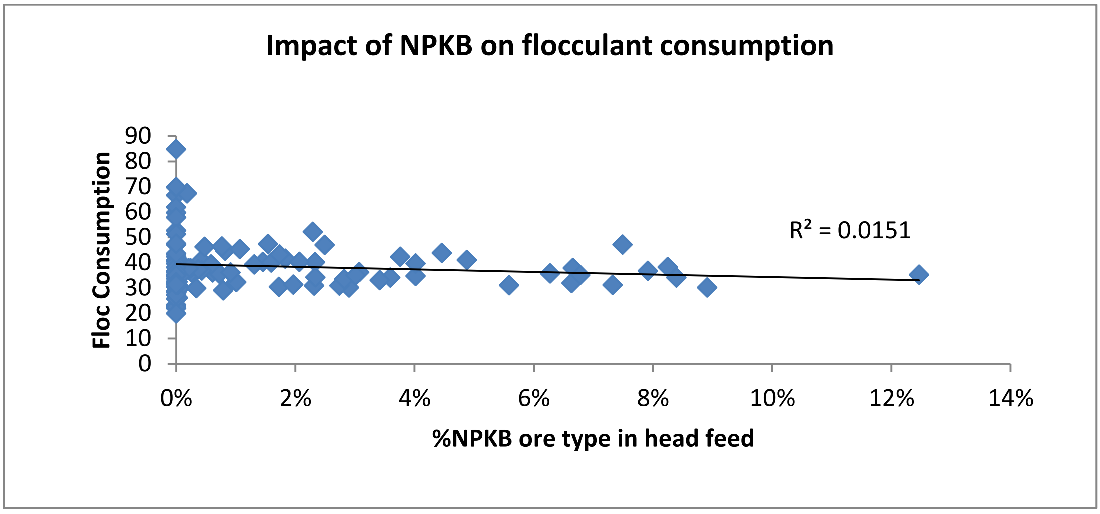


Figure 46: Line fit plot of NPKB rock type on flocculant consumption

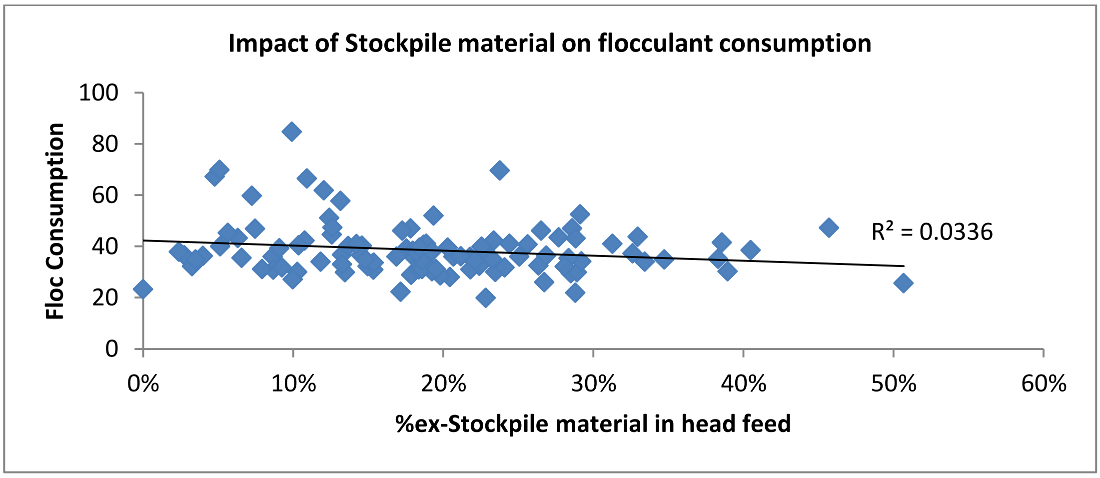


Figure 47: Line fit plot of ex-stockpile material on flocculant consumption

APPENDIX 8: Acronyms, Abbreviations and Definitions

CAPM	Capital Asset Pricing Model
DCF	Discounted Cash-Flow
DMS	Dense Medium Separation
ESP	Exchangeable Sodium Percentage
Flocculent	A water soluble substance that aids settling of solids in a water clarification process.
Flocs	Agglomerated particles of solids in slurry as a result of flocculent action. Due to a higher mass, the flocs will settle faster than the individual particles.
g/ton	Grams per tonne
IRR	Internal Rate of Return
mbgl	Metres below ground level
Mine mix	The blend of ore fed or being fed into the plant
NPV	Net Present Value
ODS	Ore Dressing Studies
OREP	Orapa Resource Extension Project
Mineral Resource	In situ estimation of mineralization, where there are reasonable and realistic prospects for eventual economic extraction. Mineral Resources are subdivided, in order of increasing confidence in respect of geoscientific evidence, into *Inferred, Indicated and Measured* categories. (extracted from The SAMREC Code)
Mineral Reserve	Mineable production estimate, derived from a Measured and/or Indicated Mineral Resource. Mineral Reserves are sub-divided in order of increasing confidence into *Probable Mineral Reserves and Proved Mineral Reserves.* (extracted from The SAMREC Code)
Sliming	A technical term that refers to poor clarification of water from thickeners, resulting in murky water being recovered.
Thickener	A mineral processing unit used for clarified water recovery through solids settling and generation of thicker slurries.
Tph	Tonnes per hour
Ultraseps	A new form of a thickener with no moving parts.
$/ton	Dollars per tonne

APPENDIX 9: Headfeed delays illustration due to settling challenges (without outliers)

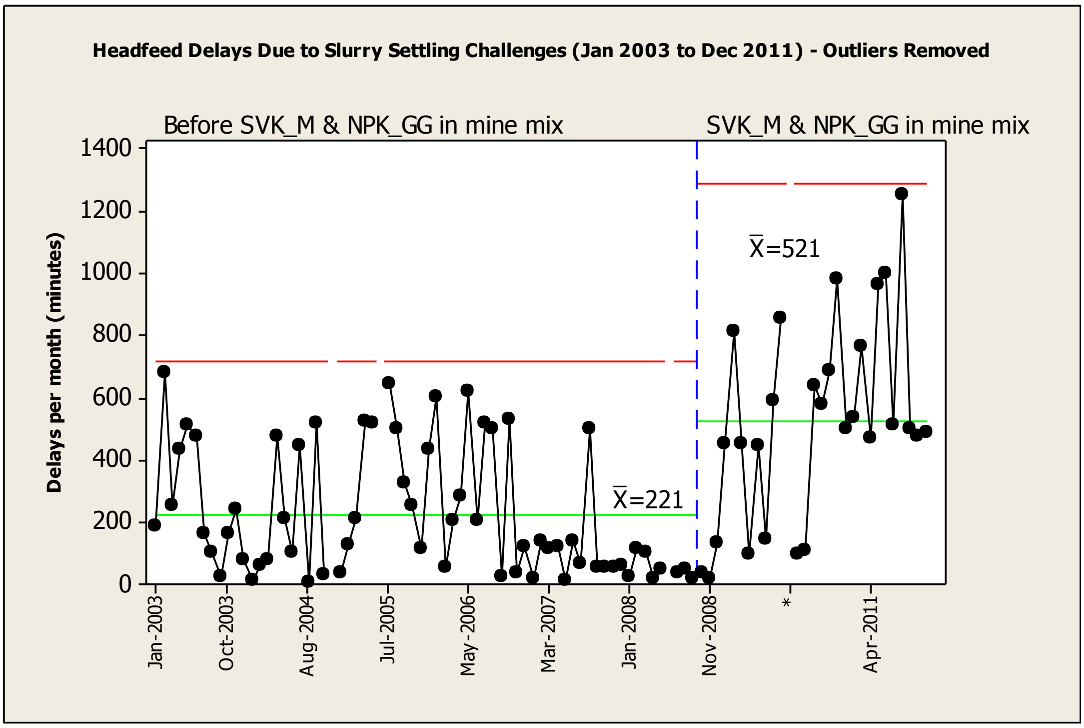


Figure 48: Slurry delays without outliers