The sensitivity of NPV to sampling and estimation decisions of a marine diamond mining project

by

Johannes Urbanus Burger

submitted in partial fulfilment of the requirements for the degree
MASTER OF SCIENCE (EARTH SCIENCE PRACTICE AND MANAGEMENT)

in the Faculty of Natural and Agricultural Sciences
University of Pretoria

September 2002

ACKNOWLEDGEMENTS

I wish to thank the management of De Beers Marine, in particular Barry Bayly, for the opportunity to pursue this work. I am also grateful for the company's financial assistance and allowing me the time to complete this study.

The following people are thanked for their assistance on this treatise: Chris Prins performed the simulation of the theoretical dataset and shared his knowledge on various statistical matters; Paul Bosma provided valuable discussion and ideas and also proof-read the manuscript; Kevin Smith provided me with an array formula that solved a nasty problem in the spreadsheet calculations; and Mike Shaw for proof-reading.

During the coursework component of the M.Sc., I thoroughly enjoyed the company of my fellow students and the various lecturers involved in the course. Thanks are also due to all the friends in Pretoria who had to put up with me during my visits.

I wish to especially thank Prof Hennie Theart for his guidance, encouragement and patience throughout the entire M.Sc. programme.

ABSTRACT

Title of treatise	:	The sensitivity of NPV to sampling and estimation decisions of a marine diamond mining project
By	:	Johannes Urbanus Burger
Study leader	:	Prof HFJ Theart
Institution	:	Department of Geology University of Pretoria
Degree	:	Magister Scientiae (Earth Science Practice and Management)
Date	:	September 2002

Marine diamond mining as practiced on the west coast of southern Africa is considered to be a high-risk venture. Investment decisions can be eased by using simulations to model likely outcomes. This study utilised Net Present Value (NPV) to measure the impact of sampling and grade estimation decisions on a marine diamond mining project. It focused on aspects such as the accurate determination of geological conditions, the influence of the degree of error in the geostatistical estimation process, sample density and sample support size. A simulated deposit was constructed that could be sampled using various parameters to measure the sensitivity of NPV. Various scenarios and their related NPV's showed that exploration costs have a small impact on a project in comparison with other cost aspects. However, the decisions made in the exploration process do have a significant impact on the NPV of a project. Inaccuracy in recovery efficiency and mining rate prediction, lead to a decline in NPV. Misfitting the semi-variogram model had a smaller impact on the NPV than the other scenarios investigated, but the importance of reflecting the true variance of the deposit in financial terms was evident. Finding the optimal sampling density and support size do have a positive effect on NPV. It is believed that the method demonstrated in this study can be used as a guide to add value in the selection of optimal parameters when planning exploration campaigns in marine mining projects.

SAMEVATTING

Titel van skripsie	:	Die sensitiwiteit van NHW ten opsigte van steekproefneming en skattings besluite van 'n mariene diamantontginning projek
Deur	:	Johannes Urbanus Burger
Studieleier	:	Prof HFJ Theart
Instansie	:	Departement Geologie Universiteit van Pretoria
Graad	:	Magister Scientiae (Aardwetenskap Praktyk en Bestuur)
Datum	:	September 2002

Mariene diamant ontginning, soos beoefen aan die weskus van suidelike Afrika, is 'n hoë risiko onderneming. Beleggings besluite kan vergemaklik word deur simulasies te gebruik om moonlike resultate te modeleer. Hierdie studie het Netto Huidige Waarde (NHW) gebruik om die impak van steekproefneming en skattings besluite op 'n mariene diamantontginning projek te meet. Aspekte soos die korrekte bepaling van geologiese kondisies, die invloed van foute in die geostatistiese skattings proses, steekproef spasiëering en steekproef grootte, is ondersoek. 'n Afsetting is met 'n simulasie geskep. Steekproewe is dan geneem van die afsetting om NHW sensitiwiteit teenoor verskeie parameters te meet. Verskeie scenarios en hulle verwante NHW's het gewys dat, in vergelyking met ander kostes, eksplorasie kostes 'n klein impak op 'n projek het. In teenstelling hiermee is die impak van besluite wat geneem word in die eksplorasie fase groot. Onakkurate herwinnings effektiwiteit en ontginningstempo skattings lei tot 'n afname in NHW. Swak modelering van die semi-variogram het 'n kleiner invloed op NHW, maar die belangrikeid daarvan om die ware variansie van die afsetting te reflekteer, was duidelik in finansiëele terme. Die gebruik van optimale steekproef spasiëering en grootte het 'n positiewe impak op NHW. Die metode wat gedemonstreer word in hierdie studie kan

gebruik word as ‘n gids om optimale parameters te selekteer wanneer eksplorasie programme beplan word vir mariene diamantontginnings projekte.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

1. The problem and its setting

1.1. Introduction

Marine diamond mining as practiced on the west coast of southern Africa is considered to be a high-risk venture. The technology is relatively new compared with the exploitation of diamond deposits on land, and marine weather conditions can be hostile. Also a factor is the high spatial variability of the ore body, both in terms of mineability and grade. The gravity of mining investment decisions is eased by strengthening the technical risk fundamentals through the use of various approaches that include a holistic view of the mineral resource cycle, the use of simulations as tools to model likely outcomes and an awareness of dependencies between technical factors. Experience has shown that effective and efficient sampling to establish mineral resources for transformation into reserves is critical for success. Identification and quantification of the significant parameters, prioritising their impact and reducing the variability of these parameters, through better understanding of the deposit, contribute to an improvement of the confidence level applied to resource estimates. The confidence level of the results will determine the ability to manage technical and financial risk effectively whilst delivering an acceptable return on investment through optimisation of the depletion strategy (Corbett, 2000).

This study was undertaken with the scenario sketched above in mind. The aim was to utilise Net Present Value (NPV) to measure the impact that sampling and estimation variables can have on a marine diamond mining project.

1.2. The statement of the problem

This study attempted to measure the impact of certain pertinent aspects in marine diamond mining in financial terms by measuring the sensitivity of NPV to various quantifiable sampling and estimation variables. It focused on aspects such as the correct determination of geological conditions, the influence of the degree of error in the geostatistical estimation process, sample density and sample support size, thus determining the impact of sampling and estimation decisions on the financial success of a project. Ultimately the geological model and the estimation and statement of grade determine the parameters according to which an extraction strategy is designed.

1.3. The subproblems

The problem statement of this study was divided into four subproblems.

- The first subproblem was to construct a baseline NPV against which the other permutations could be measured.
- The second subproblem was to quantify the impact and sensitivity of NPV with regards to misinterpretation of geological conditions that impact on mining extraction efficiency and mining rate estimations.
- The third subproblem was to quantify the influence of the degree of error in the geostatistical estimation process (the misfitting of the experimental semi-variogram with regards to range and nugget effect) on the NPV.
- The fourth subproblem focused on the measurement of aspects with regards to sample density and sample support size to see how these decisions impact on the estimate and the NPV.

1.4. Methodology

A mineralisation model of a marine diamond placer was constructed by means of a simulation. An existing dataset was ‘disguised’ and used to generate a theoretical dataset, thereby protecting the confidentiality of the original data without sacrificing the typical distribution found in diamond placers. The theoretical deposit was created using a sequential gaussian simulation. The simulated deposit could then be sampled using various parameters to generate estimates in order to do extraction/mine planning for the NPV model. After the extraction plan was designed, the deposit could then be 'mined' and a cash flow calculated for the various NPV models. These NPV models were then measured against a baseline NPV, which represents the scenario where the estimates and extraction parameters were accurately predicted.

The extraction strategy was designed with consideration of grade, mining recovery (the percentage of the total mineral content that can be recovered successfully with a given mining system) and mining rate (the tempo at which a certain area can be mined, usually expressed as m^2/day). To obtain the extractable grade of a block, the true or stated grade was factorised with a mining recovery factor. Applying the mining rate to the factorised grade, the amount of stones to be extracted per day, were obtained. Instead of a cut-off grade, a cut-off stone rate of 300 stones/day, was applied. The blocks were then grouped into mining panels of 9 adjacent blocks and the stone rate calculated for each panel. Panels were ranked from highest to lowest, based on the stone rate. This ranking was then used as an extraction sequence of extraction for the panels. Once the sequence was established, the number of stones that would be recovered in each year could be calculated and used to determine free cashflow for the NPV model.

1.5. Delimitations

This study was a modelling exercise and therefore the approach was to keep it as simple and elegant as possible, especially where elements were not directly related to sampling. As has already been stated, real grades were not used, but the distribution has been maintained. The study only looked at number of stones and a fixed value was assigned to the stones. Variations of size and quality, hence values per carat, were not considered in the study, as this would have introduced an extra level of complexity.

The choice of estimation technique is often driven by a combination of factors that include the nature and success of the sampling campaign. In this study, a standard approach, utilising kriging with the aid of semi-variogram modelling, was followed. This enabled better comparison between different scenarios.

Although geostatistical considerations play an important part in decision making with regards to exploration and sampling, it is not the aim of this study to focus on geostatistical aspects in detail, but to focus on the financial impact represented by sensitivity of NPV.

1.6. The data

The acquisition and treatment of the mineralisation data has already been described under methodology above. The other information utilised in the NPV model, such as acquisition and operating costs, were not taken directly from existing projects, but are projections based on actual figures. Care was taken however to maintain the ratios between the various cost aspects close to reality. Even though all the data in this study were simulated, the relationship with regards to actual projects has been maintained, thereby ensuring the relevance of the results to actual ventures.

1.7. The importance of this study

The offshore diamond mining industry is in many ways very different from conventional mining ventures. Commodity price forecasts, an important aspect in the valuation of more conventional resources (Rudenno, 1998), is not as critical in the diamond market. The reason is that, in relation to other commodities, diamond mining enjoys a relative stable market for their product largely due to the efforts of the Diamond Trading Company (formerly the Central Selling Organisation) to maintain supply and demand (De Beers, 2000; Stein, 2001). Another advantage is the accessibility of the ore blocks in this style of deposit where the only minor constraint on selection is imposed by the mooring system. On the other hand, the careful selection of production rates to optimise the value of a project as described by Smith (1997), is a futile exercise in offshore mining due to the narrow constraints and unpredictability imposed by the extraction technology. Another factor that induces a high level of risk and uncertainty in offshore mining projects is the high prevalence of new technology combined with a hostile and unpredictable physical environment.

It is with the relative uniqueness of the offshore mining industry in mind, that this study hoped to generate results and demonstrate an approach that may be used by mineral resource practitioners in the industry to guide in the selection of optimal parameters when planning exploration campaigns. The potential financial impact of decisions during the exploration stage over the life of the project could only be measured, once it has entered the exploitation phase. The quantification of risk, by means of NPV sensitivity, could be a valuable instrument in the production environment as it should contribute to more realistic and robust mine planning. The study hopes to highlight the levers that influence NPV and thus provide decision-makers with a tool to drive maximum value.

2. Background

2.1. Geology of the west coast placer diamond deposits

2.1.1. Introduction

The west coast of southern Africa contains one of the most spectacular regional alluvial deposits, stretching from the Namaqualand coast of South Africa northwards to the Skeleton Coast of Namibia (Figure 1). Since the principal discovery near Lüderitz in 1908, this region has produced in excess of 100 million carats of gem quality (>93%) diamonds. In this regional province, the diamonds are hosted in a variety of world-class placer types that, broadly speaking, encompass fluvial, marine, deflation and aeolian settings, with temporal ranges from at least the Cretaceous to the modern day (Ward, 1998).

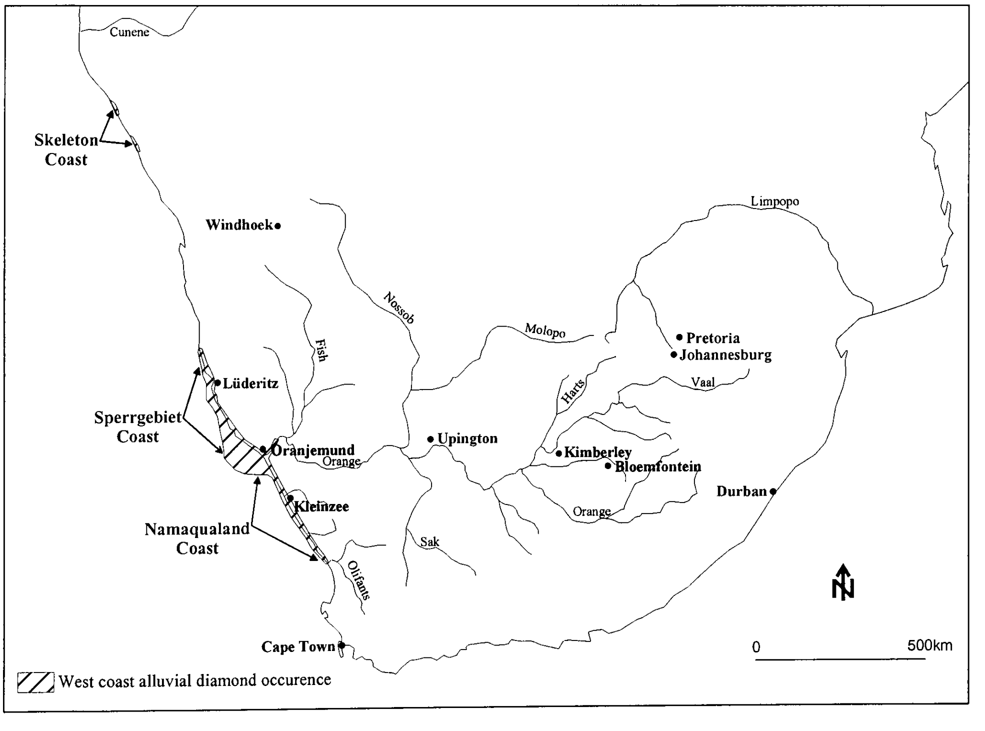

Figure 1. Distribution of the principal coastal alluvial diamond occurrences in southern Africa. The Orange-Vaal system is the principal westward directed drainage from the cratonic area of southern Africa (after Ward, 1998).

Marine diamonds originated mostly from Cretaceous-age kimberlites intruded into the interior of South Africa between 120 and 80 million years ago. Erosion followed continental uplift after the break-up of Gondwana and as much as 1400m of the original depth of numerous kimberlites and the surrounding country rocks have been eroded (Tankard *et al*, 1982; Gurney *et al*, 1991). The major period of diamond supply, along with vast quantities of eroded material, to the west coast was during the humid Cretaceous when most of the denudation of the subcontinent took place. Two major rivers drained most of the interior: The Kalahari River in the north and the Karoo river in the south (Figure 2). By the Late Cretaceous-Early Tertiary, the gross arrangement of the Orange/Vaal River drainage system had established itself more or less as it is today, implying the lower part of the Karoo River had already been captured in the Early Tertiary by the Kalahari drainage (De Wit, 1999).

The climate during the Cainozoic induced a prevailing southerly wind regime that generated powerful, northward directed longshore currents and also induced arid conditions along the southern African coast. Consequently, much of the debouched sediments on the western margin has been moved northwards along the coast in a shelf bypass system, giving rise to thick mudbelts offshore central Namibia, extensive desert sand seas and dunefields onshore, thereby leaving a gravel rich southern head in coastal deposits. These deposits have been subjected to considerable reworking during repeated marine transgressions and regressions, giving rise to the major diamond plays along the west coast (Corbett, 1996; Ward, 1998).

The complex interaction of high-energy fluvial, marine and sedimentary environments over the last 80 Ma was responsible for the formation of the greatest placer concentration of high quality gem diamonds known on earth (Corbett, 2000). During 90 years since mining began, the global significance of the deposits remain unsurpassed (Corbett, 1996).

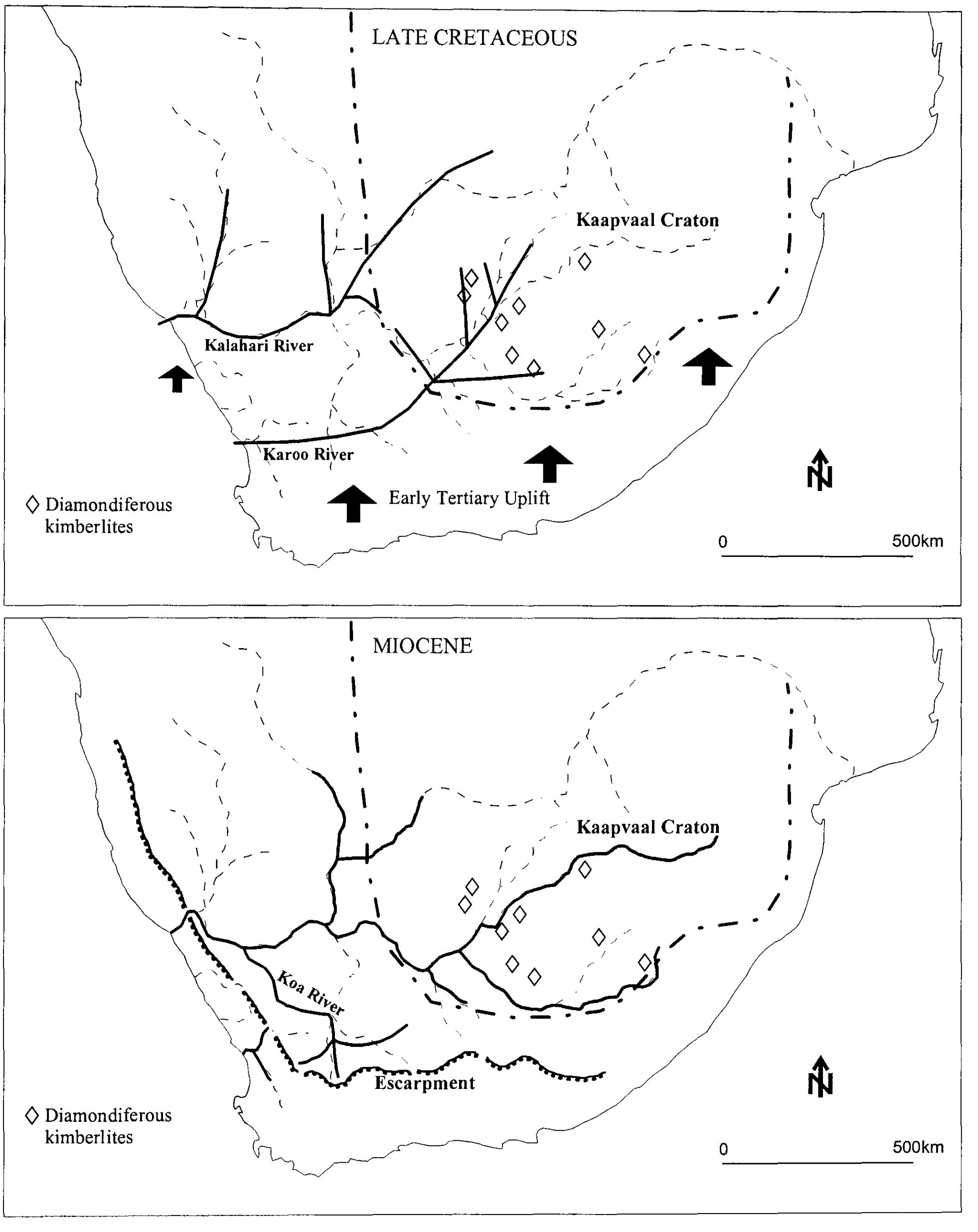

Figure 2. Paleodrainage reconstruction of the Cretaceous and Miocene (after De Wit, 1999).

2.1.2. Geology of the marine diamond placers

The understanding of the development of the southern African continental shelf has been aided by extensive geophysical data acquisition across the shelf and continental slope. The regional offshore framework is being refined through integration with very high-resolution geophysics for orebody characterisation. This is supplemented by micro-palaeontological dating that has been extensively cross-referenced using radiometric techniques (Corbett & McMillan, 1998).

The difficulty in dating key onshore sedimentary sequences do however complicate the determination of the role played by tectonics in driving post-Gondwana changes in fluvial system behaviour and sea-level. The presence of major unconformities and the consequent absence of much of the sedimentary record, does pose problems with correlation. The offshore framework however provides clear evidence for the persistent influence of subtle tectonics, even on a so-called passive margin. The tectonically driven sequence stratigraphy is probably based on Milankovich cycles for all sequences preserved on the western continental margin. Each cycle represents unique uplift (erosion), stillstand (non-deposition), subsidence (sedimentation) conditions which, coupled to the nature of the palaeoclimate and the type and quantity of terrestrial sediment supply, has led to the stratigraphic succession on the continental margin (Corbett & McMillan, 1998). A simplified cross section of the continental shelf illustrating the stratigraphic succession is shown in Figure 3.

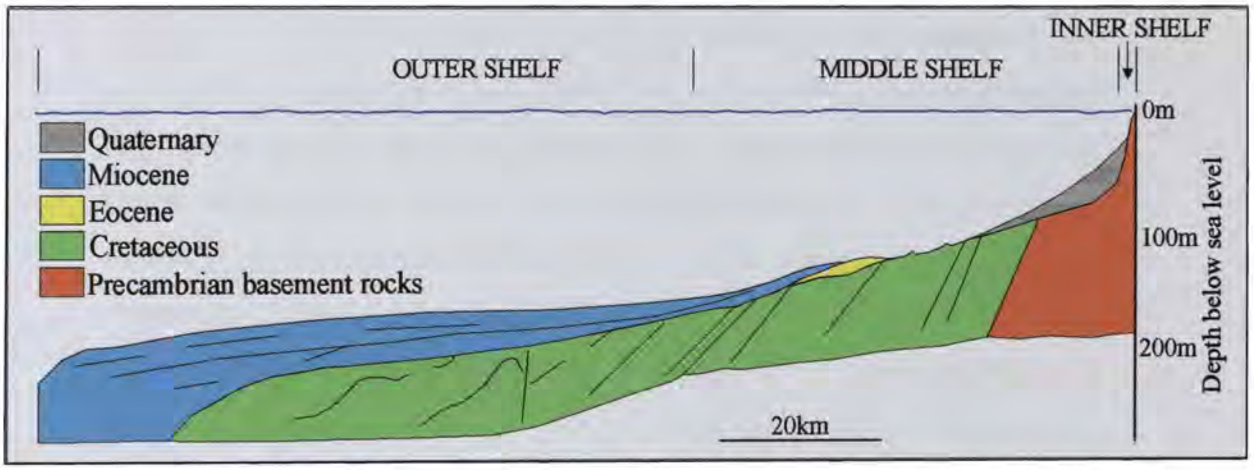

Figure 3. Simplified cross section of the western continental shelf of southern Africa as interpreted from airgun seismics (Stevenson, I. unpubl.).

The Orange River has largely been responsible for the introduction of diamonds to the continental margin of Namibia (Corbett, 2000). Investigation of the Miocene and Plio-Pleistocene sequences exposed by mining operations within the Orange River valley, indicate that the proto-Orange operated as a canyon-type system through the Richtersveld Mountains (Jacob *et al*, 1999). Incision contributed significant quantities of coarse clastics to the continental margin, which created ideal conditions for the extensive development of coarse clastic beaches that are characteristic of the high-energy coastal plain and continental shelf. The availability of coarse bedload undoubtedly contributed to the extreme erosional power of shoreface processes, resulting in the development of complex patterns of rugged terrain formed by gullies and cliffs on coastal platforms composed of Precambrian rocks (Corbett & McMillan, 1998). The clearest evidence of this is that of a stand at approximately –20m where a distinct wave-cut nick is usually found. Deep gullies have been incised into the face of the cliff as seen in Figure 4 (Murray *et al*, 1970).

Due to the high-energy coastal conditions, fan-delta sequences are probably reworked rapidly during shoreface progradation on the west coast. The history of coarse gravel accumulations along submerged palaeoshorelines is complex as multiple episodes of sediment induction and reworking are represented (Corbett & McMillan, 1998). This consisted of repeated marine regressions and transgressions spanning 60Ma. The destruction of exposed clastic beaches by arid zone processes has released diamonds for further transport by the aggressive sand laden southerly winds. The combined interaction of fluvial, marine and aeolian sedimentary environments along the Namibian coastline essentially acts as an enormous ‘sediment transport conveyor’ breaking down, recycling and transporting large volumes of sediment to the north (Corbett, 2000). This ‘sediment transport conveyor’ also effectively separated the fine and coarse-grained components from each other, hence the gravels were deposited on beaches, very fine material on the outer shelf and the sand fraction in the sand-sea of the Namib.

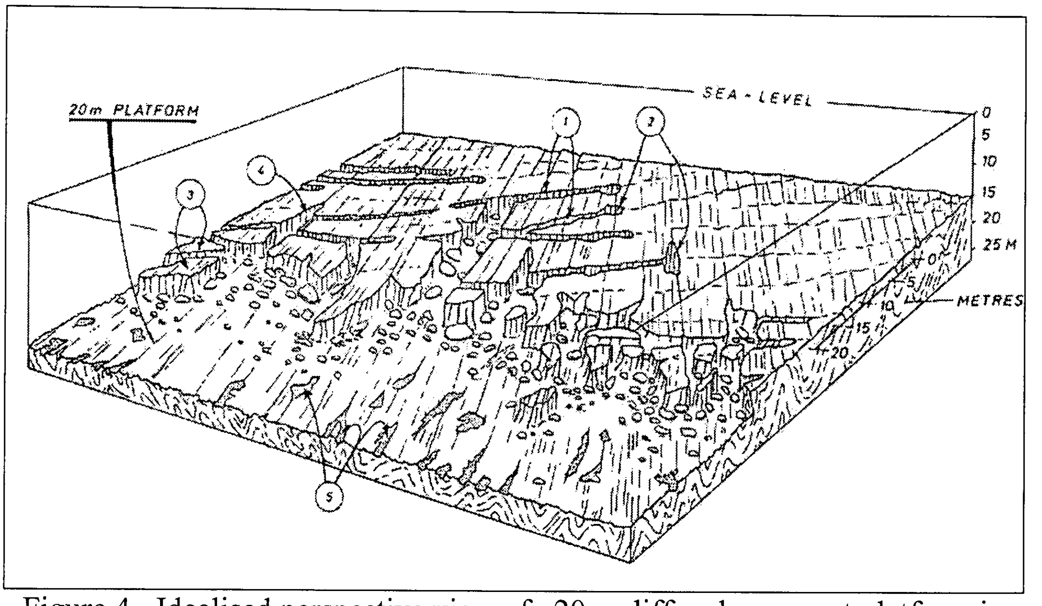

Figure 4. Idealised perspective view of –20m cliff and wave-cut platform in Kerbehuk area (Namibia) showing (1) joint gullies, (2) terminal potholes, (3)stacks, (4) strike gully and (5) slope gullies in –20m platform (Murray *et al*, 1970).

The onshore and shallow-water orebodies are usually underlain by Precambrian footwall that often form a gullied terrain (Figure 4). In contrast, the footwall of the middle and outer shelf consists of both unconsolidated and cemented Cretaceous and Tertiary sediments. This fundamentally influences the physical character of the orebody. The deep-water orebody is a large, laterally extensive orebody with highly variable physical character. This is in contrast to the narrow linear beaches that are characterised by rugged, fixed bedrock trapsite morphology. Repeated regressions and transgressions across the middle to outer shelf has presented the high-energy shoreface with numerous opportunities to erode, truncate and plane the Cretaceous and Tertiary shelf sequences. Gravel deposits were comprehensively reworked to produce a coarse gravel lag containing large intraclasts (meters in diameter) of locally derived shelf sedimentary rocks plus quartzite dominated terrigenous cobble gravels (Corbett, 2000). A cross section view of the gravel lag can be seen in Figure 5.

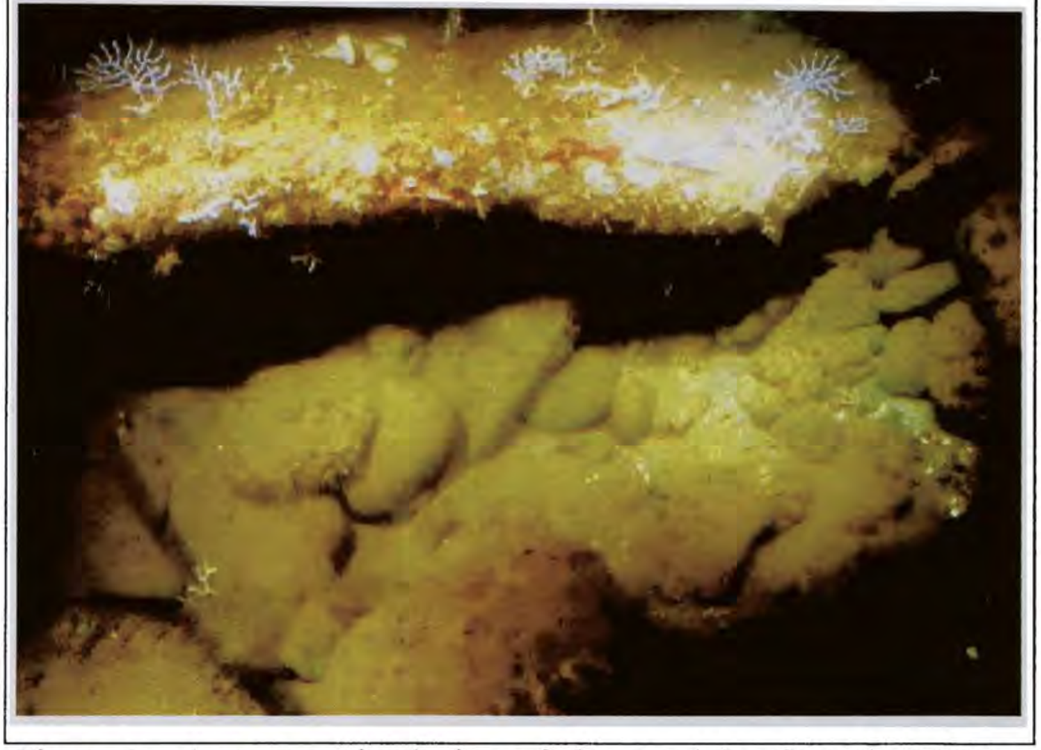

Figure 5. A cross sectional view of the gravel lag from the JAGO submersible. Quartzite cobbles are overlain by a locally derived sandstone slab.

Although fixed trapsites do occur, they are less resistant to erosion than the more competent Precambrian bedrock of the inner shelf. Diamond distribution is therefore more likely to be controlled by sedimentary processes than bedrock lithology and structure. Hence the deeper water deposits are generally less patchy and exhibit greater continuity compared to the onshore linear and pocket beach environments, but they are usually characterised by lower grades (Corbett, 2000).

2.2. The development of the offshore diamond industry

The first discovery of diamonds related to marine deposition in southern Africa was on land in 1908 near Lüderitz, Namibia. This led to subsequent discovery of rich diamond fields along the west coast of the then German South West Africa, and the development, within a few years, of a huge industry in this arid and inhospitable region. Later, diamonds were also discovered and mined elsewhere along the vast coastline from south of the Olifants River in South Africa to north of Hottentot Bay in Namibia, a distance of about a 1000 km (Gurney *et al*, 1991).

Sammy Collins, a Texan oilman laying an offshore pipeline for the delivery of diesel fuel to Oranjemund, questioned whether extensions of the onshore marine diamond deposits could exist offshore. The MDC (Marine Diamond Corporation) came into being and Collins had the *Emerson K*, a 670 ton steam tug, converted into a sampling/mining vessel. In November 1961, the first marine diamonds were recovered near Lüderitz, and in total 45 stones weighing 9 carats were found. Further exploration in water shallower than 30m yielded results, justifying the conversion of a series of barges and vessels into mining and sampling platforms during the 1960's. Despite successes, MDC was in financial trouble and in 1965 De Beers became a majority shareholder in MDC (Haage, 1997). Ocean Science engineering was contracted by De Beers to perform a comprehensive evaluation of the shallow water economic potential along the west coast in 1964 using the *m.v. Rockeater*. This was the first vessel to undertake a comprehensive offshore seismic and sampling programme to locate diamond deposits. The Oceanographic Research unit of Anglo American Corporation provided services for detailed mapping of nearshore marine deposits from 1965. In 1971 mapping was undertaken using the newly developed technique of side scan sonar (MDC tested the third unit ever produced). Seismic and side scan sonar data aquisition continues to be important to this day (Corbett, 1996).

By 1971 known resources were depleted in the MDC shallow water concessions and the mining operations were scaled down. During 1971 the sampling vessel was prevented from continuing work in the inshore due to a period of bad weather. A series of grab samples were collected in water depths of 100m. These samples indicated the presence of significant concentrations of heavy minerals in lines parallel to the coastline. This was the first indication of palaeo beaches beyond 35m water depth (Haage, 1997).

Between 1972 and 1983, survey and sampling delineated a low-grade, patchy, but aerially extensive diamond deposit over tens of square kilometres off the Namibian coast. The realisation that new subsea mining technology would be required to exploit the deep water deposits led to the formation of De Beers Marine in 1983 (Corbett, 1996). Various mining systems and equipment were conceptualised between 1985 and 1988 while detailed geological work was being carried out simultaneously in the Atlantic 1 concession. The *m.v. Louis G. Murray* was the first platform acquired by De Beers Marine and was utilised as an experimental mining vessel for various concepts (Haage, 1997).

The conversion of the *m.v. Louis G. Murray* with a crawler mining system heralded the start of official production in the Atlantic 1 concession off the Namibian coast, when 28,663 carats were recovered in 1990. An alternative approach was also pursued in the form of the large diameter drill mining system, a combination of offshore oil drilling and tunnel boring technology. The *m.v. Coral Sea* was acquired in 1989 for this purpose. The *m.v. Coral Sea* started production in 1991 and excellent results prompted the expansion of drill ship mining as well as increased recoveries (Haage, 1997).

Expansion of the mining fleet continued throughout the nineties with production vessels *m.v. Grand Banks*, *m.v. Debmar Atlantic*, *m.v. Debmar Pacific* and *m.v. !Gariep* contributing. Two sampling vessels *m.v. Douglas Bay* and *m.v. Coral Sea* (now fulfilling dual sampling and mining roles) are still actively involved in resource and reserve generation, whilst the *m.v. Zealous*, which performed geosurvey work, was replaced with an AUV (autonomous underwater vehicle) in 2001. Production from water depths that exceed 100m in the Atlantic 1 concession passed the half-million carat mark for the first time in 1999. Year 2000 saw production at 576,470 carats. (Corbett & McMillan, 1998; De Beers, 2000).

De Beers Marine remains the largest diamond exploration and mining contractor on the west coast of southern Africa with a fleet of 6 vessels and the AUV capable of providing a full range of geophysical, sampling and mining services. There have, however, been various other companies active in marine mining on the west coast over the past decade and the industry has seen a lot of activity and intrigue, especially over the last three years (Cape Business News, 1999). The main players, apart from De Beers Marine, are Namibian Minerals Corporation Limited (Namco), Trans Hex and Diamond Fields International.

Namibian Minerals Corporation Limited (Namco) is a Canadian based company that until recently showed potential to unseat De Beers Marine as the number one marine diamond miner (Hasenfuss, 2000). However, in January 2001, the wheels came off when already tight cash-flows were exasperated by the loss of production, after one of their seabed crawlers broke in two off Lüderitz when the surface vessel tried to raise it. Surviving provisional liquidation, the company remains a player on the diamond coast (Spicer, 2001).

Trans Hex, traditionally an onshore and shallow water diamond miner, was catapulted into a fully-fledged deep water mining operation with the company's joint venture with Canadian Diamond Fields International (DFI) to mine diamonds off the Namibian coast (Trans Hex News, July 2001).

2.3. Techniques in mineral resource generation

The development of mineral resources and their transformation into reserves is a complex process involving many variables. This process requires widespread collaboration across many different functions to deliver optimised mine plans to achieve required and acceptable levels of profitability and to ensure that the mineral resource is depleted in a responsible manner (Corbett, 2000). De Beers Marine developed a holistic approach to mineral resource management that is illustrated in Figure 6.

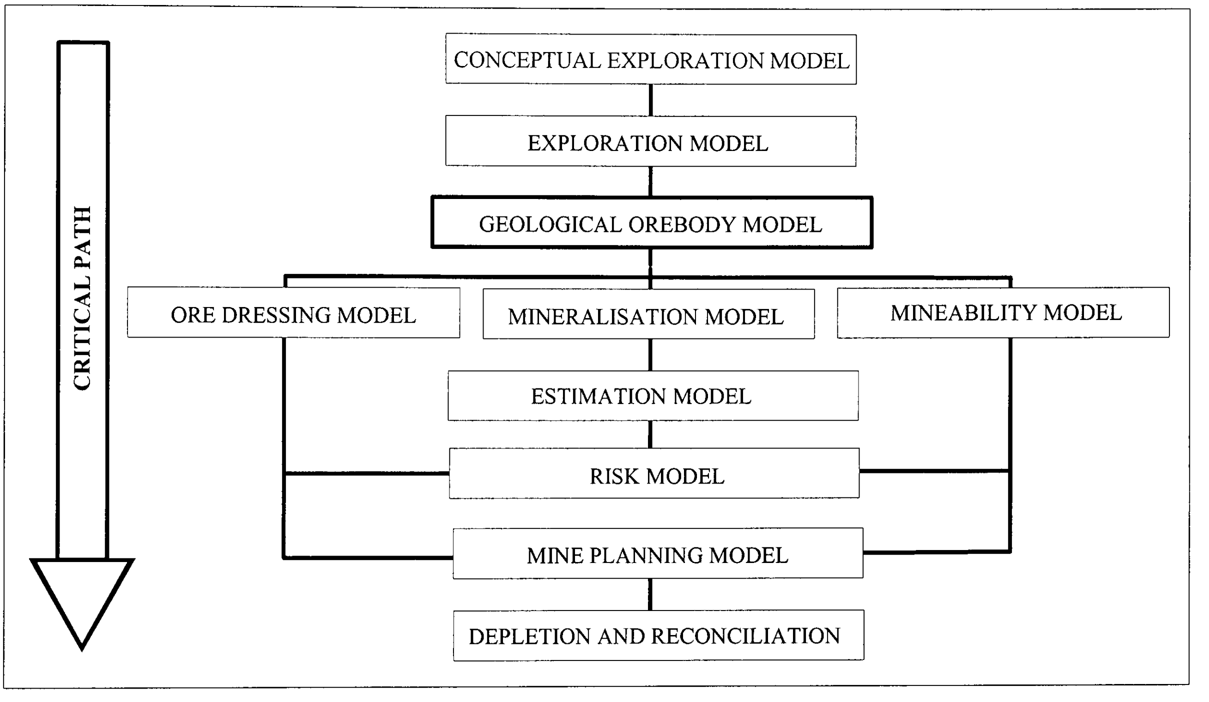

Figure 6. The mineral resource management cycle as employed by De Beers Marine (Corbett, 2000).

Bujtor and McMahon (1983) divided the components of a mineral reserve estimate as follows:

- Sampling
- Geology
- Estimation methods
- Mining factors
- Metallurgical factors
- Environmental factors
- Financial and other factors

Each of these components impact to varying degrees on the recoverable metal or product of a new mining project.

2.3.1. Generation of a geological model

Some of the geological factors which need to be considered in mineral reserve estimation include:

- The size, shape and attitude of the orebody,
- The structural and lithological characteristics of the orebody and country rock,
- The relationship and distribution of the economic mineral with other components of the orebody,
- Enrichment processes,
- Assay populations, means, grades, thicknesses and specific gravities.

The impact of poor geological information on the viability of a project can be disastrous. Many problems arising in the estimation of mineral reserves can be ascribed to a lack of emphasis placed on the role that geology plays throughout the development process (Bujtor & McMahon, 1983).

Various techniques are used to gain information about the orebody in the offshore environment. This includes the acquisition and interpretation of geophysical data, geotechnical sampling, palaeontological dating and seabed observation.

The most common techniques in geophysical data acquisition are side scan sonar, seismic sub-bottom profiling and swathe bathymetry. Geophysics has seen rapid technological development since the early days of marine exploration, largely driven by the growth in the information technology industry and advancements in positioning systems. Low resolution paper recording and manual mosaicing has made way for digital data acquisition, processing and interpretation (Stevenson & Nicholson, 1998). More accurate positioning is achieved using an acoustic baseline array on the seafloor combined with differential global positioning systems. The demand for increased resolution of sub-bottom profiling was met through the co-development of one of the world's first CHIRP sub-bottom profiler systems by De Beers Marine and the Institute for Maritime Technology in Simonstown. The CHIRP sonar is a frequency modulated, quantitative sub-bottom profiler, which produces high resolution, low noise, wide band acoustic signatures of the seabed sediments that is used to construct high resolution shallow seismic sections (Stevenson, 1998). De Beers Marine was also the first operator to utilize an autonomous underwater vehicle (AUV) in 2001. Apart from the cost advantages of utilizing an AUV, the use of an untethered vehicle provides platform stability that enhances data quality. It can deliver photo quality side scan sonar, swathe bathymetry and 3D seismics (Ricketts, L. pers. comm.).

The advantages of geosurvey include the rapid and relatively inexpensive coverage of large areas of the seafloor that can be interpreted for potential targets, as well as contribute to the geological model of an area under consideration. Geotechnical sampling consists of a host of techniques that include vibracoring, rockdrilling or cactus grab sampling. Cores in particular are popular and are used for various geological interpretations that include ground-truthing of geophysical observations, sedimentological analysis, stratigraphic reconstruction and dating. Micropalaeontology is used as a dating technique and forms an essential part of diamond exploration on the continental shelf. Micropalaeontological studies are aimed at establishing the age and environment of deposition of rock samples collected from the seafloor. Comparable samples from both onshore and offshore deposits are combined with geophysical interpretations, environment data and age, to build a geological model. Consequently a detailed history of the distribution and depositional environment of potential diamond deposits as they accumulated through time, is obtained (Dale & McMillan, 1999).

Another technique that is utilised along the west coast is direct visual observation of the seabed. In areas where water depth prohibits access to divers, remotely operated vehicles, with cameras attached, and mini submersibles have been used. The direct observation of mine faces, sample holes and natural seafloor exposures provides information about the characteristics of the orebody and also serves as a method to ground truth geophysical observations. Ultimately, information about the physical characteristics of the orebody on the shelf, the environment of deposition and the dynamics of the sedimentary systems that formed it, contribute to a better understanding of the geology (Corbett & McMillan, 1998).

2.3.2. Evaluation of the orebody: Sampling

Sampling is the primary means by which the dimensions and values of an orebody are measured. Any errors in the sampling data will carry through to all subsequent stages of an evaluation and no amount of sophisticated mathematical manipulation can compensate for poor quality data. Bias and inaccuracies can also be introduced in the analysis or processing of the sample. A lack of reliability and representativity of sample results could have serious repercussions throughout a project (Bujtor & McMahon, 1983).

Covacic and Clarke (1990) give a good definition of what a sample is:
'Sampling is defined as the operation of removing a representative part, convenient in size for testing, from a whole of much greater bulk, in such a way that the proportion and distribution of the quality measured are, within reasonable limits, the same in both the whole and the part removed: the sample.'

Aspects of the geological model, such as geological homogeneity must be considered in the sampling campaign design. A well planned approach to sampling can keep costs down by minimising the number of samples necessary and can also be changed on short notice if insufficient information is obtained from it.

The following aspects needs to be considered before a sampling campaign is started (Prins, 1991):

- Sample support size
- Sample size, i.e. how many samples are necessary
- Sample contiguity
- Sample grid orientation
- Sample frequency if a continuous process must be monitored
- Minimum sample density

For the diamond deposits in the offshore environment, area as defined by the sample tool footprint, is the sample support size. This is in turn used in the estimation process to express the grade per m^2. There are two reasons for this: firstly, the orebody is laterally extensive and thin and it is very difficult to define the thickness accurately with current technology; and secondly, the extraction of the sample utilising airlift, adds large quantities of water to the sample, making accurate weight measurements difficult.

The overall objective of sampling is to determine whether the deposit, or parts of it, is worth mining. After completion of geophysical interpretation and ground-truthing, the exploration sampling phase is intended to determine the average grade of the global resource and to identify mineralised targets worthy of further investigation. Follow up sampling campaigns are designed to delineate blocks with a level of accuracy so that a mine plan can be developed (Garnett, 1998). In the case of De Beers Marine, two sampling vessels are being operated. The *m.v. Douglas Bay* is equipped with a 'megadrill' and plays an important role in finding and developing resources. In order to overcome the rugged orebody conditions encountered in the Atlantic 1 concession area and deliver a more accurate resource estimate, the *m.v. Coral Sea* has been equipped with a large bore 'decadrill system' seen in Figure 7 (Corbett, 2000).

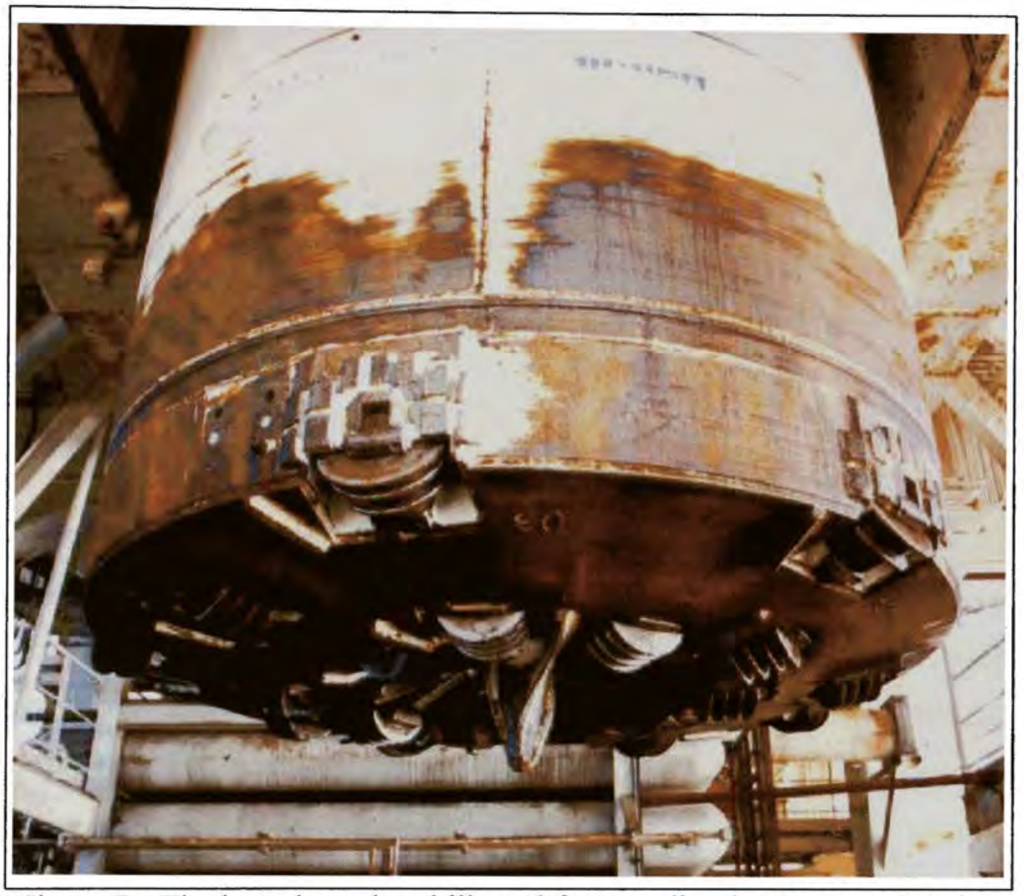

Figure 7. The large bore decadrill used for sampling by De Beers Marine was designed by Wirth GmbH and is related to the technology used for drillship mining.

Samples are collected and treated using the standard metallurgical techniques for diamond recovery. Sample integrity is maintained by utilising a batch process and a purging cycle. Samples are logged for geological information as it passes through the plant. Important information includes data on the nature of the gravel, footwall type and volume of plantfeed material. Offshore sampling operations are expensive and it is important to optimise the design of these programmes. The high spatial variability of the orebody coupled with the often robust nature of the gravels, presents considerable challenges to the sampling system. The accuracy and resolution of the geological model plays a critical role in determining the spacing and orientation of a sampling grid required to produce results at the desired confidence level with a specific sample size (Corbett, 2000).

2.3.3. Evaluation of the orebody: Estimation

Many different methods exist for the estimation and assessment of mineral bearing ground. A specialised field of statistics called geostatistics was developed as the understanding of geological phenomena increased and specialised techniques exist today for the successful evaluation of mineral deposits. Modern mining ventures are often marginal in terms of economic viability, thus putting more and more pressure on stable and reliable estimation techniques for mineral evaluation. Where in previous years, techniques like averaging, guessing or inverse distancing were sufficient, more advanced techniques need to be applied today to ensure more reliable results (Prins, 1991).

Classical statistics makes the underlying assumption that individual sample values are independent of one another and randomly distributed. The special significance of the relative positions of the samples are ignored and it is assumed that all sample values in the mineral deposit have an equal probability of being selected. The likely presence of trends and zones of enrichment in the mineralisation is generally ignored. The fact that two samples taken close to each other are more likely to have similar values than if taken far apart is similarly ignored. Geostatistics on the other hand incorporates the spatial relationship of samples and their geometrical relationships. The basic assumption being made is that the value of a sample is a function of its position in a mineralisation zone of any given deposit, and the relative position of the sample is taken into consideration. The similarity between samples is quantified as a function of the distance between samples by means of a semi-variogram. The objective is to overcome the limitations of the use of classical statistics, and thus incorporate the effect of geological controls on mineralisation trends (Bujtor & McMahon, 1983).

Geostatistics has proved itself to be successful in the modelling of geological phenomena. The big difference between geostatistics and other purely mathematical estimation processes is that there is an opportunity for the interpretation by the geologist of the deposit to be incorporated into the estimation process, thus making use of all available information obtained from the sampling and the geological data, and not only using the grade of the mineral under investigation. Geostatistics is therefore a statistical method of incorporating all information (grade and geological) into an estimation method (Prins, 1991).

The resource and reserve estimation of marine diamond deposits should be based on a systematic sampling grid. The distribution of the sample grade results tends to be very skew and has a large variance. A bias is easily introduced due to the fact that the grade distribution of the samples is frequently very different from the grades of the area of influence, indicating an erratic diamond distribution. The distribution of the diamonds within the deposits can be simulated, based on the histogram of the sample results and on the semi-variogram (Rombouts, 1998). More accurate mineral resource estimation has been aided by many new geostatistical techniques that have been developed in the offshore environment over the years (Corbett, 2000).

2.4. Mining in the marine environment

Seagoing, 'pick-up-and-move' floating mines have one basic design to tackle a variety of deposit types. Due to variations in geology, it is not financially feasible to have an aircraft carrier sized vessel with a variety of dredging techniques. According to Louw (1998) the vessel must have five basic requirements:

- It must be versatile enough to exploit a large spectrum of terrain types,
- It must have huge dredging capacity and a processing plant large enough to handle the dredged gravels,
- It must be big enough, with a matching mooring system, to withstand the sometimes fierce onslaught of the sea,
- The dredging equipment must be easily deployable in adverse weather conditions,
- It must have low operating costs to allow mining of low grade deposits.

The most common methods employed in the industry to extract diamondiferous gravels from the seabed includes airlift dredging, remotely operated seabed crawlers and large diameter airlift drilling. Figure 8 illustrates the horizontal (crawler) and vertical (drillship) approach utilised by De Beers Marine.

The overall *in situ* global ore resource estimates for a mineral deposit, although containing general information about the deposit, are of limited value for a feasibility study. What are really required are the mineable reserves. This generally implies a cut-off grade and other mining constraints, applied to the potentially mineable areas. When estimating the mining reserves, it is necessary to have a good understanding of the mining method involved and its likely impact upon the parameters used in the evaluation – particularly recovered grade. There are various mine planning and mine operating factors which combine to alter and usually reduce the potential recoverable grade of the *in situ* ore (Bujtor & McMahon, 1983).

Both historically and at present, the greatest variance in recoveries against estimates, result, from significantly lower mining rates than thought possible by the operators and not from grade shortfalls. The most obvious risks result from the working environment and technical risk that reduces the availability of the mining equipment (Garnett, 1998a).

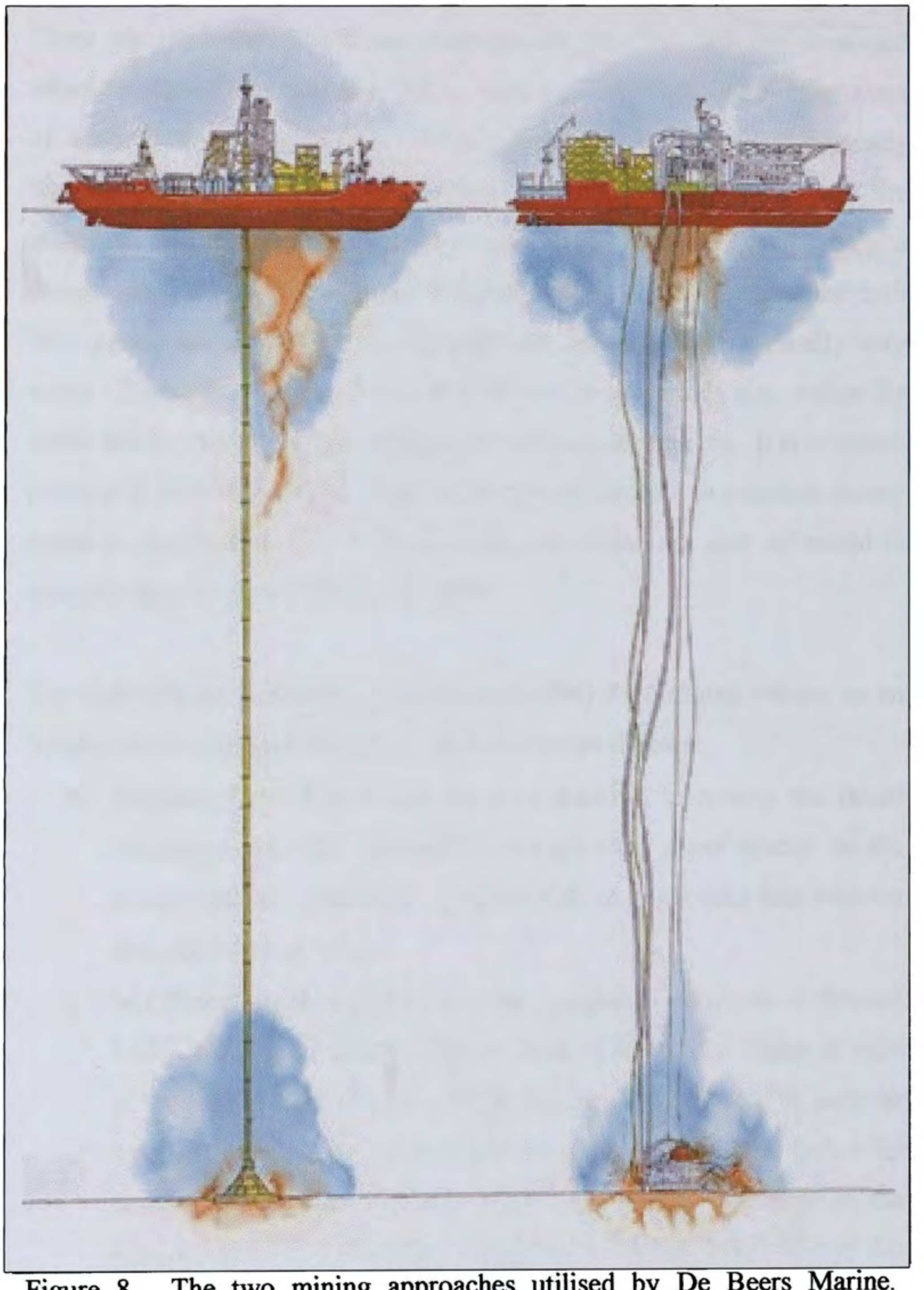

Figure 8. The two mining approaches utilised by De Beers Marine. Vertical or drillship mining is illustrated on the left and horizontal or crawler mining on the right (De Beers Marine Corporate Communications).

2.5. Net Present Value (NPV)

2.5.1. Measures of economic performance

There are a number of different methods available for defining economic returns on mining investments. All of them are based on some or other form of analysis of the Cash Flows. A ten year period of analysis is normally adequate to calculate return on investment. This is because most of the useful techniques are based on the time value of money and imply a compounded discount to successive cash flows, so that by the time the cash flow period exceeds 10 years their effect on the analysis is normally very small. Standardisation on the number of cash flow periods also makes for easier and more meaningful comparison between alternatives. It is common practice to record all of the pre-production cash outflow in constant money terms at inception or year 0. Production cash flows are also estimated in constant money terms (Mallinson, 1994).

The main criteria described by Mallinson (1994) for defining returns on an investment as represented by the cash flows are as follows:

- **Payback Period** indicates the time required to recoup the initial investment capital. Thereafter, average returns per annum on the investment are calculated. This method does not take into account the time value of money.
- **Net Present Value (NPV)** and the associated measures of Present Value Ratio and Potential Capital Gain. The Present Value of each of the cash flows, that is, their would be value at year 0, may be computed by compoundly discounting successive cash flows by some specified discount rate. The NPV is then the sum of the present values of all of the cash flows so discounted. Should the NPV be positive, then the Internal Rate of Return (IRR) of the investment alternative exceeds the chosen discount rate. Should the NPV be negative, the reverse is true.

- **Discounted Cash Flow (DCF)** or **Internal Rate of Return (IRR)** is that rate of discount that causes the negative and positive cash flow to be exactly equal and thus to cancel. Thus, if the NPV equals zero, the discounting rate is equal to the IRR.
- **Wealth Growth Rate (WGR)** and **External Rate of Return (ERR)** both require that the cost of capital to the investor and the reinvestment rate available to the investor be known, before either can be calculated. Returns are computed assuming that cash surplus generated from the investment is reinvested at the investor's average investment rate.

2.5.2. Revenues, costs and cash flows

Cash flow is the difference between revenues and costs for a specific time period. For evaluation purposes, an annual period is suitable and usually used.

Thus: **Annual Cash Flow = Annual Revenue – Annual Costs**

The cash flow estimates form the basis of any feasibility, sensitivity and risk assessments and determine the quality of the results.

Revenue accrues mainly from mineral or metal product sales. Costs may be distinguished as capital expenditure, operating costs and taxation payments. Capital expenditure is very much a function of the scale of the venture and both capital expenditure and operating costs are complex functions of the geographical and geological parameters (Mallinson, 1994).

In order to measure the economic performance of various investment alternatives, the cash flows must be expressed in constant money terms. This means that neither prices nor costs are escalated over the cash flow period, but are held constant at year 0 value. This method of ignoring both cost escalation and commodity price inflation may lead to slightly distorted values of calculated returns, especially if the rate of price inflation differs from the rate of cost escalation. For this reason, it is more correct to calculate the cash flows in current money terms, allowing for inflation and escalation, and then to deflate the resulting cash flows to reflect the various cash flows in real money terms, in other words, in the same terms as the year of inception (Mallinson, 1994).

2.5.3. Calculating NPV

The fundamental value of a project is determined by discounting future cashflows to determine the NPV. For a mining project there is an initial capital expenditure to develop the project and then net cashflows each year from the sale of a commodity. It is therefore possible to determine the internal rate of return (IRR) provided by the project by solving the following equation:

$$\text{IRR} = i \text{ when } \sum_{0}^{n} \frac{CF_n}{(1+i)^n} = 0$$

There is no direct method of solving for i, only by trial and error in selecting different interest rates until the NPV equals zero.

If the investor knows the appropriate discount rate (i), the NPV can be calculated using the following formula:

$$\text{NPV} = \sum_{0}^{n} \frac{CF_n}{(1+i)^n}$$

If the NPV has a positive value, the cashflow is capable of providing a return higher than the discount rate. If the NPV is negative, the cashflow will not be able to provide the return required by the investor's discount rate (Rudenno, 1998).

Net Present Value (NPV) is described by McLaney (2000) as a logical way of assessing investment opportunities, because it possesses the following attributes:

- It is directly related to the objective of maximisation of shareholders' wealth.
- It takes full account of the timing of the investment outlay and of the benefits, i.e. the time value of money is properly reflected. Stated differently: NPV properly takes the cost of financing the investment into account.
- All relevant, measurable financial information concerning the decision is taken into account.
- It is practical and easy to use and gives clear and unambiguous signals to the decision maker.
- The appropriate discount rate would reflect and account for the risk profile of the investment.

3. The baseline NPV

3.1. Simulation of the dataset

The dataset used in this study was generated with a simulation. It is based on a deepwater orebody as described in section 2.1.2. It is typical of the style of mineralisation found in the Atlantic 1 concession in waters deeper than 100 metres. Existing data from the Atlantic 1 deposit were used to generate a theoretical dataset. The reason for following this approach was to disguise the dataset so that confidentiality can be protected, but without sacrificing the distribution found in this style of deposit. Rectangular areas were selected to create 3 groups of data. The data was moved out of its original location so that the 3 groups are adjacent. These 3 groups represent different geological zones (Figure 9).

A sequential gaussian simulation was used as described by Deutsch & Journel (1998) to create the theoretical deposit. Multivariate gaussian models are well understood, have a record of successful application and are also the most straightforward algorithm for generating realisations (Deutsch & Journel, 1998). In order to create the deposit, the positively skewed input data had to be transformed to a normal distribution. Semi-variograms were constructed and the simulation parameters were established according to which the simulation was performed. The data were then back-transformed and the stones were seeded to create the simulated deposit seen in Figure 9. The simulated deposit consists of 1,887,798 stones in an area of 6.2 km^2.

3.2. Generation of sampling data for the NPV

A dataset was created that consists of 100×100m blocks that can be 'mined' from the theoretical deposit. This dataset reflects the true content of each mining panel. The simulated deposit in Figure 9 was then "sampled" using various parameters to generate sampling data that was used to generate

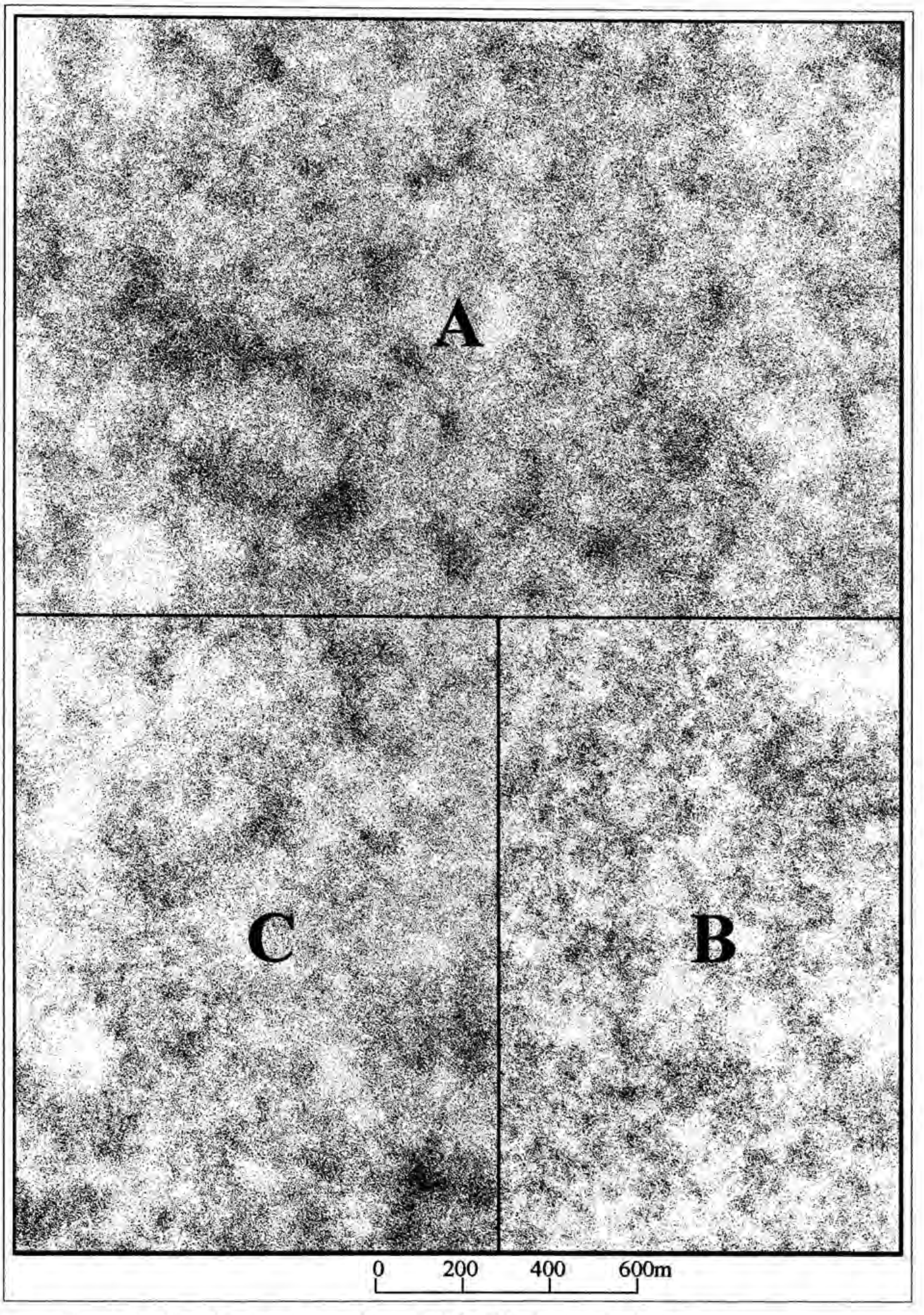

Figure 9. The simulated deposit created for this study. Each dot represents a diamond.

estimates. The estimates were then utilised to carry out extraction planning for the NPV model.

3.3. Elements of the baseline NPV

The philosophy behind the calculation of the baseline NPV, was to keep the model as uncomplicated as possible, but at the same time realistic. No attempt was made to exhaustively cover all the aspects that might arise in the real world. Instead the focus was on analysing generic situations that are commonly encountered in the style of mining venture on which the model was based.

The baseline NPV is displayed in Table 1. The baseline has a positive NPV and therefore the project could be considered feasible. The NPV = 0 at a discount rate of 25.55%, the Internal Rate of Return (IRR). The payback period is 6 years.

The elements of the NPV model will now be discussed.

3.3.1. Time

The cash flows were calculated on a yearly basis for eight years. The decision to use eight years instead of the traditional ten was based on the fact that the relatively high discount rate (20%) might cause distortion by contributions after eight years.

3.3.2. Discount rate

A discount rate of 20% was selected to reflect the relatively high risk for investors in this type of venture.

Table 1. Baseline Net Present Value (NPV) calculation.

	Year	0	1	2	3	4	5	6	7	8
Gross Revenue		0	533,020,201	431,015,197	365,084,654	387,762,839	371,877,100	321,688,079	339,558,634	322,501,778
Sampling and Survey		37,685,000								
Vessel Capital Outlay		700,000,000								
Operating Cost			109,500,000	109,500,000	109,500,000	109,500,000	109,500,000	109,500,000	109,500,000	109,500,000
Inport Cost					20,000,000			20,000,000		
Overheads			36,500,000	36,500,000	36,500,000	36,500,000	36,500,000	36,500,000	36,500,000	36,500,000
Total Costs		737,685,000	146,000,000	146,000,000	166,000,000	146,000,000	146,000,000	166,000,000	146,000,000	146,000,000
Income Before Tax		-737,685,000	387,020,201	285,015,197	199,084,654	241,762,839	225,877,100	155,688,079	193,558,634	176,501,778
Amortisation		0	221,305,500	184,421,250	147,537,000	110,652,750	73,768,500			
Taxable Income After Amortisation			165,714,701	100,593,947	51,547,654	131,110,089	152,108,600	155,688,079	193,558,634	176,501,778
Tax	30%	0	49,714,410	30,178,184	15,464,296	39,333,027	45,632,580	46,706,424	58,067,590	52,950,533
Income After Tax		-737,685,000	337,305,790	254,837,013	183,620,358	202,429,813	180,244,520	108,981,655	135,491,044	123,551,245
Discount Rate	**20%**	1.00	1.20	1.44	1.73	2.07	2.49	2.99	3.58	4.30
Cashflows		-737,685,000	281,088,159	176,970,148	106,261,781	97,622,402	72,436,230	36,497,736	37,813,064	28,734,071
NPV	**99,738,590**		-456,596,841	-279,626,694	-173,364,912	-75,742,510	-3,306,280	33,191,455	71,004,519	99,738,590

3.3.3. Taxation

The tax rate of 30% was used as this is the current rate for companies in South Africa.

3.3.4. Amortisation

Various mechanisms exist whereby capital expenditure is written off against profits in the industry. These are usually decided upon with regards to what suits the project best in agreement with the receiver of revenue (Hayes, pers. com.). Amortisation over a 5 year period on a sliding scale was selected as set out below:

Year 1	30%
Year 2	25%
Year 3	20%
Year 4	15%
Year 5	10%

3.3.5. Capital expenditure

- Cost of mining vessel: R700 million.
- Geosurvey cost: R1,085,000. This was calculated at a rate of 1km^2 per day at a cost of R155,000 per day and then multiplied with the area to be covered (7km^2).
- Sampling cost: R36,6 million. This was calculated at a rate of 20 samples per day at a cost of R300,000 per day. A 50×50m sample spacing and a 10m^2 sample support size were assumed which would result in total number of samples being 2,436 acquired in 121.8 (rounded to 122) days.

3.3.6. Operating expenditure

- Operating cost of mining vessel: R300,000 per day.
- Overheads: R100,000 per day.
- Inport cost: R20 million every 3 years for a 30 day inport refurbishment.

3.3.7. Gross revenue

The stones produced per year were calculated as discussed under the extraction strategy section. Revenue was calculated using a selling price of R2,300 per stone.

3.3.8. Extraction strategy

For the economic extraction of a block, the following parameters must be considered:

- Grade (stns/m^2),
- Mining recovery, stated as a percentage of the total grade content, and
- Mining rate (m^2/day).

Mining recovery and mining rate are both factors that are influenced by the combination of geology and mining tool. To obtain the extractable grade of a block, the true or stated grade must be factorised with a mining recovery factor. Applying the mining rate to the factorised grade, one is then able to calculate the amount of stones that can be extracted per day, from which, in turn, can be calculated a cash flow utilizing the income per stone/unit. The following factors were allocated to the simulated deposit:

Actual Mining Recovery		Actual Mining Rate	
Zone A	90%	Zone A	1500 m^2/day
Zone B	80%	Zone B	1800 m^2/day
Zone C	70%	Zone C	2000 m^2/day

During the sampling process it is not only the grade that is determined, but also rate and recovery predictions.

The estimated grade, rate and recovery figures were used to generate an extraction plan. The extractable grade was calculated from which the stones per day and the time (in days) that it would take to extract a block were in turn calculated. Instead of a cut-off grade, a cut-off stone rate of 300 stones/day, was applied. The blocks were then grouped into mining panels of 9 adjacent blocks as illustrated in Figure 10. The total stone content, as well as the time it would take to extract each panel, was calculated utilizing only the contribution of blocks with a stone per day value above stone cut-off.

Panels were ranked from highest to lowest, based on the stone rate (stones/day) for each panel. This ranking was then used to derive a sequence of extraction for the panels. Once the sequence was established, the number of stones that would be recovered in each year could be calculated and used to determine gross revenue. In the baseline NPV, the estimated and the actual recoveries were the same. In other words, for the baseline NPV, the actual content was used as the 'estimate', thereby representing the scenario of the perfect estimate. For the other NPV scenarios, the estimate was used to select contributing blocks and to sequence the panels. However, gross revenue was calculated on the basis of the actual recoveries, thereby illustrating the impact of the estimate variations on the baseline NPV. The NPV construction and extraction strategy calculations are discussed in more detail in Annexure 1.

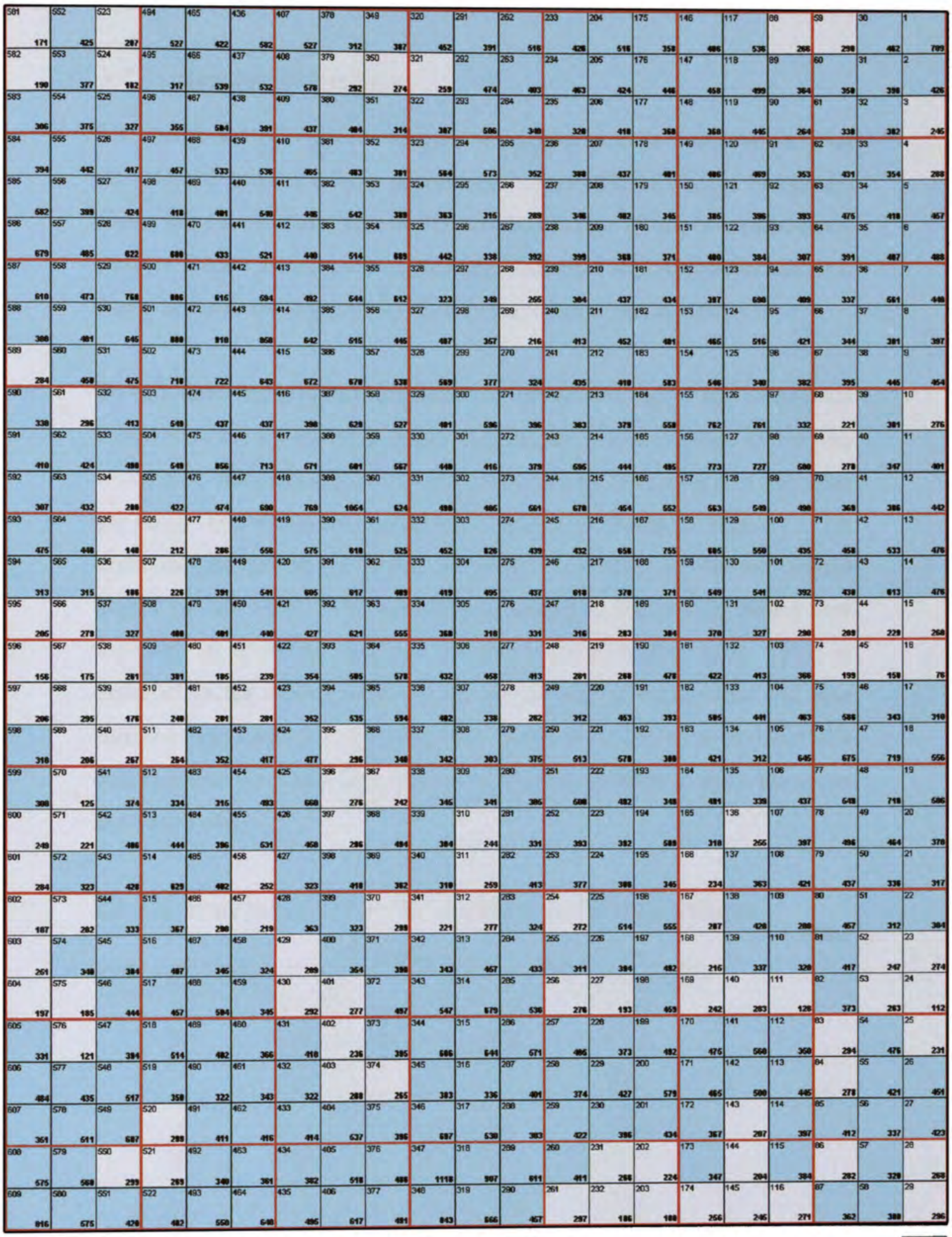

Figure 10. The simulated block model showing the 100×100m mining blocks grouped into mining panels (red outline) with contributing blocks indicated in light blue. The block number and stones per day are also indicated.

3.4. Sensitivity analysis

The model and NPV for the baseline is considered to be fairly robust. Not only does it possess a large positive NPV, but 82% of the ore blocks are above the cut-off stone rate of 300 stones per day. This is a particularly positive feature as it avoids the problems and risks experienced in marginal deposits due to the fragmentary distribution of payable ore blocks.

To gain some insight into the impact that variation of capital and fixed costs would have on the model, a sensitivity analysis was performed by varying vessel cost, sampling and survey (exploration) cost, operating cost and price per stone. The results are displayed in Table 2 and plotted in Figure 11. As expected, changes in the selling price per stone have by far the biggest impact on the NPV. The next biggest impact is the mining vessel cost followed by operating cost and sampling and geosurvey cost. This is expected as the mining vessel is the largest capital expenditure and also carries a large amount of operating cost. It can thus be concluded that of the cost elements discussed here, the model is most sensitive to price per stone and vessel cost.

Table 2. Variation of NPV to the variation of certain cost elements.

Vessel Cost

% Change in Cost	Cost	NPV	% Change in NPV
25	875,000,000	-41,037,837	-141%
20	840,000,000	-12,882,552	-113%
15	805,000,000	15,272,734	-85%
10	770,000,000	43,428,019	-56%
5	735,000,000	71,583,305	-28%
0	**700,000,000**	**99,738,590**	**0%**
-5	665,000,000	127,893,876	28%
-10	630,000,000	156,049,161	56%
-15	595,000,000	184,204,447	85%
-20	560,000,000	212,359,732	113%
-25	525,000,000	240,515,017	141%

Sampling & Survey (Exploration)

% Change in Cost	Cost	NPV	% Change in NPV
25	47,106,250	92,159,791	-8%
20	45,222,000	93,675,550	-6%
15	43,337,750	95,191,310	-5%
10	41,453,500	96,707,070	-3%
5	39,569,250	98,222,830	-2%
0	**37,685,000**	**99,738,590**	**0%**
-5	35,800,750	101,254,350	2%
-10	33,916,500	102,770,110	3%
-15	32,032,250	104,285,870	5%
-20	30,148,000	105,801,630	6%
-25	28,263,750	107,317,390	8%

Operating Cost

% Change in Cost	Cost	NPV	% Change in NPV
25	136,875,000	26,209,015	-74%
20	131,400,000	40,914,930	-59%
15	125,925,000	55,620,845	-44%
10	120,450,000	70,326,760	-29%
5	114,975,000	85,032,675	-15%
0	**109,500,000**	**99,738,590**	**0%**
-5	104,025,000	114,444,505	15%
-10	98,550,000	129,150,420	29%
-15	93,075,000	143,856,335	44%
-20	87,600,000	158,562,250	59%
-25	82,125,000	173,268,165	74%

Price per Stone

% Change in Cost	Cost	NPV	% Change in NPV
25	2,875	374,265,503	275%
20	2,760	319,360,121	220%
15	2,645	264,454,738	165%
10	2,530	209,549,355	110%
5	2,415	154,643,973	55%
0	**2,300**	**99,738,590**	**0%**
-5	2,185	44,833,207	-55%
-10	2,070	-10,072,175	-110%
-15	1,955	-64,977,558	-165%
-20	1,840	-119,882,941	-220%
-25	1,725	-174,788,323	-275%

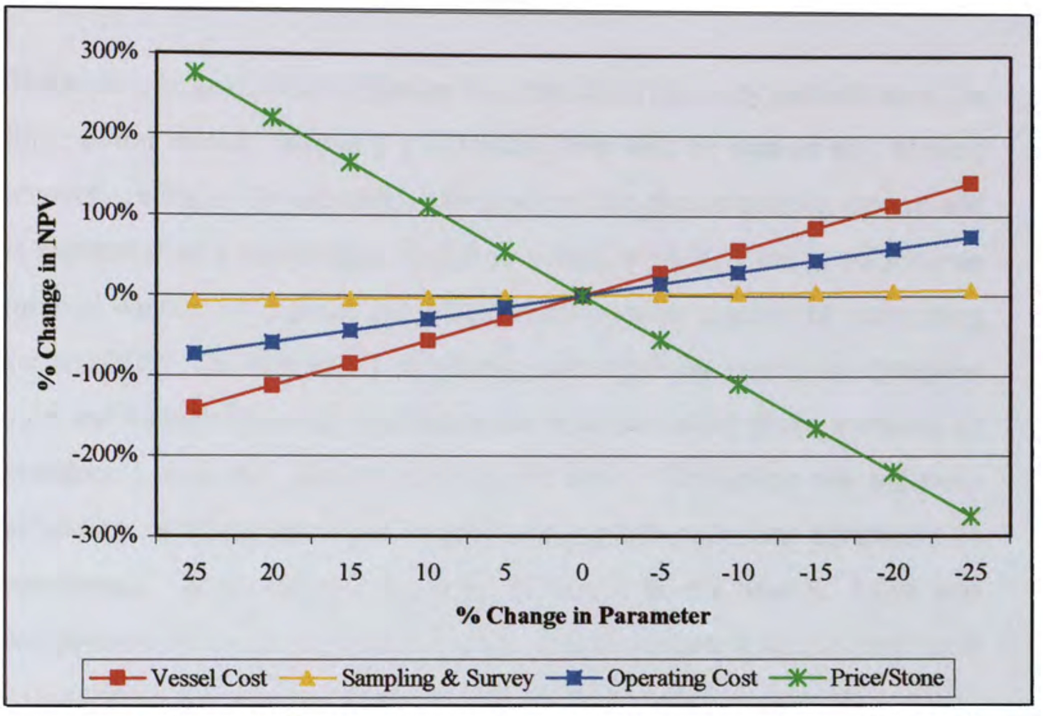

Figure 11. Sensitivity plot of variation in NPV versus the variation of certain cost elements.

4. Extraction efficiency and rate estimations

There are a large number of factors that can affect recovery performance. In this section mining recovery and mining rate will be looked at. Mining recovery refers to the extraction efficiency of the chosen mining system and is expressed as a percentage. In other words, what percentage of the true mineral content of a given ore block is the system capable of recovering successfully? The influences on mining recovery in the marine environment vary and include the nature and operation of the recovery plant, accuracy of positioning and the type of mining tool used. Predicting the recovery efficiency of plant and tool is fairly easy when only the equipment is considered. However, the geology in terms of the nature, habit and composition of the orebody, has a major impact on how both tool and plant will perform. In practice, as much relevant information as possible needs to be collected in the exploration phase to make accurate recovery predictions in the planning phase.

In a similar fashion the geology impacts on the mining rate and inaccurate rate predictions can have grave consequences, as has already been mentioned in section 2.4. Rate is traditionally measured in the marine mining environment as area covered in a given period of time (m^2/day).

4.1. Sensitivity of NPV to mining recovery predictions

As has already been stated, the model consists of three geological zones, each of which has a different mining recovery percentage of which the system is capable:

Mining recovery – Zone A	90%
Mining recovery – Zone B	80%
Mining recovery – Zone C	70%

The above percentages are the actual recovery efficiencies that will be achieved in each geological zone. However, extraction planning is based on the prediction of mining recovery percentage. Six scenarios with regards to mining recovery prediction were used in the NPV calculation, the cut-off rate was applied and the blocks were sequenced for extraction. This was done to see how the NPV reacts to extraction plans based on recovery factor predictions of varying accuracy. They were as follows:

- 100% - No mining recovery factors applied (100% recovery expected)
- 90% - All zones were factored according to Zone A
- 80% - All zones were factored according to Zone B
- 70% - All zones were factored according to Zone C
- 60% - All zones were given a single underestimated factor of 60%
- 50% - All zones were given a single underestimated factor of 50%

The NPV for the various scenarios can be seen in Table 3 and the percentage change in NPV for the six scenarios were plotted in Figure 12.

Table 3. NPV of the various predicted recovery scenarios.

Predicted Recovery	NPV	% Change in NPV
Baseline	**99,738,590**	**0%**
100% Scenario	68,575,364	-31%
90% Scenario	74,886,698	-25%
80% Scenario	85,151,492	-15%
70% Scenario	78,384,638	-21%
60% Scenario	-11,856,343	-112%
50% Scenario	-201,607,920	-302%

The least change in NPV is observed when the mining recovery prediction is equal to the actual mining recovery of Zone B (80%). This is due to the fact that the mining recovery of Zone B is the closest to the average for the whole deposit. The NPV drops above and below this figure. Even though the 80% scenario delivers the best NPV, it is still 15% below the baseline (which utilised a geological zone specific recovery factor) and illustrates the danger of not delineating geologically different areas and making correct recovery predictions based on the geology.

The effect of the low predicted recovery percentages is dramatic as large numbers of economic ore blocks are pushed below the cut-off and the life-of-mine is shortened. At 60% there are only enough blocks available to mine for 5 years and at 50%, for 3 years. The opposite, when mining recovery is overestimated could also have a potentially devastating effect on a project, especially if the project is marginal with fairly low actual mining recoveries. This would essentially constitute a global overestimation of the extractable grade content.

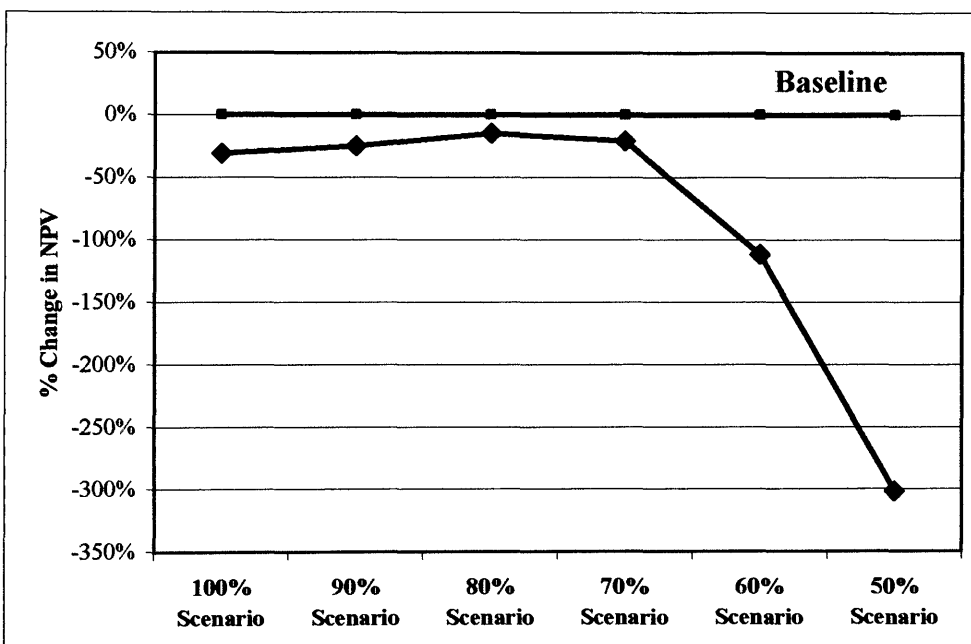

Figure 12. Sensitivity plot of variation in NPV for the various recovery predictions.

4.2. Sensitivity of NPV to mining rate predictions

The model contains a unique mining rate for each of the three geological zones. Nine scenarios with regards to mining rate prediction were used in the NPV calculation, the cut-off rate was applied and the blocks were sequenced for extraction. A uniform rate was applied over all the zones in each of the scenarios. Three scenarios utilised the rate of each of the zones and six scenarios were created by varying the rate at 10% intervals. An average rate for the three zones were calculated from which this variation took place. The mining rates and the NPV for each of the rates applied can be seen in Table 4. The rate in Table 4 was sequenced from high to low. Rate was plotted against the percentage change in NPV in Figure 13.

The NPV dropped as the predicted mining rate was moved up and down. It is apparent from Figure 13 that the NPV is less sensitive to overestimation of rate than underestimation. This is due to the same effect observed when mining recovery is underestimated whereby economic ore blocks are pushed below the cut-off rate.

Table 4. NPV of the various predicted rate scenarios.

Predicted Rate	(m^2/day)	NPV	% Change in NPV
30%	2,240	71,289,151	-29%
20%	2,068	73,470,961	-26%
Rate at C	2,000	77,331,818	-22%
10%	1,895	80,139,363	-20%
Rate at B	1,800	84,015,812	-16%
Baseline	1,723	**99,738,590**	**0%**
-10%	1,551	74,380,717	-25%
Rate at A	1,500	75,907,803	-24%
-20%	1,378	47,475,144	-52%
-30%	1,206	21,591,134	-78%

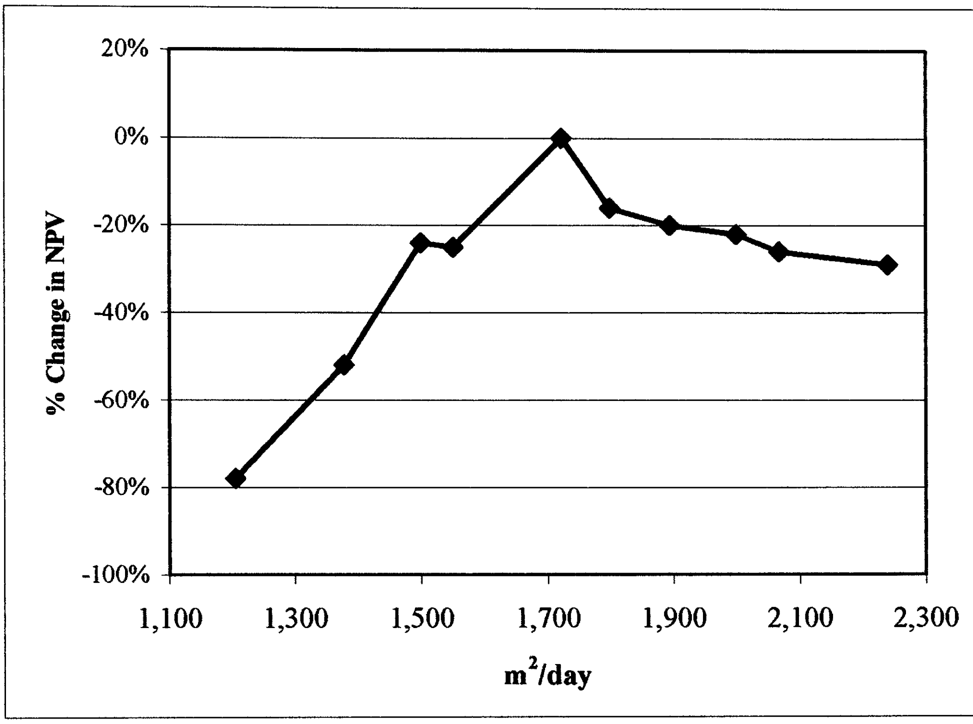

Figure 13. Sensitivity plot of variation in NPV for various mining rate predictions.

5. Estimation process influences on NPV

The objective of this section was to investigate the influence of the degree of error in the geostatistical estimation process on the NPV. More specifically, the effect of misfitting of the nugget effect and the range on the experimental semi-variogram, were investigated. The results from these semi-variograms were then used to generate an estimate for the NPV calculation.

5.1. Methodology

All the estimates produced in this section utilised a 50×50m sample spacing and a sample support size of $10m^2$. The methodology followed will be explained below by means of the 'best fit' experimental semi-variogram. The AESTIMATUS software package was used for the estimations. This package was written for De Beers and is an in-house application that uses algorithms developed by C.F. Prins.

The semi-variance ($\gamma^*(h)$) of the sample pairs were calculated using the following parameters:

Number of lags:	9
Lag distance:	50
Distance tolerance:	25
Angle tolerance:	22.5
Directions:	0; 45; 90; 135 and the Average.

The experimental semi-variogram was then plotted and a spherical model was fitted on the average, as illustrated in Figure 14. The sill, range and nugget were used in the kriging process to generate the estimate. A negative binomial kriging technique was used as it provides more accurate

h	$\gamma^*(h)$	Model
0	0	1.95
60.25	2.73	2.68
107.81	3.15	3.15
151.90	3.36	3.43
203.89	3.48	3.50
256.86	3.52	3.50
304.60	3.58	3.50
353.01	3.63	3.50
400.27	3.70	3.50

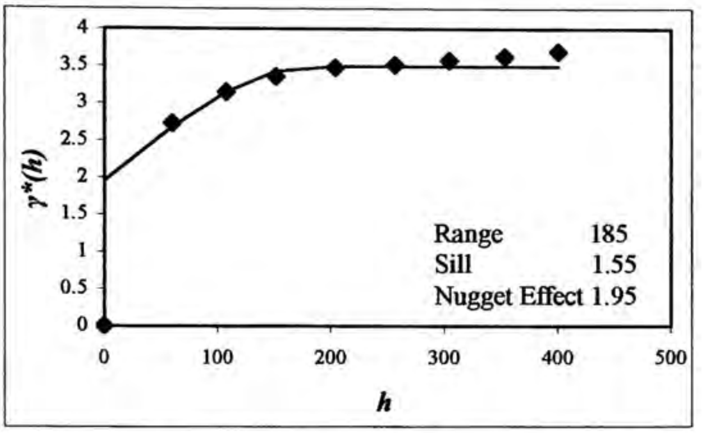

Figure 14. The experimental semi-variogram with a spherical model fitted for the 'best fit' scenario.

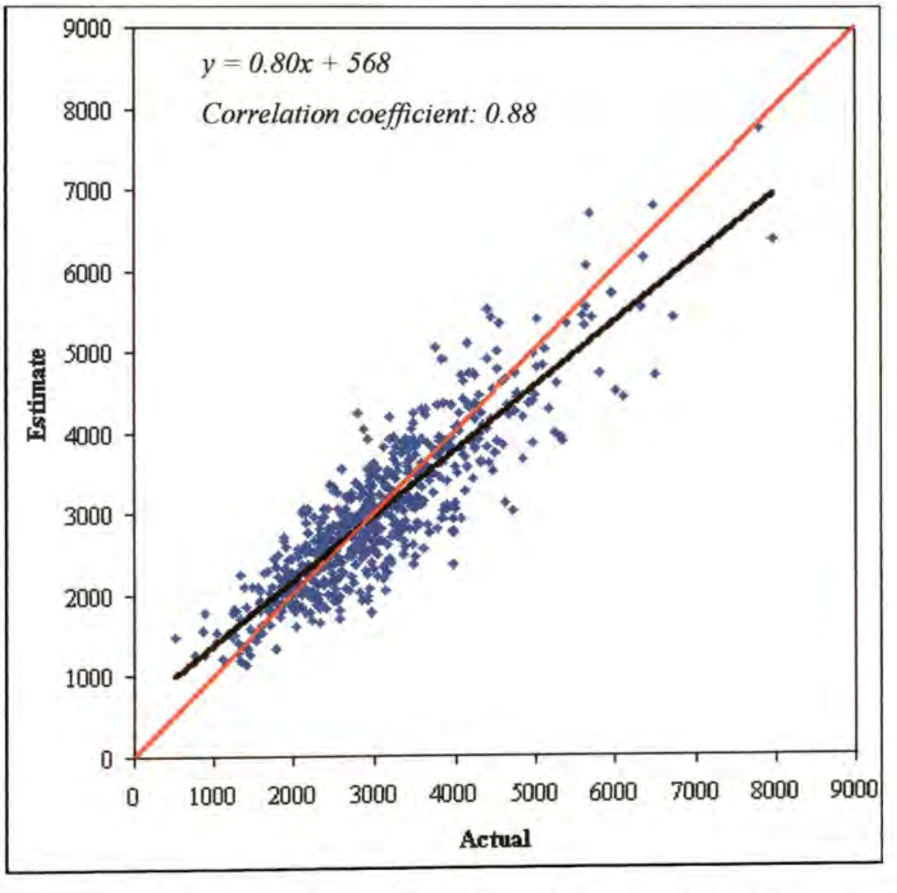

Figure 15. Correlation plot between the estimated and actual stone value for the 100×100m blocks for the 'best fit' semi-variogram model.

local block estimate than conventional linear kriging techniques in the style of deposit simulated for this study (Duggan, 1995). Kriging was performed using a sample support size of $10m^2$, a 3×3 krig plan, a 5×5 plan for the calculation of the rolling mean and discretisation of 12. A comparison between the estimates and the actual stone content can be seen in Figure 15. A fairly good correlation can be observed. The average of the estimates and the actuals remained the same (the average being located on the point on the graph where the red line and the black line, the linear regression, crosses). Note however that the estimates below the average are slightly overestimated and those above the average, slightly underestimated.

This scenario returned an NPV of R81,921,623 which is 18% below the baseline NPV.

5.2. Nugget effect variations

The same process was followed as stated in the previous section, but the nugget effect was not fitted properly onto the experimental semi-variogram. Instead the following scenarios were tested:

- Nugget effect was set at 0
- Nugget effect was understated by 50%
- Nugget effect was overstated by 50%
- Moving average (nugget effect at maximum, sill zero)

The experimental semi-variograms, as well as the correlation with actuals for these four scenarios are displayed in Figures 16 to 23. The sensitivity plot of variation in NPV for the various nugget effect variation scenarios can be seen in Table 5 and Figure 24. It can be deduced that misfitting the nugget effect on the semi-variogram does affect the NPV negatively, except

h	$\gamma^*(h)$	Model
0	0	0
60.25	2.73	1.65
107.81	3.15	2.71
151.90	3.36	3.34
203.89	3.48	3.50
256.86	3.52	3.50
304.60	3.58	3.50
353.01	3.63	3.50
400.27	3.70	3.50

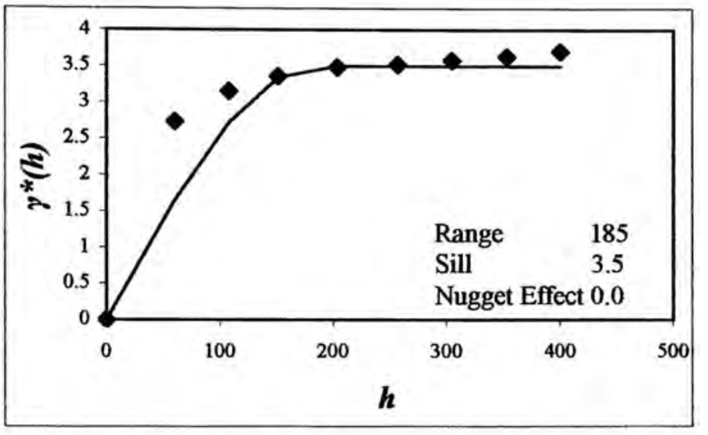

Figure 16. The experimental semi-variogram with a spherical model fitted for the zero nugget effect scenario.

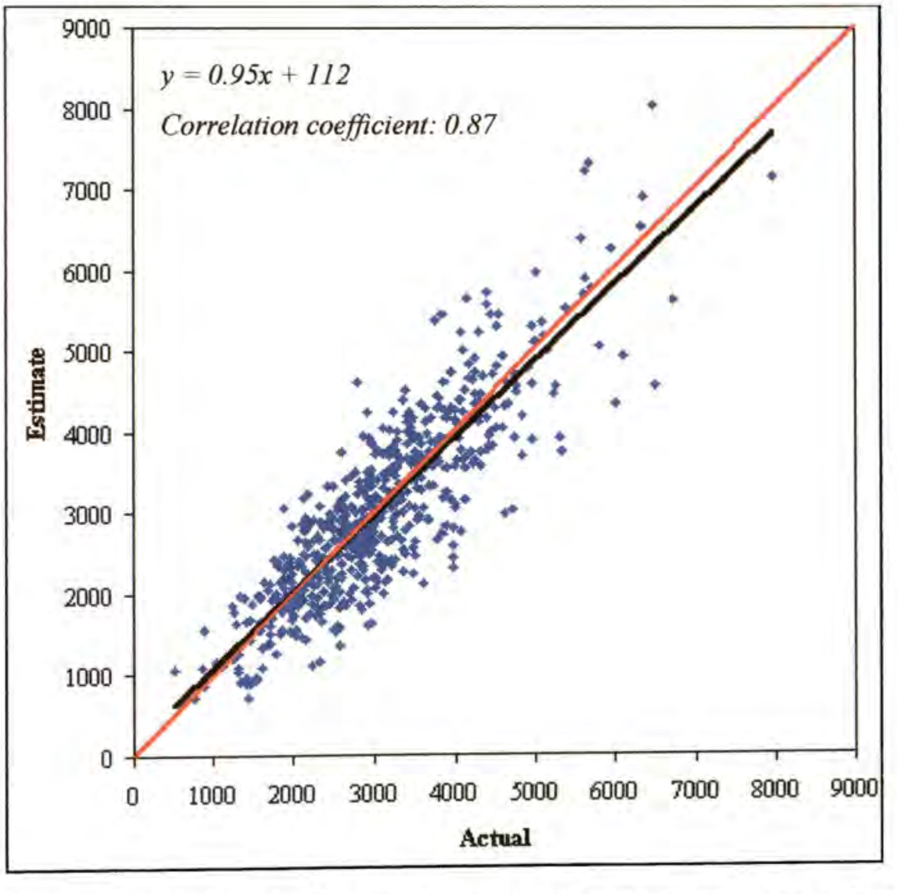

Figure 17. Correlation plot between the estimated and actual stone value for the zero nugget effect scenario.

h	$\gamma^*(h)$	Model
0	0	0.98
60.25	2.73	2.16
107.81	3.15	2.93
151.90	3.36	3.39
203.89	3.48	3.50
256.86	3.52	3.50
304.60	3.58	3.50
353.01	3.63	3.50
400.27	3.70	3.50

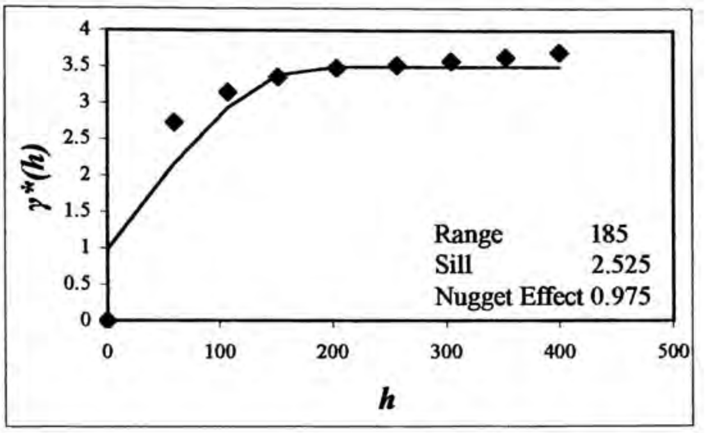

Figure 18. The experimental semi-variogram with a spherical model fitted for the nugget effect understated by 50% scenario.

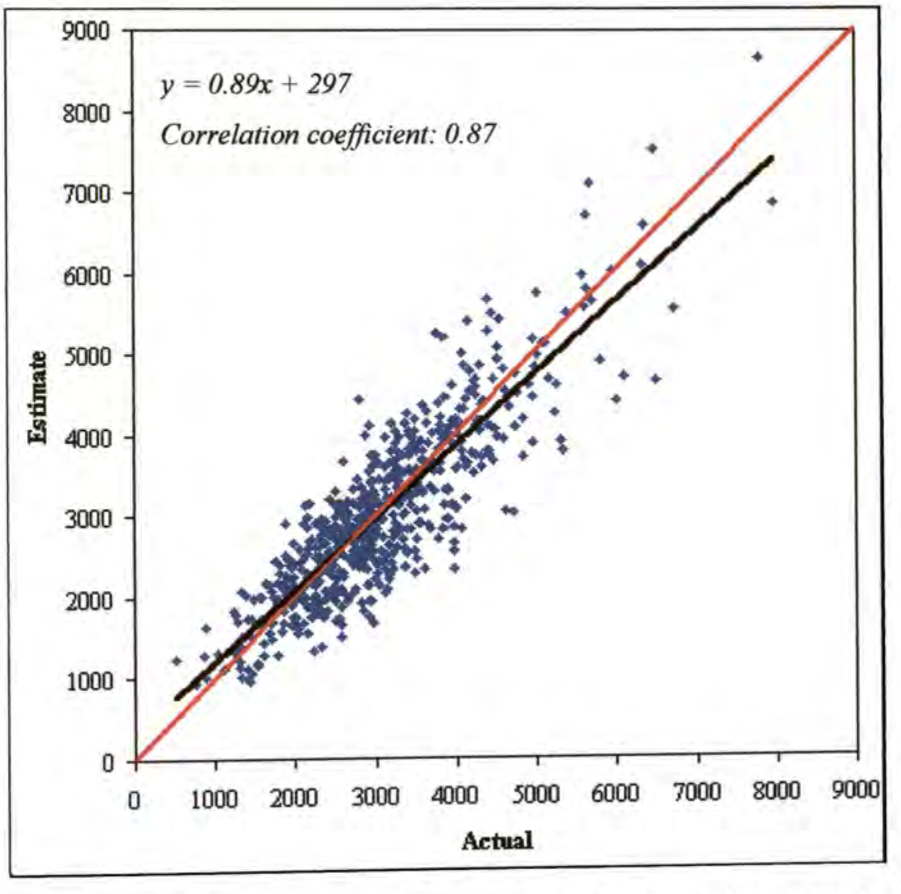

Figure 19. Correlation plot between the estimated and actual stone value for the nugget effect understated by 50% scenario.

h	$\gamma^*(h)$	**Model**
0	0	9.93
60.25	2.73	3.20
107.81	3.15	3.37
151.90	3.36	3.47
203.89	3.48	3.50
256.86	3.52	3.50
304.60	3.58	3.50
353.01	3.63	3.50
400.27	3.70	3.50

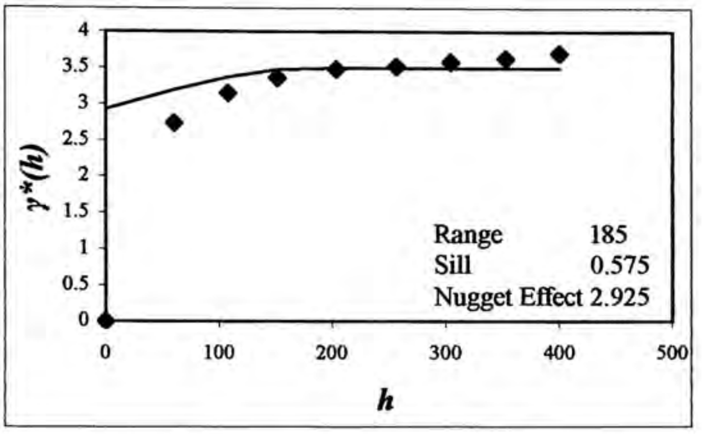

Figure 20. The experimental semi-variogram with a spherical model fitted for the nugget effect overstated by 50% scenario.

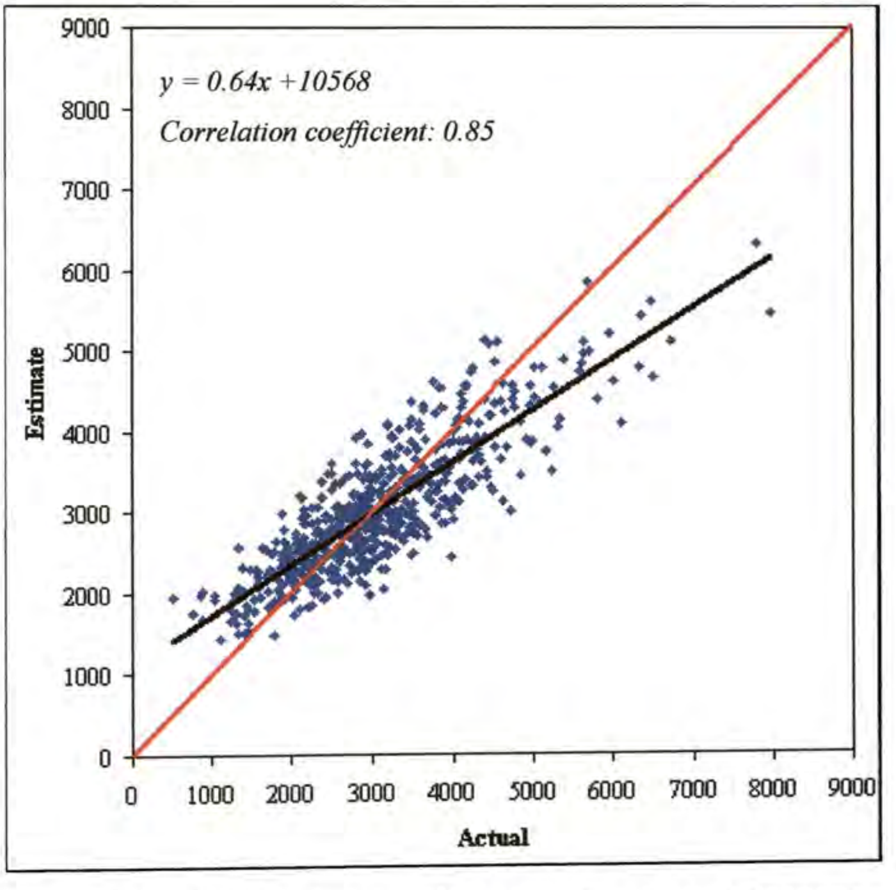

Figure 21. Correlation plot between the estimated and actual stone value for the nugget effect overstated by 50% scenario.

h	***γ*(h)***	**Model**
0	0	3.50
60.25	2.73	3.50
107.81	3.15	3.50
151.90	3.36	3.50
203.89	3.48	3.50
256.86	3.52	3.50
304.60	3.58	3.50
353.01	3.63	3.50
400.27	3.70	3.50

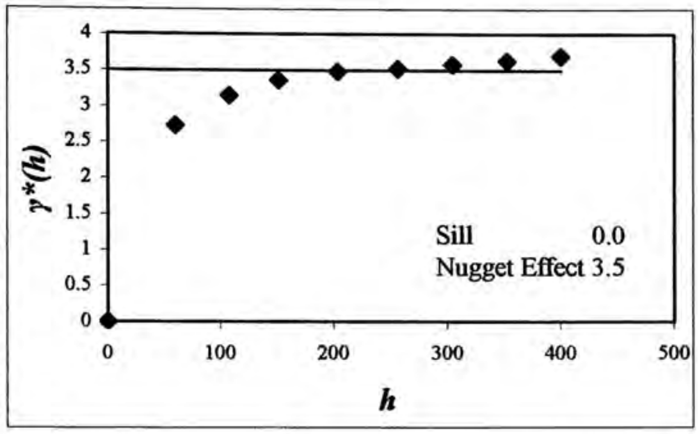

Figure 22. The experimental semi-variogram with a spherical model fitted for the moving average/maximum nugget effect scenario.

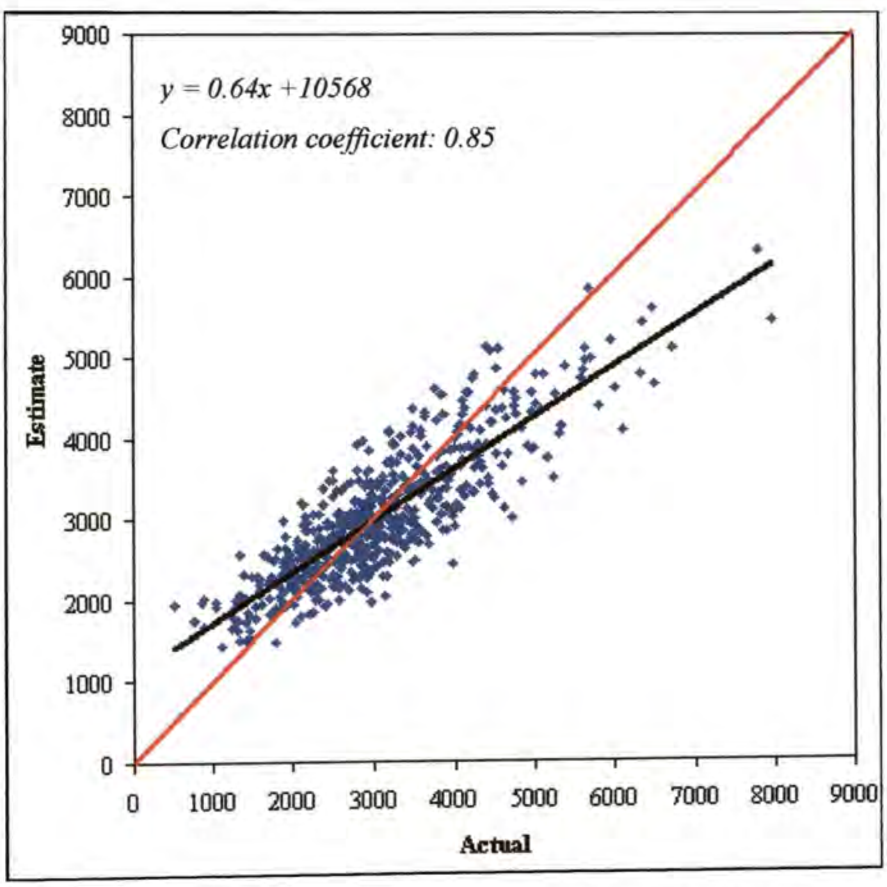

Figure 23. Correlation plot between the estimated and actual stone value for the moving average scenario.

for the nugget effect understated by 50% scenario, which delivered a better NPV than the best fit model. The better result achieved in this scenario, as we shall see later, is due to the fact that the simulation has a short range structure that is not detected by a 50×50m sample grid, which means that the best fit semi-variogram model is not necessarily the best semi-variogram model to describe the variance of the simulation. What is also noted is that the poorest NPV is achieved when a moving average is applied, illustrating that utilising geostatistics to generate estimates certainly makes more financial sense.

Table 5. NPV of the various nugget effect variation scenarios.

Nugget Effect Variation	NPV	% Change in NPV
Baseline	**99,738,590**	**0%**
Zero NE	78,760,100	-21%
NE under by 50%	89,231,939	-11%
Best fit	81,921,623	-18%
NE over by 50%	74,892,096	-25%
Moving Average	65,301,035	-35%

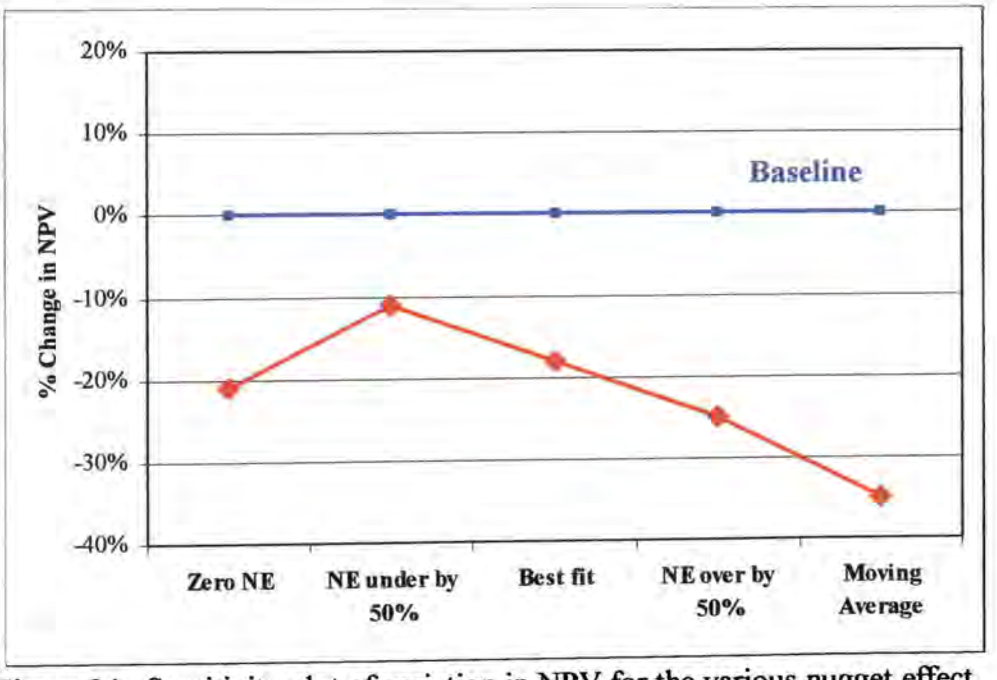

Figure 24. Sensitivity plot of variation in NPV for the various nugget effect variation scenarios.

5.3. Range variations

The same process was followed as described in section 5.1, but range was not fitted properly onto the experimental semi-variogram. Instead the following scenarios were tested:

- Range understated by 50%
- Range understated by 25%
- Range overstated by 25%
- Range overstated by 50%

The experimental semi-variograms, as well as the correlation with actuals for these four scenarios are displayed in Figures 25 to 32.

The sensitivity plot of variation in NPV for the range variation scenarios can be seen in Table 6 and Figure 33. Understating the range does have a negative effect on NPV, whilst overstating it delivers a slightly better NPV than that achieved with the best fit scenario. It is not clear what causes this phenomena.

With regards to NPV sensitivity it would appear that, in comparison with the nugget effect, misfitting the range has a fairly small impact on the NPV of the model.

h	$\gamma^*(h)$	Model
0	0	1.95
60.25	2.73	3.25
107.81	3.15	3.50
151.90	3.36	3.50
203.89	3.48	3.50
256.86	3.52	3.50
304.60	3.58	3.50
353.01	3.63	3.50
400.27	3.70	3.50

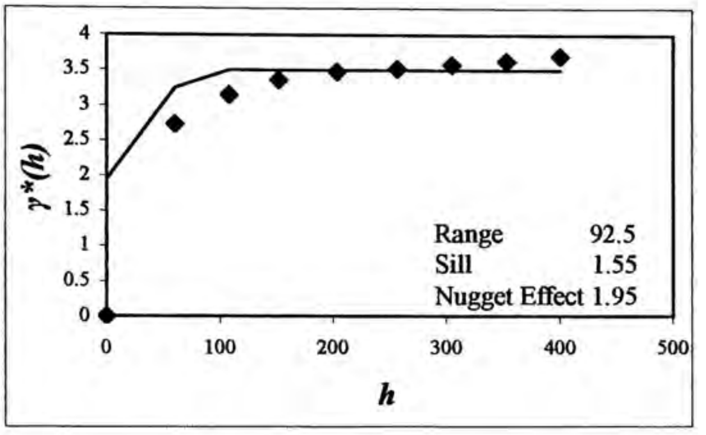

Figure 25. The experimental semi-variogram with a spherical model fitted for the range understated by 50% scenario.

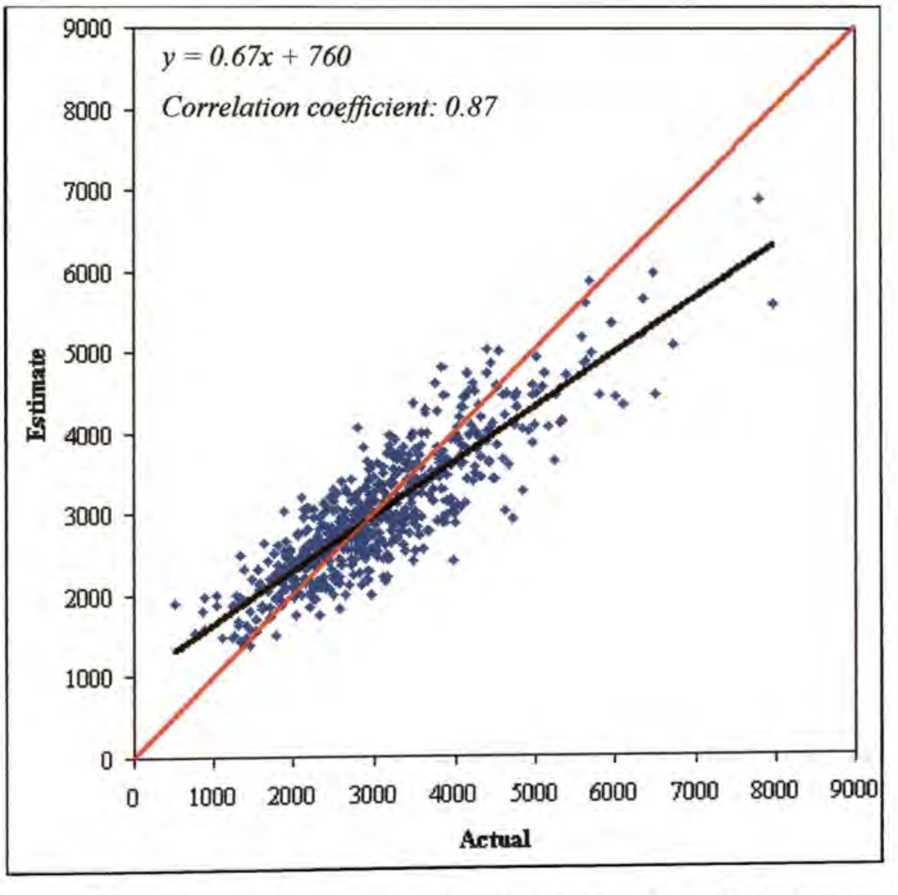

Figure 26. Correlation plot between the estimated and actual stone value for the range understated by 50% scenario.

h	$\gamma^*(h)$	Model
0	0	1.95
60.25	2.73	2.90
107.81	3.15	3.39
151.90	3.36	3.50
203.89	3.48	3.50
256.86	3.52	3.50
304.60	3.58	3.50
353.01	3.63	3.50
400.27	3.70	3.50

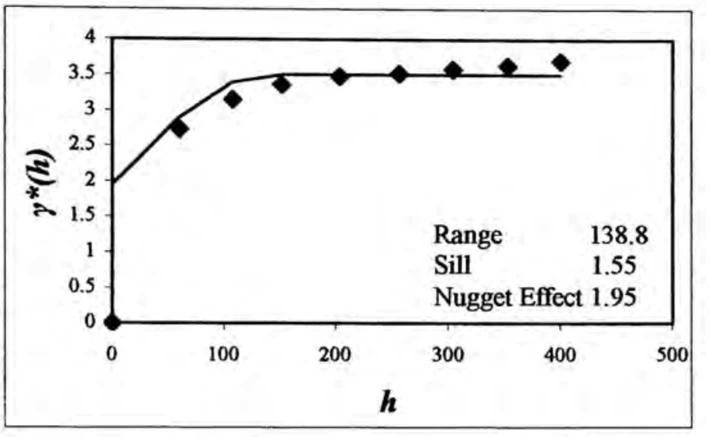

Figure 27. The experimental semi-variogram with a spherical model fitted for the range understated by 25% scenario.

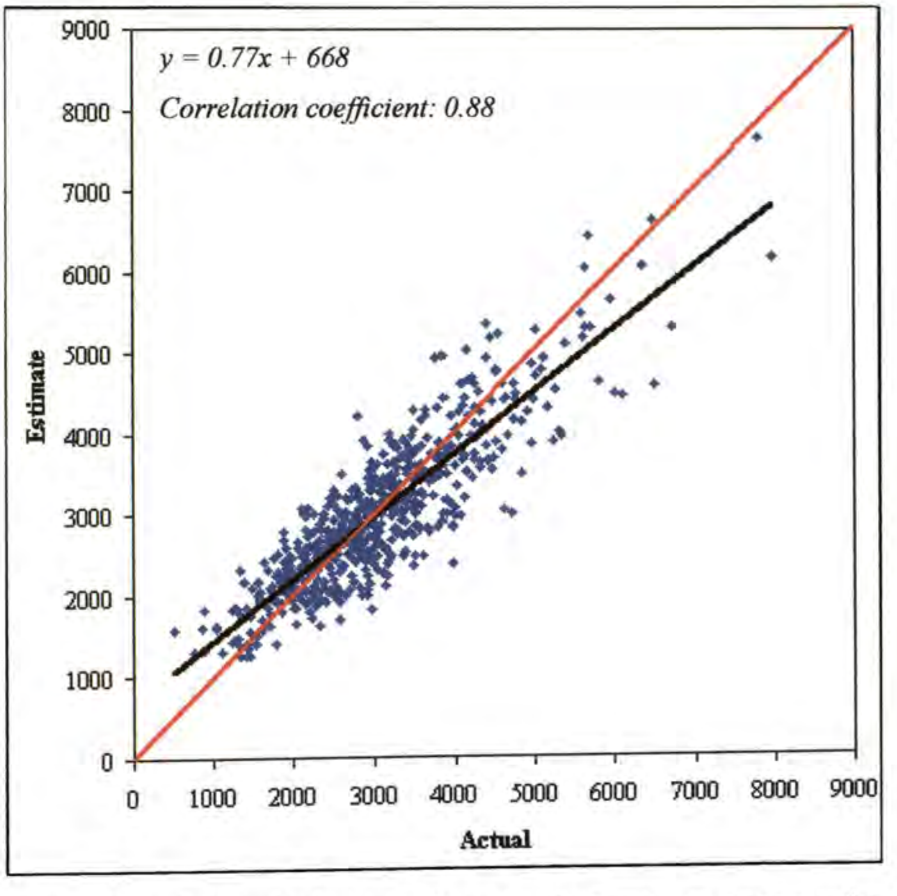

Figure 28. Correlation plot between the estimated and actual stone value for the range understated by 25% scenario.

h	$\gamma^*(h)$	Model
0	0	1.95
60.25	2.73	2.54
107.81	3.15	2.96
151.90	3.36	3.26
203.89	3.48	3.47
256.86	3.52	3.50
304.60	3.58	3.50
353.01	3.63	3.50
400.27	3.70	3.50

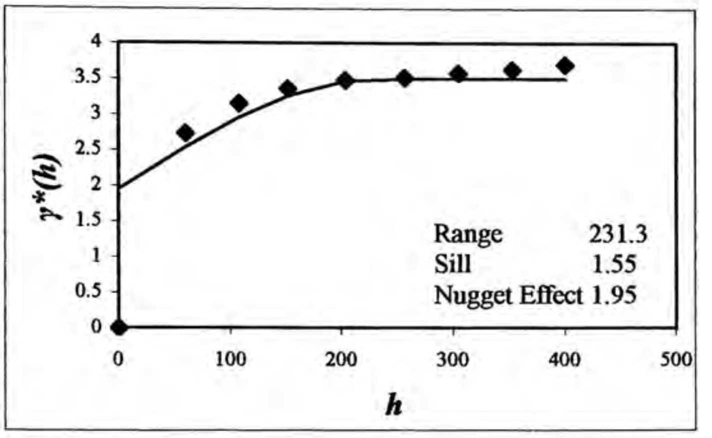

Figure 29. The experimental semi-variogram with a spherical model fitted for the range overstated by 25% scenario.

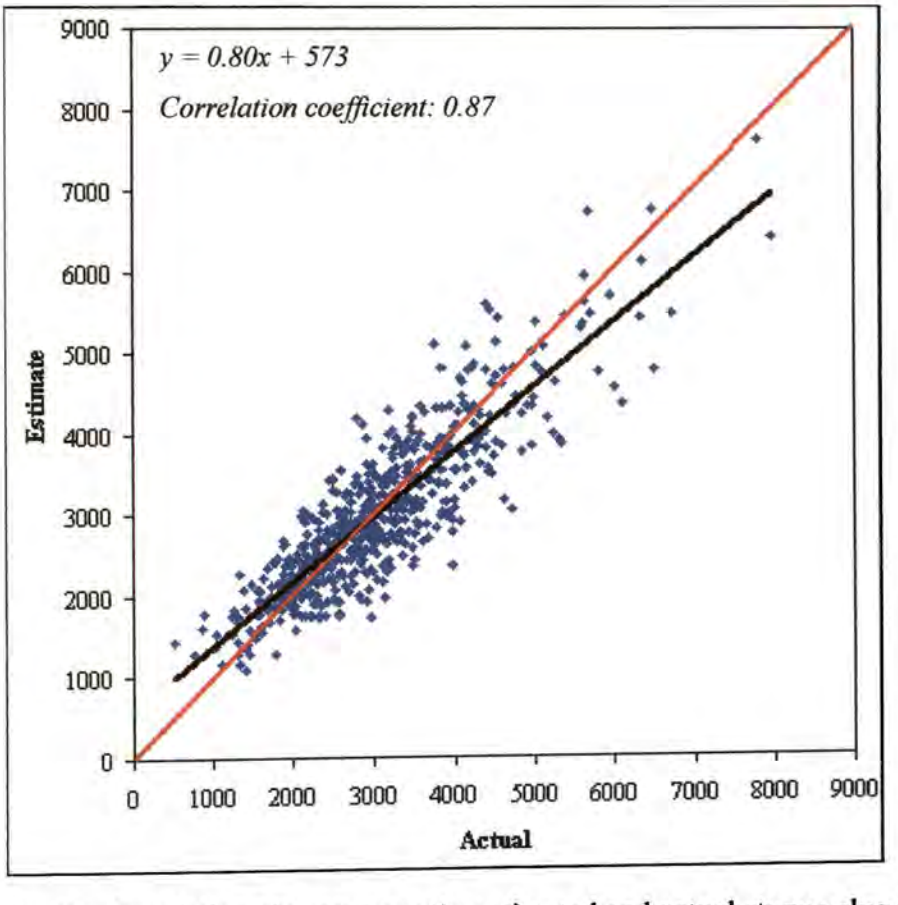

Figure 30. Correlation plot between the estimated and actual stone value for the range overstated by 25% scenario.

h	$\gamma^*(h)$	Model
0	0	1.95
60.25	2.73	2.45
107.81	3.15	2.81
151.90	3.36	3.10
203.89	3.48	3.35
256.86	3.52	3.49
304.60	3.58	3.50
353.01	3.63	3.50
400.27	3.70	3.50

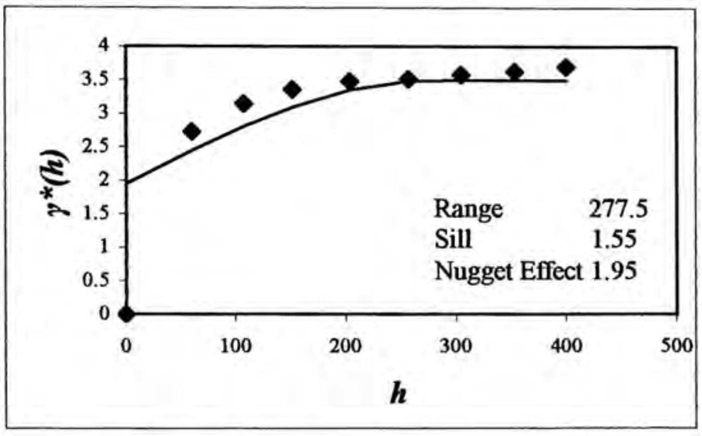

Figure 31. The experimental semi-variogram with a spherical model fitted for the range overstated by 50% scenario.

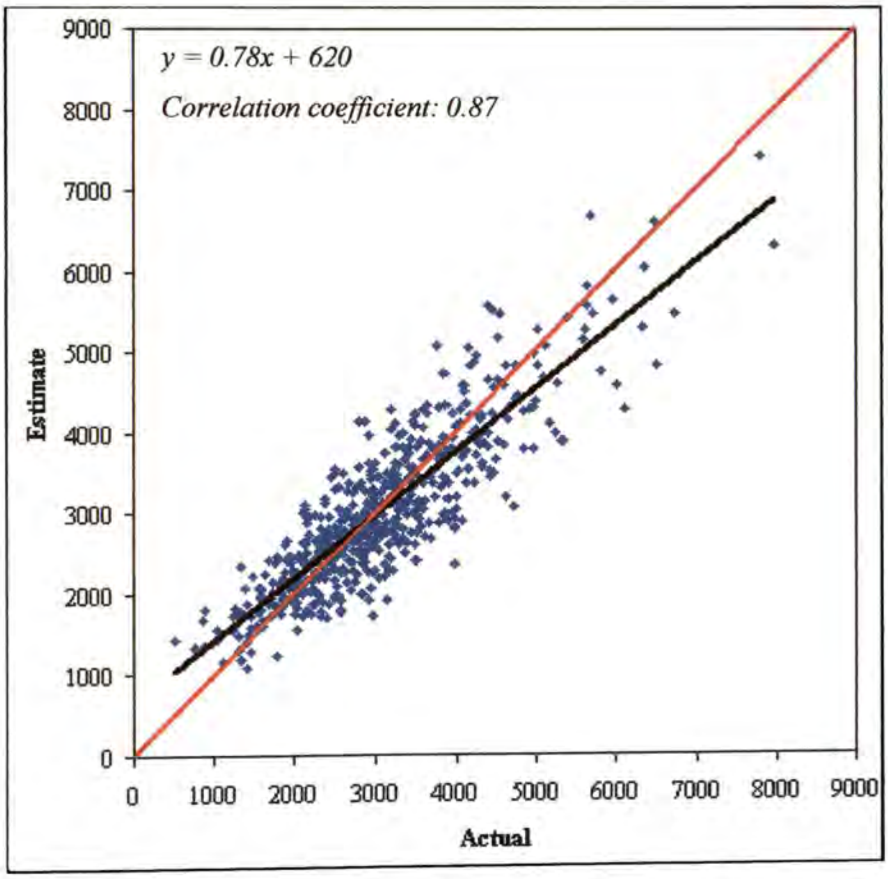

Figure 32. Correlation plot between the estimated and actual stone value for the range overstated by 50% scenario.

Table 6. NPV of the various range variation scenarios.

Range Variations	NPV	% Change in NPV
Baseline	**99,738,590**	**0%**
Range 50% smaller	75,937,151	-24%
Range 25% smaller	80,298,532	-19%
Best fit	81,921,623	-18%
Range 25% larger	83,492,903	-16%
Range 50% larger	84,446,093	-15%

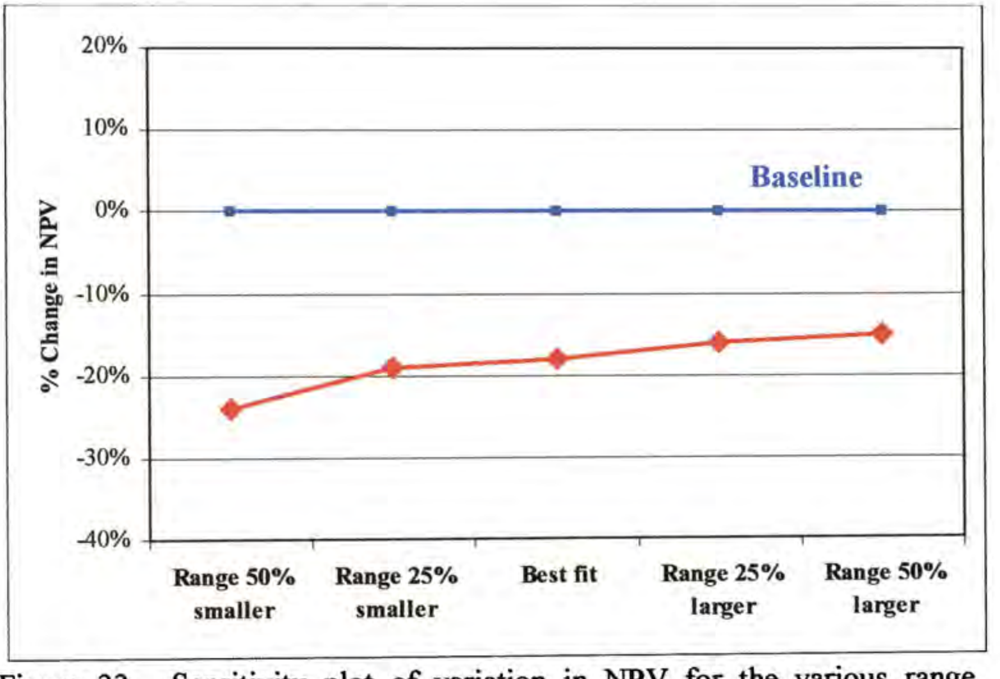

Figure 33. Sensitivity plot of variation in NPV for the various range variation scenarios.

6. Sample density and sample support size influence on NPV

6.1. Sample density

The correct sample spacing is probably one of the most important decisions because it has such a large impact on the cost aspect of the exploration phase. A balance needs to be found between keeping the sampling costs down and at the same time produce sufficient information for the purpose of estimation (Prins, 1991). In this section, the sensitivity of NPV to sample density for the model were investigated. The following scenarios were tested:

- 25×25m spaced sample grid (9744 samples)
- 50×50m spaced sample grid (2436 samples)
- 100×100m spaced sample grid (609 samples)
- 200×200m spaced sample grid (165 samples)

Table 7 shows the exploration cost that was returned by the model for the various sample density scenarios.

Table 7. The impact of sample density selection on the exploration cost.

Sample Grid	Exploration Cost	% Change in Exploration Cost in Comparison with the Baseline
25×25m	147,485,000	291%
50×50m	37,685,000	0%
100×100m	10,385,000	-72%
200×200m	3,785,000	-90%

The same methodology was followed as described in section 5.1. Correlation plots between the estimates and the actuals can be seen in Figures 35 to 37 for the 25×25m, 100×100m and 200×200m sample grids. The result for the 50×50m sample grid scenario can be seen in Figure 15.

It can be seen from the correlation plots that the accuracy of the estimate deteriorates as the sample spacing is opened up, but that the difference between the 100×100m and 200×200m sample grids are not that noticeable as between the other grids. The 25×25m grid delivered a far superior estimation than any of the others. This is due, not only to the closely spaced sampling but also to the improved experimental semi-variogram that was produced from this dataset. As can be seen in Figure 34, the 25×25m grid detected a short range structure that enabled the fitting of two nested spherical models and bring down the nugget effect, thereby generating a better estimate.

h	***γ*(h)***	**Model**
0	0	1.86
30.15	2.34	2.33
53.92	2.65	2.67
75.95	2.89	2.92
101.98	3.08	3.10
128.45	3.2	3.23
152.33	3.29	3.32
176.52	3.35	3.40
200.15	3.39	3.45
226.46	3.42	3.48
252.79	3.45	3.48
276.61	3.46	3.48
300.81	3.48	3.48
325.64	3.51	3.48
351.21	3.52	3.48

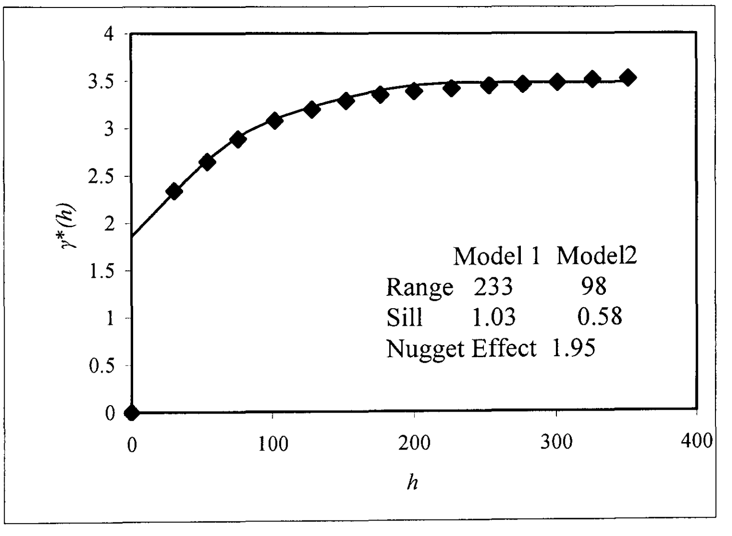

Figure 34. The experimental semi-variogram with two nested spherical models fitted for the 25×25m spaced sample grid.

Two sets of NPV's were calculated. The first set was generated using only the estimates without adjusting the sampling cost and in the second set, the sampling costs were adjusted accordingly. The resulting NPV's can be seen in Table 8 and Figure 38. If the sampling costs are not considered, it can be seen that the better estimates produced from higher sample densities lead to

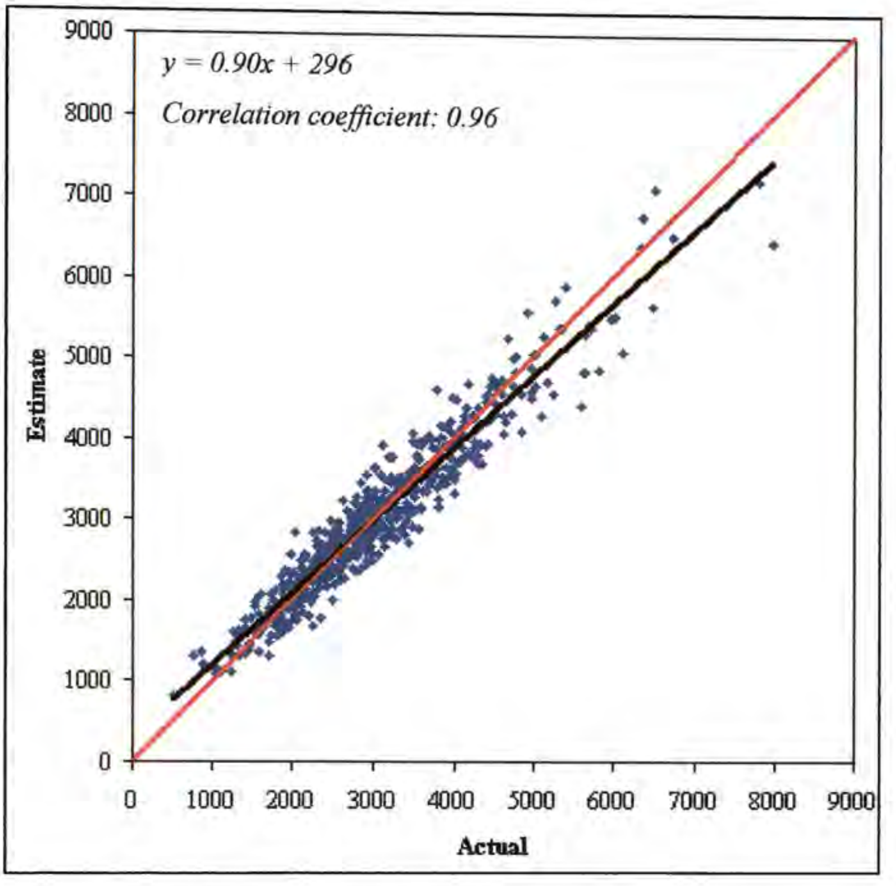

Figure 35. Correlation plot between the estimated and actual stone value for the 25×25m spaced sample grid scenario.

more efficient extraction planning that returns a higher NPV than the lower density sampling. If the cost of sampling is brought into the calculation, the effect is reversed, with the 25×25m spacing producing a considerably lower NPV than the other scenarios. The best NPV was achieved by the 200×200m spacing.

It should be kept in mind that the 200×200m spacing would be a high risk choice in the case of deposits that are of lower grade, or have more patchy mineralisation, than the simulation used here. As has been demonstrated in section 4, accurate rate and recovery predictions are important in maximising NPV. These predictions are very dependant on geological observations and metallurgical processing measurements during the sampling campaign. It is thus conceivable that low sampling density could jeopardise accurate rate and recovery predictions and thereby affect NPV negatively. By the same argument, there is little sense in sampling at 100×100m since the NPV is virtually identical to the 50×50m sample spacing.

It is therefore concluded that although the 200×200m spacing returned a slightly higher NPV as seen in Figure 38, the 50×50m spacing is preferred.

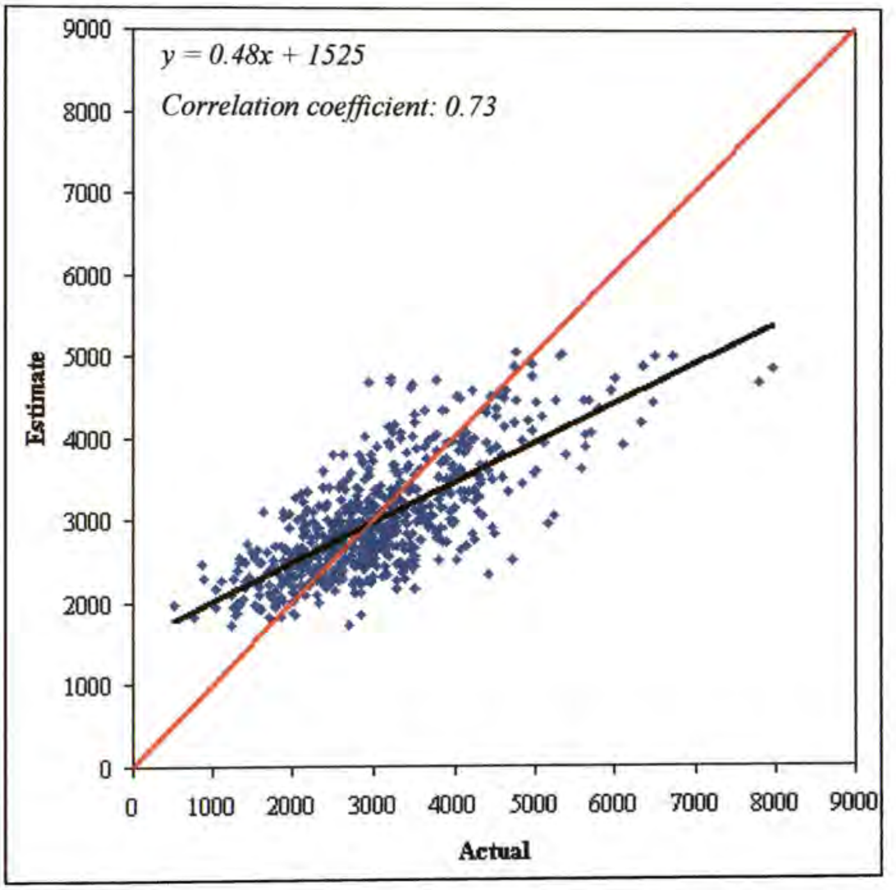

Figure 36. Correlation plot between the estimated and actual stone value for the 100×100m spaced sample grid scenario.

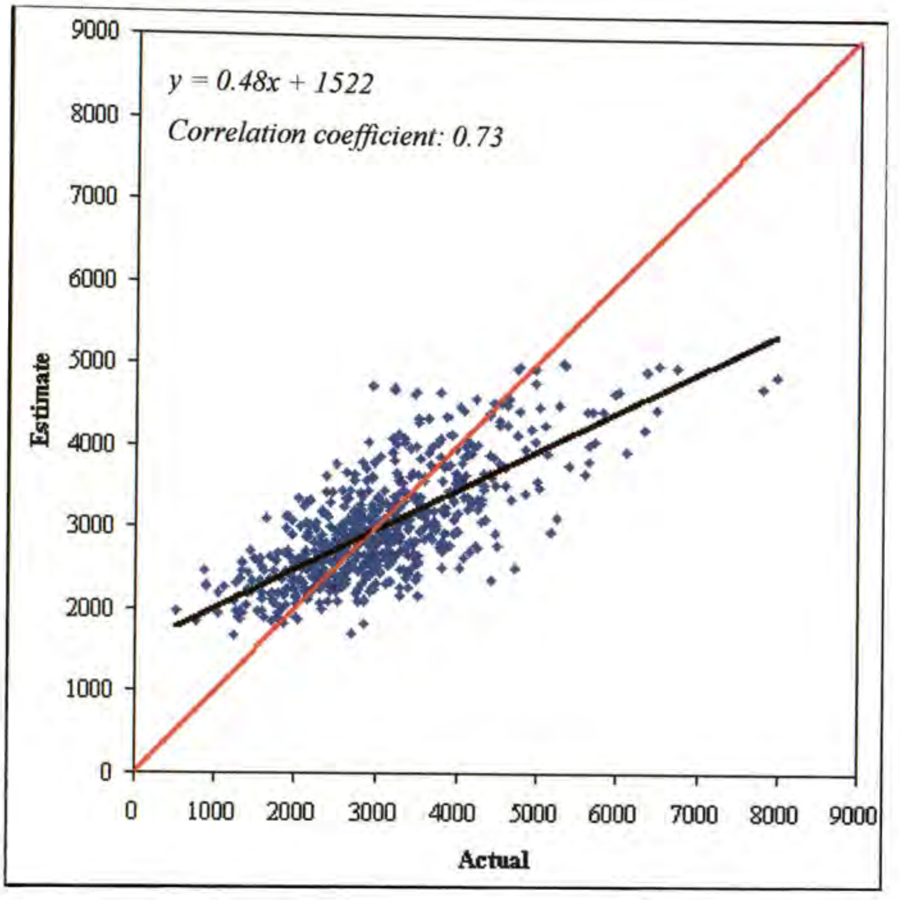

Figure 37. Correlation plot between the estimated and actual stone value for the 200×200m spaced sample grid scenario.

Table 8. NPV of the various sample density scenarios.

	Sampling cost excluded		Sampling cost included	
Sample Density	NPV	% Change in NPV	NPV	% Change in NPV
Baseline	**99,738,590**	**0%**	**99,738,590**	**0%**
25×25	91,667,039	-8%	3,339,886	-97%
50×50	81,921,623	-18%	81,921,623	-18%
100×100	59,075,740	-41%	81,036,862	-19%
200×200	59,754,392	-40%	87,024,798	-13%

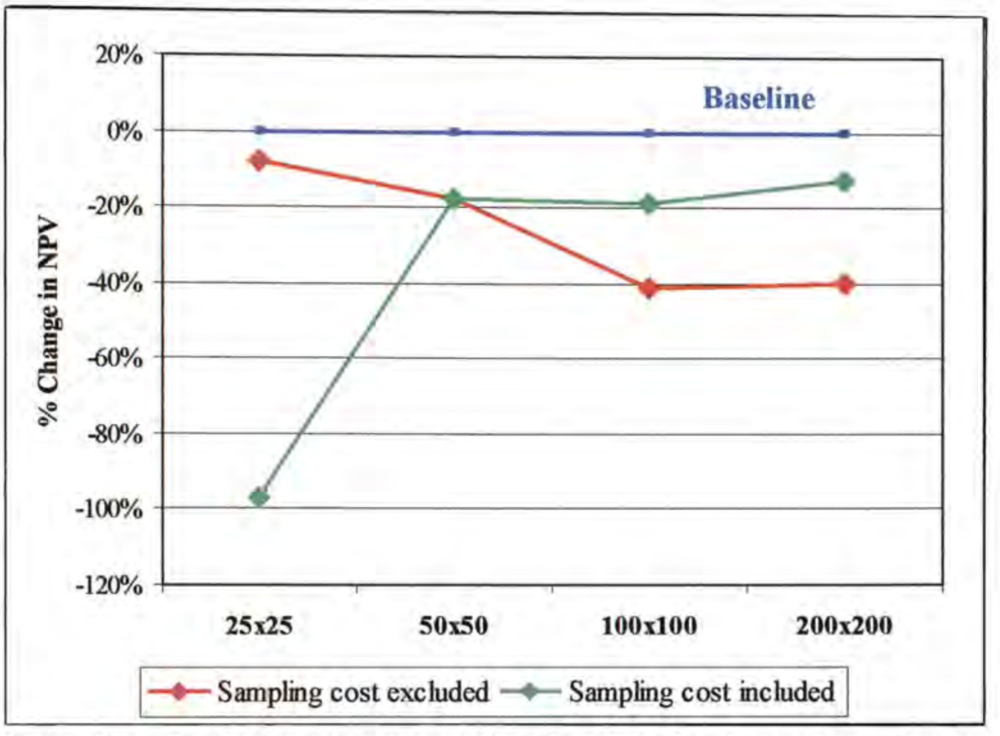

Figure 38. Sensitivity plot of variation in NPV for the various sample densities.

6.2. Sample support size

Sample support size is a decision that is usually driven by the statistical appropriateness as well as cost. The sample must be representative, but on the other hand, large sample support is costly and difficult to obtain. The adage that larger samples are more efficient for diamond deposits, does not always apply, especially in deposits where mineralisation is concentrated in small traps (Duggan, 1995).

The purpose of this section is to test various sample support sizes in the simulation in order to see how it affects the NPV. It should be seen as a pure financial measurement on sample support size and not a measure of statistical appropriateness.

The metallurgical processing plant through which the samples must be processed has a constraint with regards to volume per time unit. It was therefore assumed for the NPV model that the larger the sample support size, the more material must be processed, implying a reduction in sampling rate. In reality it is however not as simple, as the batch process activities and ship location activities also influence the sampling rate. For the purpose of this study the following assumptions were made with regards to changing the support/size of the sample in the model:

- $20m^2$ - 15 samples per day
- $10m^2$ - 20 samples per day
- $5m^2$ - 25 samples per day
- $2.5m^2$ - 30 samples per day

Table 9 shows the exploration cost that was returned by the model for the various sample support size scenarios.

Table 9. The impact of sample support size selection on the exploration cost.

Sample Support Size	Exploration Cost	% Change in Exploration Cost in Comparison with the Baseline
$20m^2$	49,985,000	33%
$10m^2$	37,685,000	0%
$5m^2$	30,485,000	-19%
$2.5m^2$	25,85,000	-32%

The same methodology was followed as described in section 5.1. Correlation plots between the estimates and the actuals can be seen in Figures 39 to 41 for the $20m^2$, $5m^2$ and $2.5m^2$ sample support size. The result for the $10m^2$ sample support size can be seen in Figure 15. It is clear from these correlation plots that there is a definite deterioration in the accuracy of the estimates for the simulation as the sample support size is reduced.

Two sets of NPV's were also generated for the sample support size scenarios. The first set was generated using only the estimates without adjusting the sampling cost and in the second set, the sampling costs were adjusted accordingly. The resulting NPV's can be seen in Table 10 and Figure 42. If the sampling costs are not considered, it can be seen that the better estimates produced from the larger sample support sizes lead to more efficient extraction planning that returns a higher NPV than the smaller sample support sizes.

If the cost of sampling is brought into the calculation, the highest NPV is returned by the $10m^2$ sample.

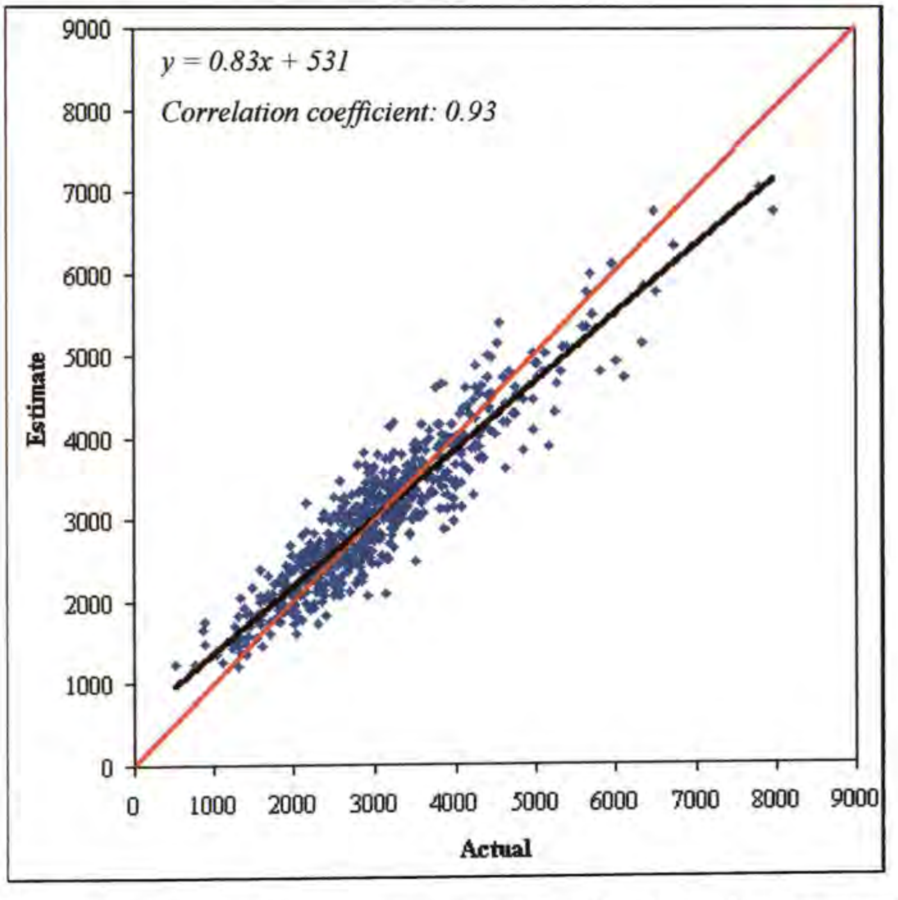

Figure 39. Correlation plot between the estimated and actual stone value for the $20m^2$ sample support size scenario.

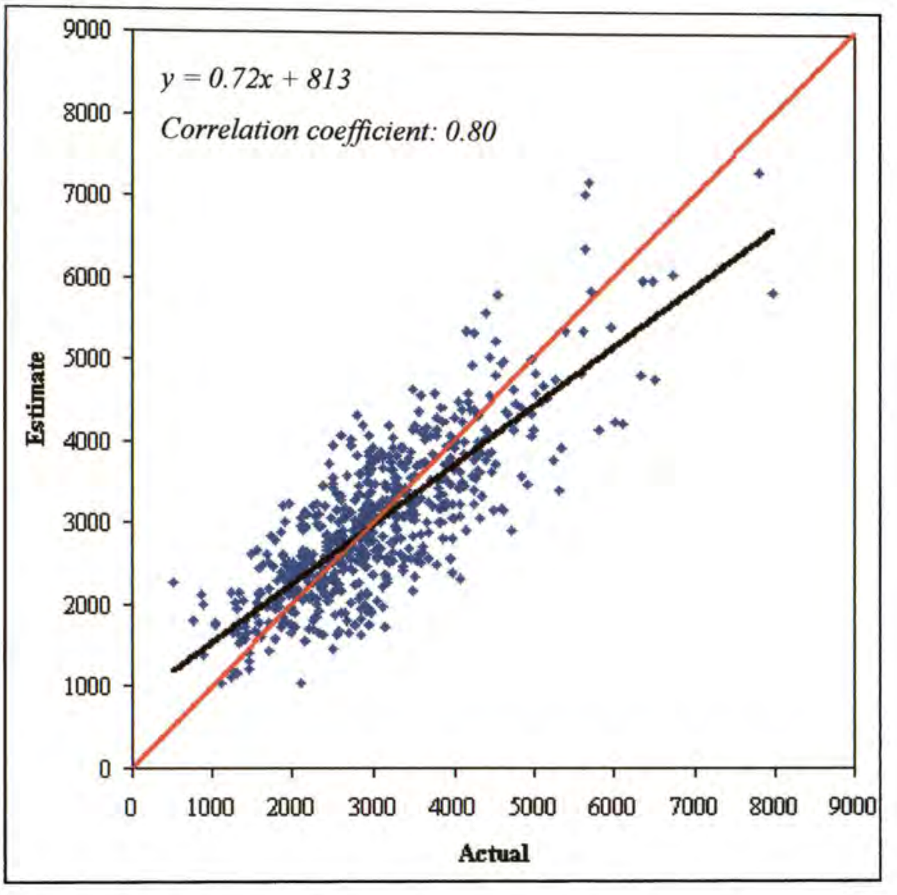

Figure 40. Correlation plot between the estimated and actual stone value for the $5m^2$ sample support size scenario.

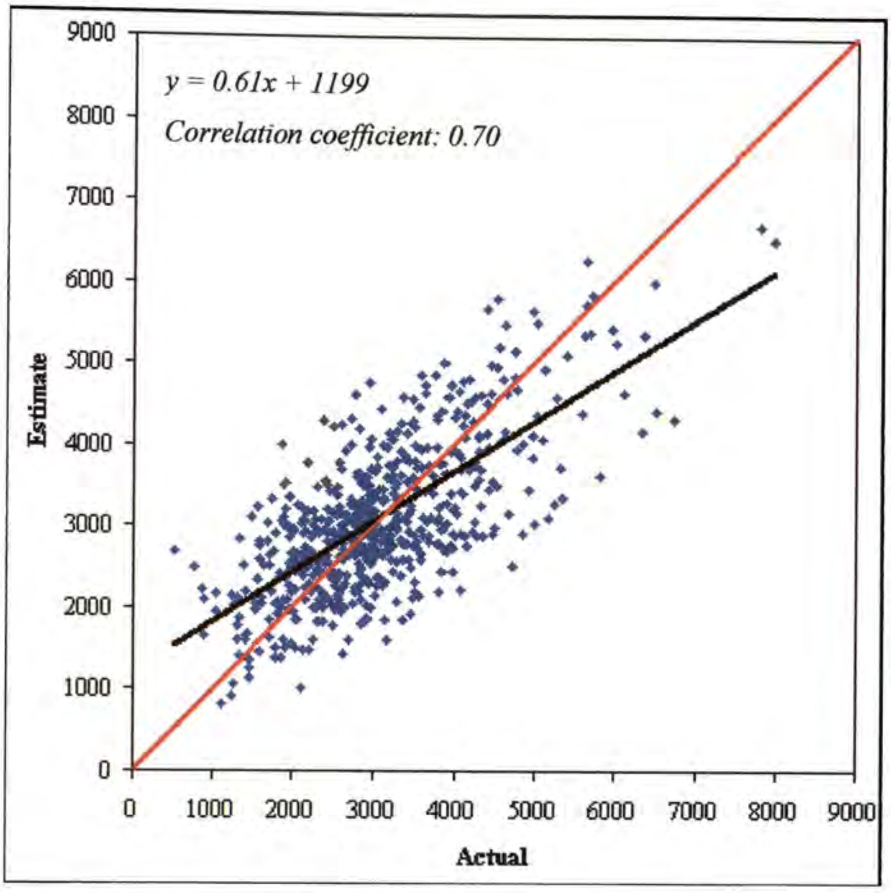

Figure 41. Correlation plot between the estimated and actual stone value for the 2.5m^2 sample support size scenario.

Table 10. NPV of the various sample support size scenarios.

Sample Support Size	Sampling cost excluded NPV	Sampling cost excluded % Change in NPV	Sampling cost included NPV	Sampling cost included % Change in NPV
Baseline	**99,738,590**	**0%**	**99,738,590**	**0%**
20m^2	87,253,753	-13%	77,359,181	-22%
10m^2	81,921,623	-18%	81,921,623	-18%
5m^2	74,665,335	-25%	80,457,279	-19%
2.5m^2	61,368,008	-38%	71,021,249	-29%

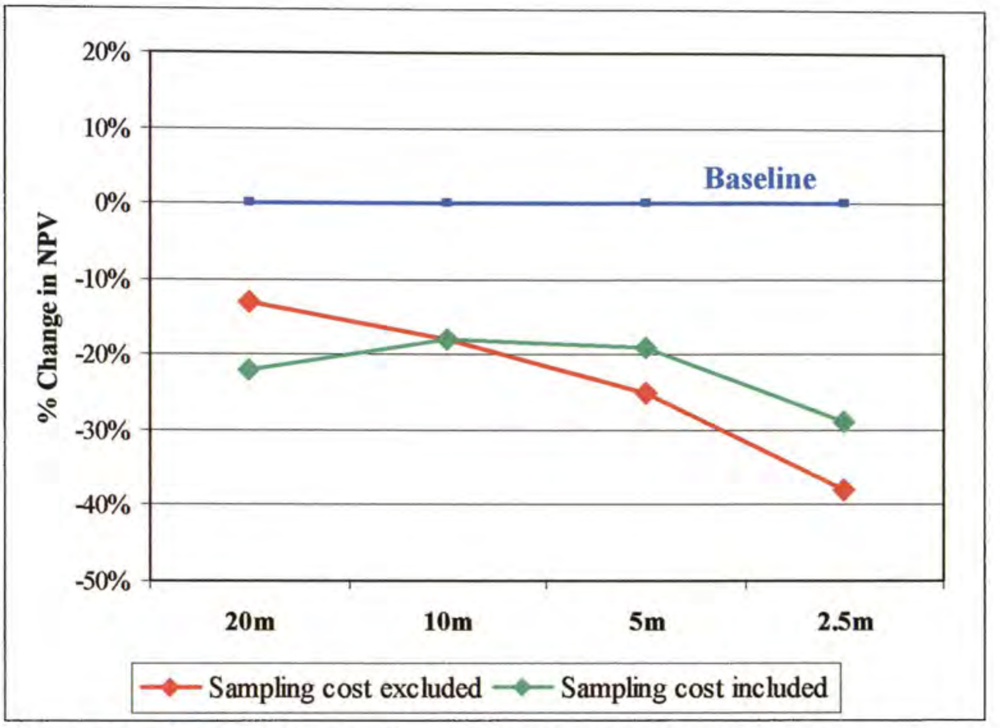

Figure 42. Sensitivity plot of variation in NPV for the various sample support sizes.

7. Discussion and conclusions

7.1. Discussion

The various scenarios and their related NPV's presented in the previous sections aimed to cover the relevant issues and decisions commonly encountered by mineral resource practitioners during the exploration and mining of marine diamond deposits. The aim was not to run scenarios *ad infinitum*, but to limit the scenarios presented in the previous section to realistic choices and problems.

The sensitivity plot in Figure 11 showed that, in comparison with other cost aspects, variation between -25% and 25% in the cost of exploration activities has a small impact on the NPV of the model. This could lead to a conclusion that, from a financial point of view, exploration aspects are unimportant and should not be given as much consideration as other cost aspects. It was however shown, that it is not only exploration costs that impacts on a project's NPV, but also the decisions made within the exploration process. In other words, the impact of sampling and evaluation practice on NPV can play a larger role than the exploration costs.

Figures 12 and 13 showed that it is important to predict recovery efficiency and mining rate as accurately as possible as any misinterpretation in these areas leads to a decline in NPV. Under-prediction of these aspects showed a more rapid decline in NPV than over-prediction, due to the fact that economic ore blocks were 'removed' from the extraction plan. The negative impact of over-prediction was not as strongly illustrated by the model as would be expected from marginal deposits, or deposits that are less mineable than those represented by the model.

When the impact of misfitting the semi-variogram model on the NPV was investigated, smaller variations in NPV were seen than with the scenarios covering the other aspects considered. NPV was seen to be less sensitive to misinterpreting the range (Figure 33) than it was to misinterpreting the nugget effect (Figure 24). It was also seen that the wrong nugget effect and range actually improved the estimate in cases where the misfitted semi-variogram model better reflected the true variance of the deposit than the variance reflected by the sampling. The 'worst' estimate and lowest NPV were returned when a moving average was utilised instead of geostatistics.

Increasing sample density and utilising larger sample support sizes improved the estimates that in turn lead to better extraction planning. However, the cost involved with improving the estimates by these methods, had a negative effect on the NPV. This is illustrated in Figure 42 where the $10m^2$ sample support size returned the highest NPV of the scenarios. The smaller sample support sizes returned less accurate estimates that could not be outweighed by the saving in sampling cost.

When sample density was considered (Figure 38) the cost of the 25×25m spaced sampling grid far outweighed the advantages of the improved estimate. Decreasing the sample density beyond 50×50m had very little effect on the NPV. This result was perceived to be strongly related to the type of deposit. Marginal and less contiguous deposits would be most likely be less forgiving to low sample density than the simulation used in this model. The result could also mean that beyond a certain point, opening up the sample density to save on exploration cost, does not add significant value to the project. Lower sample density is also accompanied by the risk of not getting enough geological information for rate and recovery predictions and could also result in poor variography.

A noticeable outcome of the scenarios calculated is that even the best estimates produced an 18% lower NPV than would have been possible if the actual stone content of the oreblocks were known before extraction, as was the case in the baseline.

The various decisions taken in the exploration and evaluation of the project impact directly upon the extraction strategy adopted for the deposit, which in turn impacts upon the NPV. In marginal deposits, correct orebody delineation can be considered the main driver of NPV. However, the simulation used here cannot be considered marginal as 82% of the ore blocks were above the cut-off stone rate of 300 stones per day. When mining rate and recovery were investigated, it was seen that under prediction of these factors had a negative effect on NPV as economic blocks were pushed below the cut-off stone rate, which lead to poor delineation of the orebody. One could therefore argue that it is probably a good idea in this type of non-marginal deposit to make a global resource estimate and simply mine all of it. It is however proposed that it is not only orebody delineation that drives NPV, but also the sequence of extraction. This is particularly useful in marine diamond mining projects to maximise NPV as there are very few constraints on the sequence of extraction compared to land-based mining projects.

A way of looking at how the extraction sequence impacted on the NPV calculation, was to look at gross revenue. The gross revenue values entered into the NPV calculation for each year is directly related to the sequence of extraction. The gross revenues for the various scenarios tested were compared to each other and the baseline by calculating the ratio of the scenario gross revenue to the baseline gross revenue. These are illustrated in Figures 43 to 48. The data used to draw up the diagrams are illustrated in Table 17 and 18 in Annexure 2.

In Figures 43 and 44 the mining recovery and mining rate scenarios where these factors were under-predicted show a dramatic drop in gross revenue that can be directly related to incorrect orebody delineation. For the scenarios dealing with estimation processes, sample density and sample support size, illustrated in Figures 45 to 48 orebody delineation is not such an important factor in NPV maximisation, but rather obtaining the correct extraction sequence. Figure 47 is a good example where the 25×25m and 50×50m sample densities deliver gross revenue that is fairly close to the baseline for all eight years. The 100×100m and 200×200m sample densities

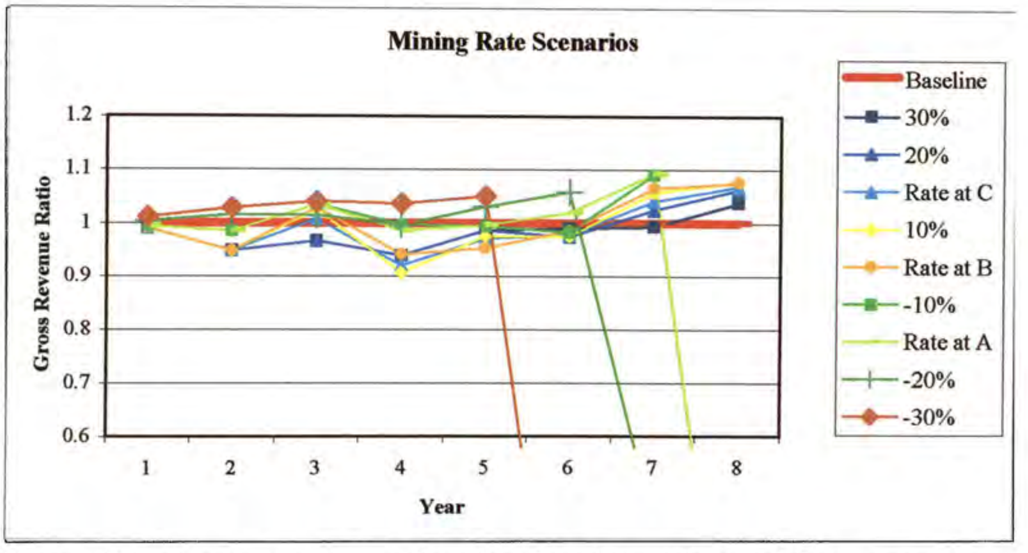

Figure 43. Gross revenue ratio plot for the various mining recovery scenarios.

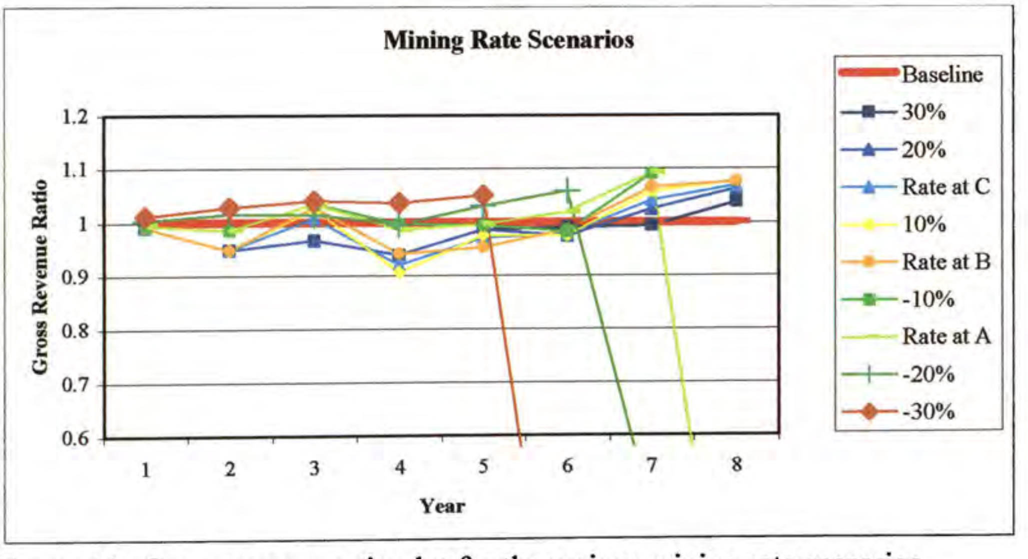

Figure 44. Gross revenue ratio plot for the various mining rate scenarios.

deliver gross revenue close to the baseline in the first three years, but in year four there is a dramatic drop in gross revenue compared to the baseline due to less effective sequencing caused by the influence of lower density sampling on the accuracy of the estimate.

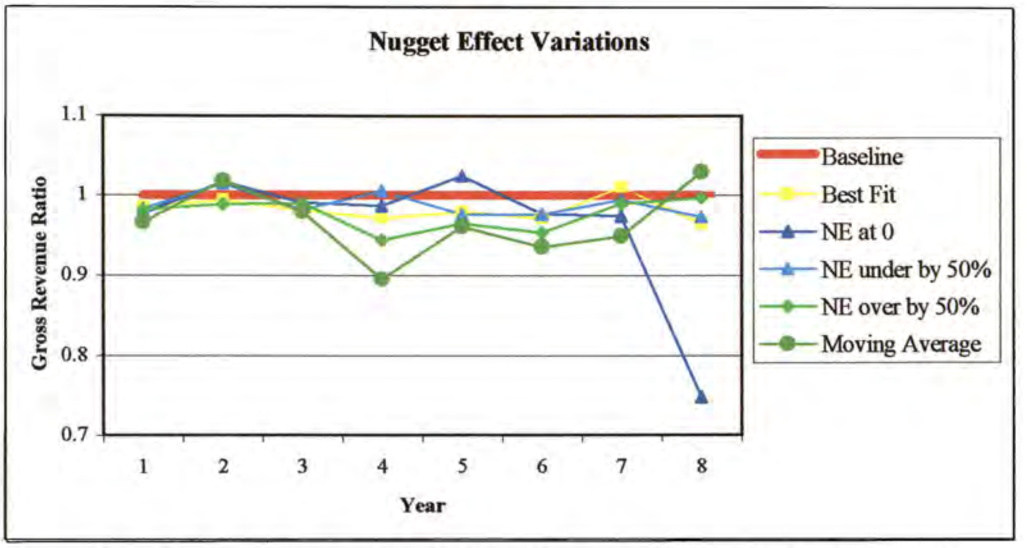

Figure 45. Gross revenue ratio plot for the various nugget effect variation scenarios.

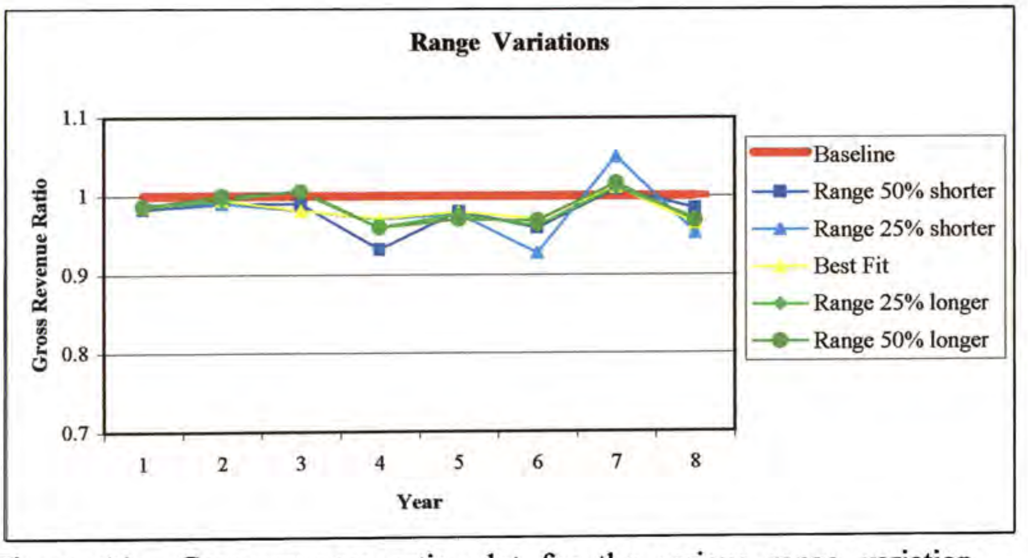

Figure 46. Gross revenue ratio plot for the various range variation scenarios.

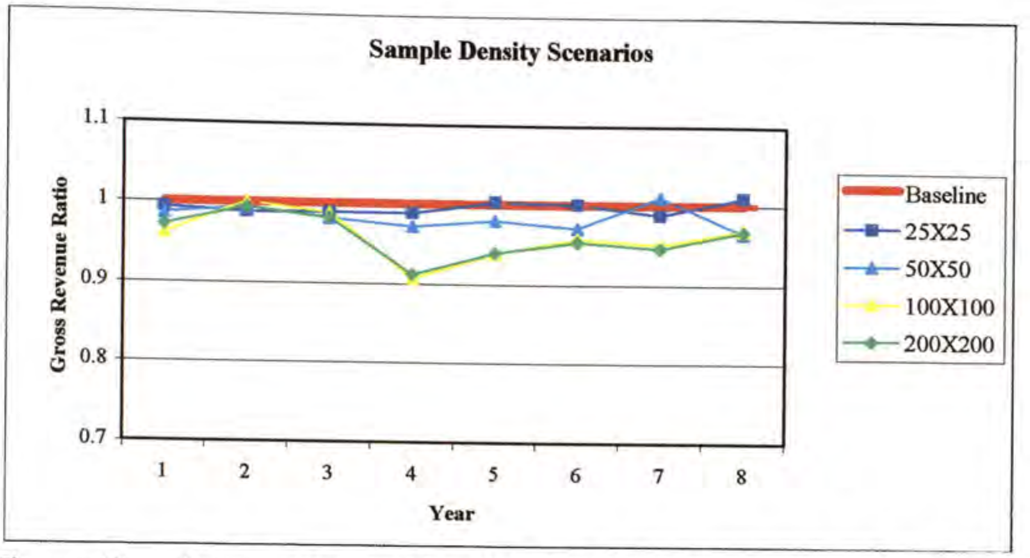

Figure 47. Gross revenue ratio plot for the various sample density scenarios.

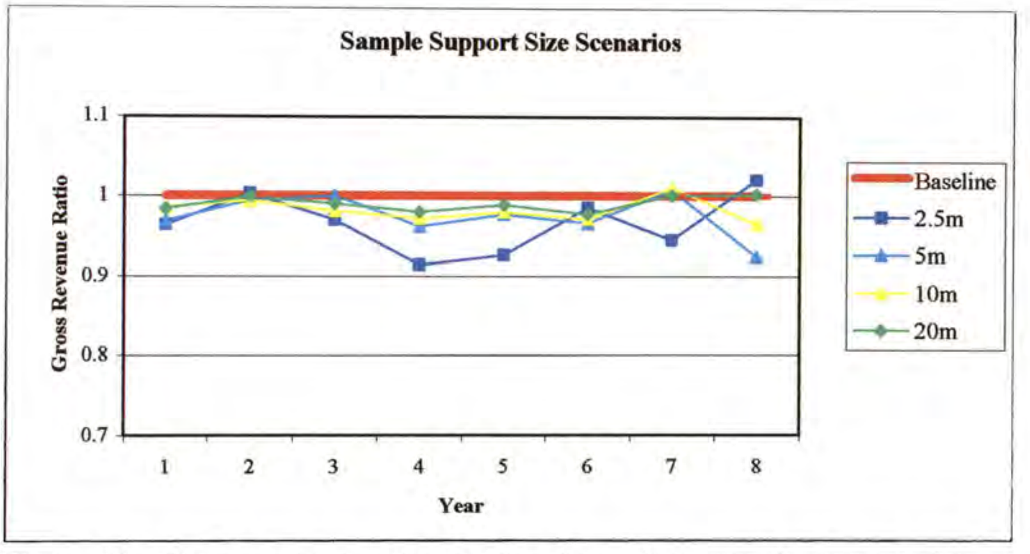

Figure 48. Gross revenue ratio plot for the various sample support size scenarios.

7.2. Conclusions

The impact of certain pertinent aspects in marine diamond mining was measured in financial terms by utilising the sensitivity of NPV to various sampling and estimation variables. It is believed that the method demonstrated here by means of a simulation can be used in actual marine mining projects to bring a financial perspective on sampling and estimation decisions and ultimately add value.

The following conclusions and deductions were made:

- Accurate rate and recovery predictions are very important for optimising the NPV of a marine mining project. This can be achieved, not only by a thorough geological understanding of the deposit, but also by adequate knowledge of the interaction between mining tool and orebody.
- Making the wrong choices in the modelling of the experimental semi-variogram does have a negative impact on NPV. It is also important that the sample spacing is adequate to reflect the variography of the deposit.
- Finding the optimal sampling density and support size do have a positive effect on NPV. For this study the $10m^2$ support size was optimal, but it is unfortunately not possible to directly deduct the optimal density and support size for other deposits from this study. The study does however demonstrate a financial approach that can be combined with geostatistics to find this optimum value on a deposit by deposit basis.
- It is not only the correct delineation of the orebody that has a major impact on the NPV, but also the optimal sequencing of the oreblocks.
- The impact of other possible sampling and estimation considerations, of which sampling errors is probably the most important, were not included in this study, but it is postulated that their impact could be measured in a similar fashion as demonstrated here.

8. References

Bujtor, G.J. & McMahon, D.W. 1983. *Mineral reserve estimation for project development purposes.* The Australian Institute for Mining and Metallurgy, Sydney Branch, Project Development Symposium, November 1983, 12 pp.

Cape Business News 1999. *Rippling Diamond Intrigue*. August 1999, p. 11.

Corbett, I.B. 1996. A review of diamondiferous marine deposits of western Southern Africa. *Africa Geoscience Review*, 3 (2), pp. 157-174.

Corbett. I.B. & McMillan, I.K. 1998. From Shore to Shelf and Back Again, *7th International Kimberlite Conference, Extended Abstracts*. April 1998: Cape Town, pp. 167–169.

Corbett, I.B. 2000. A case study in the development of the offshore diamond mining industry in Namibia. *Workshop on Mineral Resources of the International Seabed Area*. June 2000: Kingston, Jamaica, 35 pp.

Covacic, D. & Clarke, J. 1990. Sampling techniques. *Mining Magazine*. October 1990, pp. 270-273.

Dale, D.C. & McMillan, I.K., 1999. *How do microscopic fossils help in diamond exploration?* Earthyear: The Essential Environmental Guide, Edition 20, pp. 114-115.

De Beers Consolidated Mines & De Beers Centenary AG. 2000. *Annual Reports 2000*. Johannesburg: De Beers Corporate Communications Department, p. 41.

Deutsch, C.V. & Journel, A.G. 1998. *GSLIB: Geostatistical Software Library and Users Guide.* Second Edition. Oxford University Press, New York, pp. 139-146.

De Wit, M.C.J. 1999. Post-Gondwana Drainage and the Development of Diamond Placers in Western South Africa. *Economic Geology*. Vol. 94, pp. 721-740.

Duggan, S.P. 1995. *Geostatistical Evaluation of a Marine Deposit.* Unpubl. MSc Thesis, University of the Witwatersrand, Johannesburg, 81pp.

Garnett, R.H.T. 1998. *Estimation of marine mineral reserves*. Transactions: Society for Mining, Metallurgy and Exploration. Vol. 304, pp. 69-82.

Garnett, R.H.T. 1998a. Risks in marine diamond mining – Lessons from the past and present. Abstracts. 29th Underwater Mining Institute Conference. Toronto, Canada.

Gurney, J.J., Levinson, A.A., & Smith, H.S. 1991. Marine mining of diamonds off the west coast of Southern Africa. *Gems & Gemology*. Winter 1991, pp. 206-219.

Haage, D. 1997. *Deep water diamond mining, past, present and future.* Minorco, Anglo American and De Beers - Group mining Symposium, 15pp.

Hasenfuss, M. 2000. *Namco to lift its diamond production by 45%.* Cape Times Business Report, 1 March 2000.

Jacob, R.J., Bluck, B.J., & Ward, J.D. 1999. Tertiary-age diamondiferous fluvial deposits of the Lower Orange River Valley, Southwestern Africa. *Economic Geology*, Vol. 24, pp. 749-758.

Louw, A.C. 1998. Ocean Diamond Mining's quest for more efficient diamond recovery. Abstracts. 29th Underwater Mining Institute Conference. Toronto, Canada.

Mallinson, C. 1994. *An Introduction to Financial Evaluation and Risk Analysis of Mineral Deposits*. Course notes, Anglo O.E.D., 95pp.

McLaney, E.J. 2000. *Business Finance, Theory and Practice.* 5th Edition. England: Pearson Education Limited, pp. 69-77.

Murray, L.G., Joynt, R.H., O′Shea, D.O.C., R.W. & Kleinhan, L. 1970. The geological environment of some diamond deposits off the coast of South West Africa. In: *The geology of the East Atlantic continental margin – 1 General Economic Papers.* Inst. Geol. Sci. Rep. 70/13, pp. 121-141.

Prins, C.F. 1991. *Assessing the Potential of Deposits*. Unpubl. MSc Thesis, University of the Witwatersrand, Johannesburg, 47pp.

Rombouts, L. 1998. Resource/reserve estimation of marine diamond deposits. Abstracts. 29th Underwater Mining Institute Conference. Toronto, Canada.

Rudenno, V. 1998. *The Mining Valuation Handbook.* Victoria: Wrightbooks, pp. 105-107

Smith, L.D. 1997. A critical examination of the methods and factors affecting the selection of an optimum production rate. *CIM Bulletin*, 90, pp. 48-54.

Spicer, D. 2001. *Stalwart marine miner conquering rough seas*. Mining Weekly, 15-21 June 2001, p. 6.

Stein, N. 2001. *The De Beers story: A new cut on an old monopoly*. Fortune. February19, 2001, pp. 80-95.

Stevenson, I.R. 1998. *Remote sediment classification using CHIRP sonar.* Abstracts. Abstract. Anglo American Corporation 5th Geophysical Conference. Drakensberg, South Africa.

Stevenson, I.R. & Nicholson, P. 1998. *An overview of marine geophysical operations at De Beers Marine.* Abstract. Anglo American Corporation 5th Geophysical Conference. Drakensberg, South Africa.

Tankard, A.J., Jackson, M.P.A., Eriksson, K.A., Hobday, D.K., Hunter, D.R. and Minter, W.E.L. 1982. *Crustal Evolution of Southern Africa: 3.8 Billion Years of Earth History*. New York: Springer, pp. 433-450.

Trans Hex. 2001. *Making deep-water waves.* Trans Hex News. Vol.2:2. July 2001, p. 1.

Ward, J.D. 1998. West coast field excursion. *Field Guide, 7th International Kimberlite Conference*. Cape Town, South Africa, pp. 11-12.

ANNEXURE 1

The NPV and Extraction Sequence Calculations

The various NPV calculations performed for this treatise were carried out using the Microsoft Excel 2000 spreadsheet application. The scenario used in this example consists of the baseline input parameters and the estimate generated with the 50×50m sample spacing and the 10m^2 sample support size. The NPV calculation can be seen in Table 11. All the information feeding into the NPV calculation, except the gross revenue, was derived from an Input table illustrated in Table 12.

The following calculations took place to derive the NPV: Total Costs were calculated by adding the expenditure for each year. Income Before Tax were derived by deducting Total Costs from the Gross Revenue. The appropriate Amortisation were deducted from the Income Before Tax to arrive at Taxable Income After Amortisation, from which the Tax amount was calculated at 30%. The Tax amount was then deducted from Income Before Tax to derive Income After Tax (this value is negative in Year 0, as no income was generated). The Discount Rate was then applied to Income After Tax to calculate the present value of the Cash Flows for each year. Finally the Cash Flows were added up to derive the NPV.

Gross Revenue was influenced by the blocks selected to be mined, as well as, the extractable content of the selected blocks. The data for the calculations were contained in two tables, the Actual table and the Estimate table. The layout of these two tables were identical and as follows:

Block	X	Y	Stones	Area	Panel	Zone	Recovery %	m^2/day	Recovered

- **Block** indicates the number of the 100×100m mining block.
- The **X** and **Y** coordinate of the mining block.
- **Stones** refer to the actual/estimated number of stones in the block.
- **Area** indicates the size of the block (10,000m^2 in all cases), the value is used to calculate the time it will take to extract the block.
- Each block belongs to a **Panel**. A panel consists of 9 blocks and it is used as the unit that a mining vessel can extract on a single anchor spread.
- **Zone** indicates to which of the three geological zones the block belongs as each zone has its own recovery efficiency and mining rate factor.
- **Recovery %** is obtained from the Input table and the value is dependent on the zone associated with the block. The Actual table derives this value from the actual mining parameters, whilst the Estimate table utilises the predicted mining parameters.
- **m^2/day** or the Rate is obtained from the Input table and the value is dependent on the zone associated with the block. The Actual table derives this value from the actual mining parameters, whilst the Estimate table utilises the predicted mining parameters. This value is used to calculate the amount of time it will take to extract a given block.
- **Recovered** represents the amount stones that can be extracted from the block and it is derived by applying the recovery factor to the actual of estimated stone content.

The Actual and Estimate table for this example were combined in Table 13.

The next step was to exclude all blocks for which the estimated production rate fell below 300 stones/day. This was done by entering a logical formula that returned a zero value on the Block Calculation table, if the estimated production rate for a given block is below 300 stones/day. If however the estimated production rate was above 300 stones/day, the actual and estimated recoverable stones and time to complete the block were fed into the Block Calculation table (Table 14).

The panel contributions were then calculated by grouping the blocks utilising an array formula. The Panel Contribution table can be seen in Table 15. The table contains the estimated number of stones/day for the panel, panel number, the actual number of extractable stones in the panel, and the number of days it will take to mine the panel. This table is sorted from highest to lowest estimated number of stones/day. The extraction sequence has now been established. The stone contribution from each panel were then allocated to each year and the total number of stones for each year multiplied by the price per stone from the Input table to derive Gross Revenue (Table 16).

The whole procedure of generating an NPV was fairly automated. The correct input values and the appropriate set of estimates were needed and the correct NPV was returned. The allocation of the panel contributions to the correct year did however require some manual intervention. This step was fairly complex as blocks falling between years had to be apportioned correctly. Although the grouping and totalling of the values for every year was derived using array formulas and logical arguments, the switch in year 3 and 6 to 330 days, to accommodate the inport period, required manual manipulation of the calculation. In some of the scenarios, mining ended before year 8 and this required close inspection of the NPV calculation table and corrections if necessary.

Table 11. The Net Present Value (NPV) calculation.

	Year	0	1	2	3	4	5	6	7	8
Gross Revenue		0	525,371,060	428,403,750	358,208,217	376,561,926	364,213,308	312,368,151	342,980,320	311,160,043
Sampling and Survey		37,685,000								
Vessel Capital Outlay		700,000,000								
Operating Cost			109,500,000	109,500,000	109,500,000	109,500,000	109,500,000	109,500,000	109,500,000	109,500,000
Inport Cost					20,000,000			20,000,000		
Overheads			36,500,000	36,500,000	36,500,000	36,500,000	36,500,000	36,500,000	36,500,000	36,500,000
Total Costs		737,685,000	146,000,000	146,000,000	166,000,000	146,000,000	146,000,000	166,000,000	146,000,000	146,000,000
Income Before Tax		-737,685,000	379,371,060	282,403,750	192,208,217	230,561,926	218,213,308	146,368,151	196,980,320	165,160,043
Amortisation		0	221,305,500	184,421,250	147,537,000	110,652,750	73,768,500			
Taxable Income After Amortisation			158,065,560	97,982,500	44,671,217	119,909,176	144,444,808	146,368,151	196,980,320	165,160,043
Tax	30%	0	47,419,668	29,394,750	13,401,365	35,972,753	43,333,443	43,910,445	59,094,096	49,548,013
Income After Tax		-737,685,000	331,951,392	253,009,000	178,806,852	194,589,173	174,879,866	102,457,706	137,886,224	115,612,030
Discount Rate	**20%**	1.00	1.20	1.44	1.73	2.07	2.49	2.99	3.58	4.30
Cashflows		-737,685,000	276,626,160	175,700,694	103,476,187	93,841,229	70,280,296	34,312,878	38,481,515	26,887,663
NPV	**81,921,623**		-461,058,840	-285,358,146	-181,881,958	-88,040,729	-17,760,433	16,552,445	55,033,960	81,921,623

Table 12. Input data for the NPV calculation.

Actual Mining Parameters	
Mining recovery A	90%
Mining recovery B	80%
Mining recovery C	70%
Rate A	1500 m^2/day
Rate B	1800 m^2/day
Rate C	2000 m^2/day
Predicted Mining Parameters	
Mining recovery A	90%
Mining recovery B	80%
Mining recovery C	70%
Rate A	1500 m^2/day
Rate B	1800 m^2/day
Rate C	2000 m^2/day
Cut off Rate	300 stones/day
Financial Information	
Price per Stone	R 2,300
Operating Cost of Vessel	R 300,000 /day
Sampling Cost	R 300,000 /day
Sampling Rate	20 /day
Survey Cost (1×1km/day)	R 155,000
Taxation Rate	30%
Overheads	R 100,000 /day
Inport (30 days)	R 20,000,000

Table 13. The Actual data was used to calculate the Gross revenue if a block is selected for extraction, whilst the Estimate data was used to derive the optimal extraction sequence.

							ACTUAL		ESTIMATE	
X	**Y**	**Area**	**Panel**	**Zone**	**Rec%**	**m^2/day**	**Actual**	**Rec**	**Estimate**	**Rec**
-50	-50	10000	1	A	90%	1500	5255	4730	3983	3584
-50	-150	10000	1	A	90%	1500	3152	2837	2630	2367
-50	-250	10000	1	A	90%	1500	1817	1635	2480	2232
-50	-350	10000	2	A	90%	1500	2136	1922	2329	2096
-50	-450	10000	2	A	90%	1500	3388	3049	3040	2736
-50	-550	10000	2	A	90%	1500	3614	3253	3616	3254
-50	-650	10000	3	A	90%	1500	3315	2984	3896	3506
-50	-750	10000	3	A	90%	1500	2943	2649	3931	3538
-50	-850	10000	3	A	90%	1500	3360	3024	3123	2810
-50	-950	10000	4	A	90%	1500	2045	1841	1799	1619
-50	-1050	10000	4	A	90%	1500	2967	2670	2077	1869
-50	-1150	10000	4	A	90%	1500	3275	2948	3421	3079
-50	-1250	10000	5	A	90%	1500	3524	3172	3733	3360
-50	-1350	10000	5	B	80%	1800	3304	2643	3127	2501
-50	-1450	10000	5	B	80%	1800	1859	1487	2009	1607
-50	-1550	10000	6	B	80%	1800	525	420	1470	1176
-50	-1650	10000	6	B	80%	1800	2156	1725	2277	1822
-50	-1750	10000	6	B	80%	1800	3864	3091	2943	2354
-50	-1850	10000	7	B	80%	1800	4072	3258	2930	2344
-50	-1950	10000	7	B	80%	1800	2627	2102	2259	1807
-50	-2050	10000	7	B	80%	1800	2198	1758	2360	1888
-50	-2150	10000	8	B	80%	1800	2664	2131	2759	2207
-50	-2250	10000	8	B	80%	1800	1901	1521	2022	1618
-50	-2350	10000	8	B	80%	1800	781	625	1256	1005
-50	-2450	10000	9	B	80%	1800	1602	1282	2045	1636
-50	-2550	10000	9	B	80%	1800	3135	2508	2348	1878
-50	-2650	10000	9	B	80%	1800	2940	2352	1962	1569
-50	-2750	10000	10	B	80%	1800	1861	1489	1805	1444
-50	-2850	10000	10	B	80%	1800	2055	1644	2027	1622
-150	-50	10000	1	A	90%	1500	3424	3082	3781	3402
-150	-150	10000	1	A	90%	1500	2934	2641	2756	2481
-150	-250	10000	1	A	90%	1500	2828	2545	3005	2705
-150	-350	10000	2	A	90%	1500	2624	2362	2801	2521
-150	-450	10000	2	A	90%	1500	3039	2735	3514	3163
-150	-550	10000	2	A	90%	1500	3014	2713	3623	3261
-150	-650	10000	3	A	90%	1500	4154	3739	3566	3210
-150	-750	10000	3	A	90%	1500	2823	2541	3325	2992
-150	-850	10000	3	A	90%	1500	3296	2966	3263	2937
-150	-950	10000	4	A	90%	1500	2823	2541	2022	1820
-150	-1050	10000	4	A	90%	1500	2574	2317	1798	1618
-150	-1150	10000	4	A	90%	1500	2861	2575	2919	2627
-150	-1250	10000	5	A	90%	1500	3948	3553	3956	3561
-150	-1350	10000	5	B	80%	1800	4255	3404	3820	3056
-150	-1450	10000	5	B	80%	1800	1588	1270	1903	1522
-150	-1550	10000	6	B	80%	1800	1043	834	1441	1153
-150	-1650	10000	6	B	80%	1800	2379	1903	2630	2104
-150	-1750	10000	6	B	80%	1800	4993	3994	4469	3575
-150	-1850	10000	7	B	80%	1800	4931	3945	4367	3493
-150	-1950	10000	7	B	80%	1800	3222	2578	3196	2557
-150	-2050	10000	7	B	80%	1800	2333	1866	2816	2253
-150	-2150	10000	8	B	80%	1800	2164	1731	2336	1869
-150	-2250	10000	8	B	80%	1800	1713	1370	1828	1462
-150	-2350	10000	8	B	80%	1800	1824	1459	2207	1766
-150	-2450	10000	9	B	80%	1800	3303	2642	3224	2580
-150	-2550	10000	9	B	80%	1800	2924	2339	2980	2384

X	Y	Area	Panel	Zone	Rec%	m^2/day	Actual	Rec	Estimate	Rec
-150	-2650	10000	9	B	80%	1800	2337	1870	1653	1323
-150	-2750	10000	10	B	80%	1800	2222	1778	2540	2032
-150	-2850	10000	10	B	80%	1800	2638	2110	2658	2126
-250	-50	10000	1	A	90%	1500	2148	1933	3050	2745
-250	-150	10000	1	A	90%	1500	2595	2336	2230	2007
-250	-250	10000	1	A	90%	1500	2503	2253	1926	1733
-250	-350	10000	2	A	90%	1500	3190	2871	2391	2151
-250	-450	10000	2	A	90%	1500	3521	3169	3186	2867
-250	-550	10000	2	A	90%	1500	2895	2606	2663	2397
-250	-650	10000	3	A	90%	1500	2493	2244	2564	2308
-250	-750	10000	3	A	90%	1500	2551	2296	2477	2230
-250	-850	10000	3	A	90%	1500	2925	2633	2885	2597
-250	-950	10000	4	A	90%	1500	1639	1475	2274	2046
-250	-1050	10000	4	A	90%	1500	2061	1855	2176	1958
-250	-1150	10000	4	A	90%	1500	2730	2457	2570	2313
-250	-1250	10000	5	A	90%	1500	3394	3055	2908	2617
-250	-1350	10000	5	B	80%	1800	2989	2391	3234	2587
-250	-1450	10000	5	B	80%	1800	2008	1606	2091	1673
-250	-1550	10000	6	B	80%	1800	1380	1104	2106	1685
-250	-1650	10000	6	B	80%	1800	4070	3256	3818	3055
-250	-1750	10000	6	B	80%	1800	4686	3749	4218	3375
-250	-1850	10000	7	B	80%	1800	3813	3050	3759	3008
-250	-1950	10000	7	B	80%	1800	3442	2754	3616	2893
-250	-2050	10000	7	B	80%	1800	3036	2429	3086	2469
-250	-2150	10000	8	B	80%	1800	3176	2541	2610	2088
-250	-2250	10000	8	B	80%	1800	2895	2316	2542	2034
-250	-2350	10000	8	B	80%	1800	2593	2074	1701	1361
-250	-2450	10000	9	B	80%	1800	2041	1633	2130	1704
-250	-2550	10000	9	B	80%	1800	1929	1543	2404	1923
-250	-2650	10000	9	B	80%	1800	2863	2290	1925	1540
-250	-2750	10000	10	B	80%	1800	1957	1566	2433	1946
-250	-2850	10000	10	B	80%	1800	2511	2009	2135	1708
-350	-50	10000	11	A	90%	1500	1967	1770	2444	2200
-350	-150	10000	11	A	90%	1500	2694	2425	2465	2218
-350	-250	10000	11	A	90%	1500	1957	1761	2431	2188
-350	-350	10000	12	A	90%	1500	2617	2355	2997	2698
-350	-450	10000	12	A	90%	1500	2914	2623	2916	2624
-350	-550	10000	12	A	90%	1500	2273	2046	2669	2402
-350	-650	10000	13	A	90%	1500	3032	2729	2862	2576
-350	-750	10000	13	A	90%	1500	3120	2808	2804	2524
-350	-850	10000	13	A	90%	1500	2826	2543	2690	2421
-350	-950	10000	14	A	90%	1500	2461	2215	3343	3009
-350	-1050	10000	14	A	90%	1500	4295	3866	3909	3518
-350	-1150	10000	14	A	90%	1500	3631	3268	3167	2851
-350	-1250	10000	15	A	90%	1500	3220	2898	2719	2447
-350	-1350	10000	15	B	80%	1800	2724	2179	3193	2554
-350	-1450	10000	15	B	80%	1800	2012	1610	2225	1780
-350	-1550	10000	16	B	80%	1800	2539	2031	1919	1535
-350	-1650	10000	16	B	80%	1800	3218	2574	3179	2543
-350	-1750	10000	16	B	80%	1800	4479	3583	3527	2821
-350	-1850	10000	17	B	80%	1800	3034	2427	2540	2032
-350	-1950	10000	17	B	80%	1800	2756	2205	2807	2246
-350	-2050	10000	17	B	80%	1800	2926	2341	2552	2042
-350	-2150	10000	18	B	80%	1800	2000	1600	2143	1714
-350	-2250	10000	18	B	80%	1800	2275	1820	1882	1506
-350	-2350	10000	18	B	80%	1800	887	710	1765	1412
-350	-2450	10000	19	B	80%	1800	2429	1943	2671	2137
-350	-2550	10000	19	B	80%	1800	3092	2474	2839	2271
-350	-2650	10000	19	B	80%	1800	2755	2204	2146	1717
-350	-2750	10000	20	B	80%	1800	2664	2131	2910	2328
-350	-2850	10000	20	B	80%	1800	1881	1505	2717	2174
-450	-50	10000	11	A	90%	1500	3972	3575	2373	2136

X	Y	Area	Panel	Zone	Rec%	m^2/day	Actual	Rec	Estimate	Rec
-450	-150	10000	11	A	90%	1500	3693	3324	2761	2484
-450	-250	10000	11	A	90%	1500	3296	2966	3074	2767
-450	-350	10000	12	A	90%	1500	3471	3124	3669	3302
-450	-450	10000	12	A	90%	1500	2936	2642	3157	2841
-450	-550	10000	12	A	90%	1500	2842	2558	2863	2577
-450	-650	10000	13	A	90%	1500	5167	4650	4295	3866
-450	-750	10000	13	A	90%	1500	3820	3438	3758	3382
-450	-850	10000	13	A	90%	1500	2519	2267	3301	2971
-450	-950	10000	14	A	90%	1500	5640	5076	6069	5462
-450	-1050	10000	14	A	90%	1500	5384	4846	5346	4811
-450	-1150	10000	14	A	90%	1500	4063	3657	4030	3627
-450	-1250	10000	15	A	90%	1500	4072	3665	3806	3425
-450	-1350	10000	15	B	80%	1800	3755	3004	3257	2605
-450	-1450	10000	15	B	80%	1800	2273	1818	2509	2007
-450	-1550	10000	16	B	80%	1800	2865	2292	2527	2022
-450	-1650	10000	16	B	80%	1800	3062	2450	2502	2001
-450	-1750	10000	16	B	80%	1800	2165	1732	2877	2301
-450	-1850	10000	17	B	80%	1800	2354	1883	2735	2188
-450	-1950	10000	17	B	80%	1800	1772	1418	2450	1960
-450	-2050	10000	17	B	80%	1800	2522	2018	1997	1598
-450	-2150	10000	18	B	80%	1800	2972	2378	1767	1414
-450	-2250	10000	18	B	80%	1800	2341	1873	1815	1452
-450	-2350	10000	18	B	80%	1800	1963	1570	2236	1789
-450	-2450	10000	19	B	80%	1800	3888	3110	3034	2427
-450	-2550	10000	19	B	80%	1800	3471	2777	3081	2464
-450	-2650	10000	19	B	80%	1800	1437	1150	1320	1056
-450	-2750	10000	20	B	80%	1800	1418	1134	1338	1070
-450	-2850	10000	20	B	80%	1800	1704	1363	1627	1302
-550	-50	10000	11	A	90%	1500	3011	2710	2313	2082
-550	-150	10000	11	A	90%	1500	3392	3053	2486	2237
-550	-250	10000	11	A	90%	1500	2724	2452	2529	2277
-550	-350	10000	12	A	90%	1500	3005	2705	2930	2637
-550	-450	10000	12	A	90%	1500	2850	2565	2789	2510
-550	-550	10000	12	A	90%	1500	2960	2664	2862	2576
-550	-650	10000	13	A	90%	1500	2942	2648	3559	3203
-550	-750	10000	13	A	90%	1500	3447	3102	3896	3507
-550	-850	10000	13	A	90%	1500	4041	3637	4248	3823
-550	-950	10000	14	A	90%	1500	5642	5078	5559	5003
-550	-1050	10000	14	A	90%	1500	5724	5152	5436	4892
-550	-1150	10000	14	A	90%	1500	4172	3755	4732	4259
-550	-1250	10000	15	A	90%	1500	4479	4031	4549	4094
-550	-1350	10000	15	B	80%	1800	3811	3049	2805	2244
-550	-1450	10000	15	B	80%	1800	2569	2055	1914	1531
-550	-1550	10000	16	B	80%	1800	2928	2342	2251	1801
-550	-1650	10000	16	B	80%	1800	3506	2805	2357	1886
-550	-1750	10000	16	B	80%	1800	2923	2338	2952	2361
-550	-1850	10000	17	B	80%	1800	3410	2728	2600	2080
-550	-1950	10000	17	B	80%	1800	2209	1767	2201	1761
-550	-2050	10000	17	B	80%	1800	1625	1300	1617	1294
-550	-2150	10000	18	B	80%	1800	1992	1594	2040	1632
-550	-2250	10000	18	B	80%	1800	1496	1197	2103	1683
-550	-2350	10000	18	B	80%	1800	1684	1347	2301	1841
-550	-2450	10000	19	B	80%	1800	3299	2639	2883	2306
-550	-2550	10000	19	B	80%	1800	3228	2582	3134	2507
-550	-2650	10000	19	B	80%	1800	2548	2038	2541	2033
-550	-2750	10000	20	B	80%	1800	2407	1926	1816	1453
-550	-2850	10000	20	B	80%	1800	1781	1425	1923	1538
-650	-50	10000	21	A	90%	1500	2649	2384	2844	2560
-650	-150	10000	21	A	90%	1500	3300	2970	3538	3184
-650	-250	10000	21	A	90%	1500	2663	2397	3006	2706
-650	-350	10000	22	A	90%	1500	2969	2672	3328	2995
-650	-450	10000	22	A	90%	1500	2554	2299	3059	2753

X	Y	Area	Panel	Zone	Rec%	m^2/day	Actual	Rec	Estimate	Rec
-650	-550	10000	22	A	90%	1500	2751	2476	2870	2583
-650	-650	10000	23	A	90%	1500	3218	2896	3213	2892
-650	-750	10000	23	A	90%	1500	2967	2670	3412	3071
-650	-850	10000	23	A	90%	1500	4317	3885	3639	3275
-650	-950	10000	24	A	90%	1500	4131	3718	3707	3336
-650	-1050	10000	24	A	90%	1500	3663	3297	4284	3856
-650	-1150	10000	24	A	90%	1500	4090	3681	4406	3965
-650	-1250	10000	25	A	90%	1500	5596	5036	5449	4904
-650	-1350	10000	25	B	80%	1800	2576	2061	2841	2273
-650	-1450	10000	25	B	80%	1800	2109	1687	2128	1703
-650	-1550	10000	26	B	80%	1800	3317	2654	3050	2440
-650	-1650	10000	26	B	80%	1800	2730	2184	2704	2163
-650	-1750	10000	26	B	80%	1800	2638	2110	2700	2160
-650	-1850	10000	27	B	80%	1800	2426	1941	2806	2245
-650	-1950	10000	27	B	80%	1800	3534	2827	2460	1968
-650	-2050	10000	27	B	80%	1800	2393	1914	2117	1694
-650	-2150	10000	28	B	80%	1800	3854	3083	3637	2909
-650	-2250	10000	28	B	80%	1800	3419	2735	3570	2856
-650	-2350	10000	28	B	80%	1800	3186	2549	3476	2780
-650	-2450	10000	29	B	80%	1800	3417	2734	3925	3140
-650	-2550	10000	29	B	80%	1800	4019	3215	3498	2799
-650	-2650	10000	29	B	80%	1800	3012	2410	3054	2443
-650	-2750	10000	30	B	80%	1800	1553	1242	1419	1135
-650	-2850	10000	30	B	80%	1800	1247	998	1780	1424
-750	-50	10000	21	A	90%	1500	3824	3442	3896	3506
-750	-150	10000	21	A	90%	1500	3144	2830	3819	3437
-750	-250	10000	21	A	90%	1500	3099	2789	2651	2386
-750	-350	10000	22	A	90%	1500	3239	2915	3511	3159
-750	-450	10000	22	A	90%	1500	2975	2678	3327	2994
-750	-550	10000	22	A	90%	1500	2724	2452	2878	2590
-750	-650	10000	23	A	90%	1500	3235	2912	2949	2654
-750	-750	10000	23	A	90%	1500	3347	3012	2777	2499
-750	-850	10000	23	A	90%	1500	3038	2734	2963	2667
-750	-950	10000	24	A	90%	1500	2808	2527	2628	2365
-750	-1050	10000	24	A	90%	1500	3288	2959	3833	3450
-750	-1150	10000	24	A	90%	1500	3365	3029	3757	3381
-750	-1250	10000	25	A	90%	1500	4859	4373	4186	3768
-750	-1350	10000	25	B	80%	1800	2569	2055	3247	2598
-750	-1450	10000	25	B	80%	1800	1962	1570	2092	1673
-750	-1550	10000	26	B	80%	1800	1864	1491	2110	1688
-750	-1650	10000	26	B	80%	1800	3148	2518	3002	2401
-750	-1750	10000	26	B	80%	1800	4014	3211	3051	2441
-750	-1850	10000	27	B	80%	1800	3420	2736	3145	2516
-750	-1950	10000	27	B	80%	1800	2721	2177	2746	2196
-750	-2050	10000	27	B	80%	1800	2127	1702	2558	2047
-750	-2150	10000	28	B	80%	1800	3569	2855	2851	2280
-750	-2250	10000	28	B	80%	1800	2736	2189	2213	1770
-750	-2350	10000	28	B	80%	1800	1341	1073	2240	1792
-750	-2450	10000	29	B	80%	1800	2588	2070	2801	2241
-750	-2550	10000	29	B	80%	1800	2966	2373	2684	2147
-750	-2650	10000	29	B	80%	1800	2753	2202	2753	2203
-750	-2750	10000	30	B	80%	1800	1844	1475	1724	1379
-750	-2850	10000	30	B	80%	1800	1293	1034	1793	1435
-850	-50	10000	21	A	90%	1500	3154	2839	3336	3003
-850	-150	10000	21	A	90%	1500	3426	3083	3494	3144
-850	-250	10000	21	A	90%	1500	2428	2185	2708	2437
-850	-350	10000	22	A	90%	1500	2875	2588	3039	2735
-850	-450	10000	22	A	90%	1500	2565	2309	2660	2394
-850	-550	10000	22	A	90%	1500	2957	2661	2893	2604
-850	-650	10000	23	A	90%	1500	2844	2560	2557	2301
-850	-750	10000	23	A	90%	1500	3056	2750	2217	1995
-850	-850	10000	23	A	90%	1500	3225	2903	2397	2157

X	Y	Area	Panel	Zone	Rec%	m^2/day	Actual	Rec	Estimate	Rec
-850	-950	10000	24	A	90%	1500	2839	2555	2679	2411
-850	-1050	10000	24	A	90%	1500	4405	3965	4900	4410
-850	-1150	10000	24	A	90%	1500	5025	4523	5404	4864
-850	-1250	10000	25	A	90%	1500	3200	2880	4181	3763
-850	-1350	10000	25	B	80%	1800	4292	3434	4451	3561
-850	-1450	10000	25	B	80%	1800	2194	1755	3070	2456
-850	-1550	10000	26	B	80%	1800	1948	1558	2080	1664
-850	-1650	10000	26	B	80%	1800	2167	1734	2061	1648
-850	-1750	10000	26	B	80%	1800	3564	2851	2892	2314
-850	-1850	10000	27	B	80%	1800	3474	2779	3361	2689
-850	-1950	10000	27	B	80%	1800	2727	2182	2823	2259
-850	-2050	10000	27	B	80%	1800	2616	2093	2878	2303
-850	-2150	10000	28	B	80%	1800	1888	1510	2188	1751
-850	-2250	10000	28	B	80%	1800	2162	1730	1776	1421
-850	-2350	10000	28	B	80%	1800	1919	1535	2594	2075
-850	-2450	10000	29	B	80%	1800	3371	2697	3863	3090
-850	-2550	10000	29	B	80%	1800	2599	2079	2704	2163
-850	-2650	10000	29	B	80%	1800	2933	2346	2812	2249
-850	-2750	10000	30	B	80%	1800	2856	2285	2547	2037
-850	-2850	10000	30	B	80%	1800	2063	1650	2512	2010
-950	-50	10000	31	A	90%	1500	3822	3440	3461	3115
-950	-150	10000	31	A	90%	1500	2984	2686	3174	2856
-950	-250	10000	31	A	90%	1500	2522	2270	3289	2960
-950	-350	10000	32	A	90%	1500	2611	2350	3560	3204
-950	-450	10000	32	A	90%	1500	2144	1930	2330	2097
-950	-550	10000	32	A	90%	1500	2900	2610	2284	2055
-950	-650	10000	33	A	90%	1500	1889	1700	2312	2080
-950	-750	10000	33	A	90%	1500	1601	1441	1914	1723
-950	-850	10000	33	A	90%	1500	2400	2160	2390	2151
-950	-950	10000	34	A	90%	1500	2932	2639	2777	2499
-950	-1050	10000	34	A	90%	1500	2811	2530	3352	3016
-950	-1150	10000	34	A	90%	1500	4155	3740	5094	4585
-950	-1250	10000	35	A	90%	1500	3249	2924	3947	3553
-950	-1350	10000	35	B	80%	1800	3033	2426	2632	2106
-950	-1450	10000	35	B	80%	1800	2302	1842	2118	1694
-950	-1550	10000	36	B	80%	1800	2871	2297	2439	1951
-950	-1650	10000	36	B	80%	1800	1960	1568	2436	1949
-950	-1750	10000	36	B	80%	1800	2606	2085	2988	2391
-950	-1850	10000	37	B	80%	1800	2672	2138	3069	2455
-950	-1950	10000	37	B	80%	1800	2300	1840	2316	1853
-950	-2050	10000	37	B	80%	1800	2869	2295	2534	2027
-950	-2150	10000	38	B	80%	1800	2249	1799	2025	1620
-950	-2250	10000	38	B	80%	1800	3008	2406	2699	2159
-950	-2350	10000	38	B	80%	1800	3725	2980	3818	3054
-950	-2450	10000	39	B	80%	1800	3963	3170	4344	3475
-950	-2550	10000	39	B	80%	1800	2786	2229	2886	2309
-950	-2650	10000	39	B	80%	1800	2658	2126	3199	2559
-950	-2750	10000	40	B	80%	1800	4241	3393	3806	3045
-950	-2850	10000	40	B	80%	1800	3174	2539	3520	2816
-1050	-50	10000	31	A	90%	1500	2898	2608	2488	2239
-1050	-150	10000	31	A	90%	1500	3508	3157	2626	2363
-1050	-250	10000	31	A	90%	1500	3745	3371	3512	3161
-1050	-350	10000	32	A	90%	1500	4241	3817	4376	3938
-1050	-450	10000	32	A	90%	1500	2336	2102	2882	2594
-1050	-550	10000	32	A	90%	1500	2505	2255	2857	2572
-1050	-650	10000	33	A	90%	1500	2587	2328	2761	2485
-1050	-750	10000	33	A	90%	1500	2645	2381	2967	2671
-1050	-850	10000	33	A	90%	1500	2795	2516	3373	3036
-1050	-950	10000	34	A	90%	1500	4415	3974	3616	3254
-1050	-1050	10000	34	A	90%	1500	3080	2772	3222	2900
-1050	-1150	10000	34	A	90%	1500	3594	3235	4297	3867
-1050	-1250	10000	35	A	90%	1500	6122	5510	4439	3995

X	Y	Area	Panel	Zone	Rec%	m^2/day	Actual	Rec	Estimate	Rec
-1050	-1350	10000	35	C	70%	2000	3539	2477	2800	1960
-1050	-1450	10000	35	C	70%	2000	2274	1592	2309	1616
-1050	-1550	10000	36	C	70%	2000	3269	2288	2771	1940
-1050	-1650	10000	36	C	70%	2000	2412	1688	2477	1734
-1050	-1750	10000	36	C	70%	2000	2167	1517	2612	1828
-1050	-1850	10000	37	C	70%	2000	2433	1703	2561	1793
-1050	-1950	10000	37	C	70%	2000	1744	1221	2107	1475
-1050	-2050	10000	37	C	70%	2000	1850	1295	2104	1473
-1050	-2150	10000	38	C	70%	2000	1981	1387	1887	1321
-1050	-2250	10000	38	C	70%	2000	3265	2286	2750	1925
-1050	-2350	10000	38	C	70%	2000	4853	3397	3679	2575
-1050	-2450	10000	39	C	70%	2000	4600	3220	3844	2691
-1050	-2550	10000	39	C	70%	2000	2399	1679	2957	2070
-1050	-2650	10000	39	C	70%	2000	3783	2648	5060	3542
-1050	-2750	10000	40	C	70%	2000	6481	4537	6815	4771
-1050	-2850	10000	40	C	70%	2000	4758	3331	4276	2994
-1150	-50	10000	31	A	90%	1500	3347	3012	2847	2563
-1150	-150	10000	31	A	90%	1500	1915	1724	2154	1939
-1150	-250	10000	31	A	90%	1500	2271	2044	2962	2666
-1150	-350	10000	32	A	90%	1500	4180	3762	4307	3876
-1150	-450	10000	32	A	90%	1500	2686	2417	3190	2871
-1150	-550	10000	32	A	90%	1500	3277	2949	3496	3147
-1150	-650	10000	33	A	90%	1500	2394	2155	3063	2757
-1150	-750	10000	33	A	90%	1500	3607	3246	3555	3200
-1150	-850	10000	33	A	90%	1500	4216	3794	3439	3095
-1150	-950	10000	34	A	90%	1500	2971	2674	3060	2754
-1150	-1050	10000	34	A	90%	1500	3257	2931	3279	2951
-1150	-1150	10000	34	A	90%	1500	3687	3318	3880	3492
-1150	-1250	10000	35	A	90%	1500	3348	3013	3800	3420
-1150	-1350	10000	35	C	70%	2000	2993	2095	2834	1984
-1150	-1450	10000	35	C	70%	2000	2632	1842	2768	1938
-1150	-1550	10000	36	C	70%	2000	3086	2160	3086	2160
-1150	-1650	10000	36	C	70%	2000	2872	2010	3247	2273
-1150	-1750	10000	36	C	70%	2000	2446	1712	3054	2138
-1150	-1850	10000	37	C	70%	2000	2461	1723	2350	1645
-1150	-1950	10000	37	C	70%	2000	2172	1520	2524	1767
-1150	-2050	10000	37	C	70%	2000	2216	1551	1965	1376
-1150	-2150	10000	38	C	70%	2000	1581	1107	1796	1257
-1150	-2250	10000	38	C	70%	2000	2448	1714	2486	1740
-1150	-2350	10000	38	C	70%	2000	3910	2737	3787	2651
-1150	-2450	10000	39	C	70%	2000	4326	3028	3621	2534
-1150	-2550	10000	39	C	70%	2000	2733	1913	3253	2277
-1150	-2650	10000	39	C	70%	2000	4982	3487	4981	3487
-1150	-2750	10000	40	C	70%	2000	7983	5588	6392	4475
-1150	-2850	10000	40	C	70%	2000	6021	4215	4517	3162
-1250	-50	10000	41	A	90%	1500	2867	2580	2432	2189
-1250	-150	10000	41	A	90%	1500	2029	1826	1887	1699
-1250	-250	10000	41	A	90%	1500	2328	2095	2595	2336
-1250	-350	10000	42	A	90%	1500	2821	2539	4232	3808
-1250	-450	10000	42	A	90%	1500	2885	2597	4035	3631
-1250	-550	10000	42	A	90%	1500	5103	4593	4830	4347
-1250	-650	10000	43	A	90%	1500	4536	4082	3901	3511
-1250	-750	10000	43	A	90%	1500	3294	2965	2831	2548
-1250	-850	10000	43	A	90%	1500	3984	3586	2769	2492
-1250	-950	10000	44	A	90%	1500	3904	3514	3406	3065
-1250	-1050	10000	44	A	90%	1500	4198	3778	3896	3506
-1250	-1150	10000	44	A	90%	1500	4621	4159	4649	4184
-1250	-1250	10000	45	A	90%	1500	3886	3497	4357	3921
-1250	-1350	10000	45	C	70%	2000	2920	2044	3103	2172
-1250	-1450	10000	45	C	70%	2000	3967	2777	2944	2061
-1250	-1550	10000	46	C	70%	2000	4132	2892	3271	2290
-1250	-1650	10000	46	C	70%	2000	4240	2968	4163	2914

X	Y	Area	Panel	Zone	Rec%	m^2/day	Actual	Rec	Estimate	Rec
-1250	-1750	10000	46	C	70%	2000	2488	1742	2526	1768
-1250	-1850	10000	47	C	70%	2000	1731	1212	1983	1388
-1250	-1950	10000	47	C	70%	2000	3528	2470	3142	2200
-1250	-2050	10000	47	C	70%	2000	2583	1808	2728	1909
-1250	-2150	10000	48	C	70%	2000	2138	1497	2058	1441
-1250	-2250	10000	48	C	70%	2000	2785	1950	2543	1780
-1250	-2350	10000	48	C	70%	2000	3550	2485	3199	2239
-1250	-2450	10000	49	C	70%	2000	2820	1974	2626	1838
-1250	-2550	10000	49	C	70%	2000	1890	1323	2651	1856
-1250	-2650	10000	49	C	70%	2000	2832	1982	3193	2235
-1250	-2750	10000	50	C	70%	2000	3471	2430	4035	2824
-1250	-2850	10000	50	C	70%	2000	3505	2454	4236	2965
-1350	-50	10000	41	A	90%	1500	2311	2080	2575	2318
-1350	-150	10000	41	A	90%	1500	2160	1944	3000	2700
-1350	-250	10000	41	A	90%	1500	2993	2694	3607	3247
-1350	-350	10000	42	A	90%	1500	3580	3222	3891	3502
-1350	-450	10000	42	A	90%	1500	4012	3611	4077	3669
-1350	-550	10000	42	A	90%	1500	3810	3429	4330	3897
-1350	-650	10000	43	A	90%	1500	4030	3627	4086	3678
-1350	-750	10000	43	A	90%	1500	3812	3431	3692	3323
-1350	-850	10000	43	A	90%	1500	4964	4468	3872	3485
-1350	-950	10000	44	A	90%	1500	4658	4192	4335	3901
-1350	-1050	10000	44	A	90%	1500	4449	4004	5418	4876
-1350	-1150	10000	44	A	90%	1500	7807	7026	7767	6991
-1350	-1250	10000	45	A	90%	1500	4520	4068	4998	4498
-1350	-1350	10000	45	C	70%	2000	4405	3084	3826	2678
-1350	-1450	10000	45	C	70%	2000	4434	3104	3634	2543
-1350	-1550	10000	46	C	70%	2000	3606	2524	3324	2327
-1350	-1650	10000	46	C	70%	2000	3819	2673	3953	2767
-1350	-1750	10000	46	C	70%	2000	2117	1482	2362	1654
-1350	-1850	10000	47	C	70%	2000	1974	1382	1865	1305
-1350	-1950	10000	47	C	70%	2000	2111	1478	2555	1789
-1350	-2050	10000	47	C	70%	2000	2987	2091	2564	1795
-1350	-2150	10000	48	C	70%	2000	2309	1616	1955	1369
-1350	-2250	10000	48	C	70%	2000	2532	1772	2083	1458
-1350	-2350	10000	48	C	70%	2000	1978	1385	2261	1582
-1350	-2450	10000	49	C	70%	2000	1685	1180	1793	1255
-1350	-2550	10000	49	C	70%	2000	2058	1441	2475	1733
-1350	-2650	10000	49	C	70%	2000	3835	2685	3446	2412
-1350	-2750	10000	50	C	70%	2000	3698	2589	3407	2385
-1350	-2850	10000	50	C	70%	2000	4405	3084	4038	2826
-1450	-50	10000	41	A	90%	1500	3906	3515	3077	2769
-1450	-150	10000	41	A	90%	1500	4279	3851	3952	3557
-1450	-250	10000	41	A	90%	1500	3237	2913	3506	3155
-1450	-350	10000	42	A	90%	1500	3447	3102	3210	2889
-1450	-450	10000	42	A	90%	1500	3302	2972	3603	3243
-1450	-550	10000	42	A	90%	1500	3260	2934	3671	3304
-1450	-650	10000	43	A	90%	1500	3647	3282	4223	3801
-1450	-750	10000	43	A	90%	1500	4752	4277	4732	4259
-1450	-850	10000	43	A	90%	1500	4976	4478	4379	3941
-1450	-950	10000	44	A	90%	1500	2949	2654	3352	3017
-1450	-1050	10000	44	A	90%	1500	4226	3803	4723	4250
-1450	-1150	10000	44	A	90%	1500	5693	5124	6695	6025
-1450	-1250	10000	45	A	90%	1500	4256	3830	4720	4248
-1450	-1350	10000	45	C	70%	2000	4319	3023	4250	2975
-1450	-1450	10000	45	C	70%	2000	3052	2136	3443	2410
-1450	-1550	10000	46	C	70%	2000	2526	1768	2383	1668
-1450	-1650	10000	46	C	70%	2000	2514	1760	2825	1977
-1450	-1750	10000	46	C	70%	2000	3410	2387	3192	2235
-1450	-1850	10000	47	C	70%	2000	4716	3301	3038	2126
-1450	-1950	10000	47	C	70%	2000	3269	2288	3743	2620
-1450	-2050	10000	47	C	70%	2000	2308	1616	2674	1872

X	Y	Area	Panel	Zone	Rec%	m^2/day	Actual	Rec	Estimate	Rec
-1450	-2150	10000	48	C	70%	2000	2594	1816	1962	1373
-1450	-2250	10000	48	C	70%	2000	2061	1443	2045	1432
-1450	-2350	10000	48	C	70%	2000	2084	1459	2299	1609
-1450	-2450	10000	49	C	70%	2000	2983	2088	2388	1672
-1450	-2550	10000	49	C	70%	2000	2300	1610	2454	1718
-1450	-2650	10000	49	C	70%	2000	2959	2071	3546	2482
-1450	-2750	10000	50	C	70%	2000	2732	1912	2971	2080
-1450	-2850	10000	50	C	70%	2000	3536	2475	3491	2444
-1550	-50	10000	51	A	90%	1500	4313	3882	3967	3570
-1550	-150	10000	51	A	90%	1500	3942	3548	3801	3421
-1550	-250	10000	51	A	90%	1500	2898	2608	2896	2607
-1550	-350	10000	52	A	90%	1500	3969	3572	2794	2515
-1550	-450	10000	52	A	90%	1500	4001	3601	3580	3222
-1550	-550	10000	52	A	90%	1500	3860	3474	4910	4419
-1550	-650	10000	53	A	90%	1500	4401	3961	5533	4980
-1550	-750	10000	53	A	90%	1500	6358	5722	6160	5544
-1550	-850	10000	53	A	90%	1500	4765	4289	4493	4043
-1550	-950	10000	54	A	90%	1500	3237	2913	3089	2780
-1550	-1050	10000	54	A	90%	1500	5282	4754	4611	4150
-1550	-1150	10000	54	A	90%	1500	5114	4603	5030	4527
-1550	-1250	10000	55	A	90%	1500	4118	3706	4395	3956
-1550	-1350	10000	55	C	70%	2000	3865	2706	3925	2748
-1550	-1450	10000	55	C	70%	2000	3140	2198	3432	2402
-1550	-1550	10000	56	C	70%	2000	1707	1195	1881	1317
-1550	-1650	10000	56	C	70%	2000	2010	1407	2100	1470
-1550	-1750	10000	56	C	70%	2000	2981	2087	3361	2352
-1550	-1850	10000	57	C	70%	2000	3520	2464	3062	2143
-1550	-1950	10000	57	C	70%	2000	3794	2656	3742	2619
-1550	-2050	10000	57	C	70%	2000	1798	1259	2337	1636
-1550	-2150	10000	58	C	70%	2000	1565	1096	2085	1460
-1550	-2250	10000	58	C	70%	2000	2317	1622	2529	1770
-1550	-2350	10000	58	C	70%	2000	2461	1723	2152	1507
-1550	-2450	10000	59	C	70%	2000	2612	1828	2298	1609
-1550	-2550	10000	59	C	70%	2000	2448	1714	2379	1665
-1550	-2650	10000	59	C	70%	2000	2971	2080	2944	2060
-1550	-2750	10000	60	C	70%	2000	2581	1807	2736	1915
-1550	-2850	10000	60	C	70%	2000	4628	3240	3120	2184
-1650	-50	10000	51	A	90%	1500	3127	2814	3178	2860
-1650	-150	10000	51	A	90%	1500	3992	3593	3126	2813
-1650	-250	10000	51	A	90%	1500	3732	3359	3501	3151
-1650	-350	10000	52	A	90%	1500	3945	3551	3090	2781
-1650	-450	10000	52	A	90%	1500	2969	2672	2832	2549
-1650	-550	10000	52	A	90%	1500	3208	2887	3933	3539
-1650	-650	10000	53	A	90%	1500	4556	4100	5356	4821
-1650	-750	10000	53	A	90%	1500	6739	6065	5422	4880
-1650	-850	10000	53	A	90%	1500	5350	4815	3899	3509
-1650	-950	10000	54	A	90%	1500	3234	2911	3116	2805
-1650	-1050	10000	54	A	90%	1500	6339	5705	5558	5002
-1650	-1150	10000	54	A	90%	1500	3508	3157	3910	3519
-1650	-1250	10000	55	A	90%	1500	2118	1906	3011	2710
-1650	-1350	10000	55	C	70%	2000	2793	1955	2937	2056
-1650	-1450	10000	55	C	70%	2000	2864	2005	2874	2012
-1650	-1550	10000	56	C	70%	2000	1324	927	1635	1145
-1650	-1650	10000	56	C	70%	2000	2009	1406	2024	1417
-1650	-1750	10000	56	C	70%	2000	2516	1761	2178	1524
-1650	-1850	10000	57	C	70%	2000	2249	1574	1672	1170
-1650	-1950	10000	57	C	70%	2000	2826	1978	2642	1849
-1650	-2050	10000	57	C	70%	2000	2874	2012	3061	2143
-1650	-2150	10000	58	C	70%	2000	2126	1488	2309	1617
-1650	-2250	10000	58	C	70%	2000	2462	1723	2762	1933
-1650	-2350	10000	58	C	70%	2000	3597	2518	3107	2175
-1650	-2450	10000	59	C	70%	2000	3444	2411	2800	1960

X	Y	Area	Panel	Zone	Rec%	m^2/day	Actual	Rec	Estimate	Rec
-1650	-2550	10000	59	C	70%	2000	2300	1610	2304	1613
-1650	-2650	10000	59	C	70%	2000	2936	2055	1926	1348
-1650	-2750	10000	60	C	70%	2000	2427	1699	2441	1709
-1650	-2850	10000	60	C	70%	2000	3929	2750	3849	2694
-1750	-50	10000	51	A	90%	1500	3901	3511	3102	2791
-1750	-150	10000	51	A	90%	1500	2351	2116	2485	2237
-1750	-250	10000	51	A	90%	1500	2629	2366	2717	2445
-1750	-350	10000	52	A	90%	1500	3385	3047	2666	2399
-1750	-450	10000	52	A	90%	1500	3097	2787	3039	2735
-1750	-550	10000	52	A	90%	1500	4443	3999	4158	3742
-1750	-650	10000	53	A	90%	1500	5971	5374	5718	5146
-1750	-750	10000	53	A	90%	1500	6518	5866	4709	4238
-1750	-850	10000	53	A	90%	1500	5320	4788	3947	3553
-1750	-950	10000	54	A	90%	1500	4065	3659	3816	3435
-1750	-1050	10000	54	A	90%	1500	4067	3660	4704	4234
-1750	-1150	10000	54	A	90%	1500	3128	2815	3381	3043
-1750	-1250	10000	55	A	90%	1500	1574	1417	1830	1647
-1750	-1350	10000	55	C	70%	2000	1617	1132	1522	1066
-1750	-1450	10000	55	C	70%	2000	2857	2000	2617	1832
-1750	-1550	10000	56	C	70%	2000	2722	1905	2048	1434
-1750	-1650	10000	56	C	70%	2000	1711	1198	2030	1421
-1750	-1750	10000	56	C	70%	2000	1889	1322	2174	1522
-1750	-1850	10000	57	C	70%	2000	2386	1670	2354	1648
-1750	-1950	10000	57	C	70%	2000	3174	2222	3340	2338
-1750	-2050	10000	57	C	70%	2000	4493	3145	3758	2630
-1750	-2150	10000	58	C	70%	2000	2622	1835	2399	1679
-1750	-2250	10000	58	C	70%	2000	2907	2035	2548	1783
-1750	-2350	10000	58	C	70%	2000	3267	2287	2866	2006
-1750	-2450	10000	59	C	70%	2000	3672	2570	2860	2002
-1750	-2550	10000	59	C	70%	2000	2498	1749	2278	1595
-1750	-2650	10000	59	C	70%	2000	2136	1495	2292	1604
-1750	-2750	10000	60	C	70%	2000	1918	1343	2312	1619
-1750	-2850	10000	60	C	70%	2000	3442	2409	3809	2666
-1850	-50	10000	61	A	90%	1500	1537	1383	1476	1328
-1850	-150	10000	61	A	90%	1500	1346	1211	1709	1538
-1850	-250	10000	61	A	90%	1500	2421	2179	2505	2254
-1850	-350	10000	62	A	90%	1500	3086	2777	2667	2400
-1850	-450	10000	62	A	90%	1500	3143	2829	3373	3036
-1850	-550	10000	62	A	90%	1500	4609	4148	4633	4169
-1850	-650	10000	63	A	90%	1500	5628	5065	5332	4799
-1850	-750	10000	63	A	90%	1500	4781	4303	4444	4000
-1850	-850	10000	63	A	90%	1500	3518	3166	3828	3445
-1850	-950	10000	64	A	90%	1500	3058	2752	3539	3185
-1850	-1050	10000	64	A	90%	1500	3629	3266	2566	2309
-1850	-1150	10000	64	A	90%	1500	2071	1864	2558	2302
-1850	-1250	10000	65	A	90%	1500	1034	931	1533	1380
-1850	-1350	10000	65	C	70%	2000	1330	931	1399	979
-1850	-1450	10000	65	C	70%	2000	2338	1637	2742	1919
-1850	-1550	10000	66	C	70%	2000	1865	1306	1862	1303
-1850	-1650	10000	66	C	70%	2000	1258	881	1511	1058
-1850	-1750	10000	66	C	70%	2000	1904	1333	1998	1398
-1850	-1850	10000	67	C	70%	2000	2672	1870	2066	1446
-1850	-1950	10000	67	C	70%	2000	3474	2432	3490	2443
-1850	-2050	10000	67	C	70%	2000	3055	2139	2969	2078
-1850	-2150	10000	68	C	70%	2000	2382	1667	2164	1515
-1850	-2250	10000	68	C	70%	2000	2742	1919	2468	1728
-1850	-2350	10000	68	C	70%	2000	3173	2221	2266	1586
-1850	-2450	10000	69	C	70%	2000	2816	1971	2939	2057
-1850	-2550	10000	69	C	70%	2000	3694	2586	3501	2451
-1850	-2650	10000	69	C	70%	2000	4339	3037	4083	2858
-1850	-2750	10000	70	C	70%	2000	2134	1494	3025	2118
-1850	-2850	10000	70	C	70%	2000	3002	2101	3114	2180

X	Y	Area	Panel	Zone	Rec%	m^2/day	Actual	Rec	Estimate	Rec
-1950	-50	10000	61	A	90%	1500	3147	2832	2046	1842
-1950	-150	10000	61	A	90%	1500	2793	2514	2370	2133
-1950	-250	10000	61	A	90%	1500	2777	2499	2786	2507
-1950	-350	10000	62	A	90%	1500	3272	2945	3236	2912
-1950	-450	10000	62	A	90%	1500	2958	2662	3479	3131
-1950	-550	10000	62	A	90%	1500	3594	3235	3909	3518
-1950	-650	10000	63	A	90%	1500	3503	3153	4116	3705
-1950	-750	10000	63	A	90%	1500	3563	3207	3447	3102
-1950	-850	10000	63	A	90%	1500	3333	3000	3215	2894
-1950	-950	10000	64	A	90%	1500	2196	1976	2752	2477
-1950	-1050	10000	64	A	90%	1500	3142	2828	2185	1966
-1950	-1150	10000	64	A	90%	1500	3201	2881	2436	2193
-1950	-1250	10000	65	A	90%	1500	3305	2975	2725	2452
-1950	-1350	10000	65	C	70%	2000	2251	1576	2168	1517
-1950	-1450	10000	65	C	70%	2000	1991	1394	2441	1709
-1950	-1550	10000	66	C	70%	2000	1249	874	1482	1037
-1950	-1650	10000	66	C	70%	2000	2106	1474	1837	1286
-1950	-1750	10000	66	C	70%	2000	1470	1029	1754	1228
-1950	-1850	10000	67	C	70%	2000	895	627	1252	876
-1950	-1950	10000	67	C	70%	2000	1582	1107	2056	1439
-1950	-2050	10000	67	C	70%	2000	2306	1614	1995	1396
-1950	-2150	10000	68	C	70%	2000	2014	1410	1793	1255
-1950	-2250	10000	68	C	70%	2000	2430	1701	1997	1398
-1950	-2350	10000	68	C	70%	2000	1319	923	1292	904
-1950	-2450	10000	69	C	70%	2000	866	606	1544	1081
-1950	-2550	10000	69	C	70%	2000	3110	2177	3062	2143
-1950	-2650	10000	69	C	70%	2000	3647	2553	3611	2528
-1950	-2750	10000	70	C	70%	2000	3999	2799	4070	2849
-1950	-2850	10000	70	C	70%	2000	4108	2876	3728	2610
-2050	-50	10000	61	A	90%	1500	1263	1137	1827	1645
-2050	-150	10000	61	A	90%	1500	1406	1265	1848	1663
-2050	-250	10000	61	A	90%	1500	2270	2043	2276	2048
-2050	-350	10000	62	A	90%	1500	2921	2629	2891	2602
-2050	-450	10000	62	A	90%	1500	4311	3880	3985	3586
-2050	-550	10000	62	A	90%	1500	5031	4528	4800	4320
-2050	-650	10000	63	A	90%	1500	4521	4069	4773	4296
-2050	-750	10000	63	A	90%	1500	2876	2588	2726	2453
-2050	-850	10000	63	A	90%	1500	2106	1895	1989	1790
-2050	-950	10000	64	A	90%	1500	2507	2256	2665	2399
-2050	-1050	10000	64	A	90%	1500	3036	2732	2758	2482
-2050	-1150	10000	64	A	90%	1500	2276	2048	2527	2275
-2050	-1250	10000	65	A	90%	1500	3518	3166	3093	2784
-2050	-1350	10000	65	C	70%	2000	2234	1564	2036	1425
-2050	-1450	10000	65	C	70%	2000	1466	1026	1256	879
-2050	-1550	10000	66	C	70%	2000	1113	779	1197	838
-2050	-1650	10000	66	C	70%	2000	1471	1030	1566	1096
-2050	-1750	10000	66	C	70%	2000	2271	1590	1937	1356
-2050	-1850	10000	67	C	70%	2000	2201	1541	1770	1239
-2050	-1950	10000	67	C	70%	2000	1782	1247	1889	1322
-2050	-2050	10000	67	C	70%	2000	2030	1421	1599	1120
-2050	-2150	10000	68	C	70%	2000	1334	934	1185	829
-2050	-2250	10000	68	C	70%	2000	1791	1254	1325	928
-2050	-2350	10000	68	C	70%	2000	1405	984	1140	798
-2050	-2450	10000	69	C	70%	2000	2362	1653	1880	1316
-2050	-2550	10000	69	C	70%	2000	3456	2419	3674	2572
-2050	-2650	10000	69	C	70%	2000	2506	1754	3318	2322
-2050	-2750	10000	70	C	70%	2000	4106	2874	4646	3252
-2050	-2850	10000	70	C	70%	2000	5825	4078	4726	3308

Table 14. A portion of the Block Calculation table. Block values were fed onto this table if the estimated production rate were above 300 stones/day. If this was not the case, a zero value was returned as these blocks were not to be mined.

Block	Recovered Stones	Actual Rate (days)	Estimated Stones	Estimated Rate (days)	Panel
1	4730	6.7	3584	6.7	1
2	2837	6.7	2367	6.7	1
3	1635	6.7	2232	6.7	1
4	1922	6.7	2096	6.7	2
5	3049	6.7	2736	6.7	2
6	3253	6.7	3254	6.7	2
7	2984	6.7	3506	6.7	3
8	2649	6.7	3538	6.7	3
9	3024	6.7	2810	6.7	3
10	0	0.0	0	0.0	4
11	0	0.0	0	0.0	4
12	2948	6.7	3079	6.7	4
13	3172	6.7	3360	6.7	5
14	2643	5.6	2501	5.6	5
15	0	0.0	0	0.0	5
16	0	0.0	0	0.0	6
17	1725	5.6	1822	5.6	6
18	3091	5.6	2354	5.6	6
19	3258	5.6	2344	5.6	7
20	2102	5.6	1807	5.6	7
21	1758	5.6	1888	5.6	7
22	2131	5.6	2207	5.6	8
23	0	0.0	0	0.0	8
24	0	0.0	0	0.0	8
25	0	0.0	0	0.0	9
26	2508	5.6	1878	5.6	9
27	0	0.0	0	0.0	9
28	0	0.0	0	0.0	10
29	0	0.0	0	0.0	10
30	3082	6.7	3402	6.7	1
31	2641	6.7	2481	6.7	1
32	2545	6.7	2705	6.7	1
33	2362	6.7	2521	6.7	2
34	2735	6.7	3163	6.7	2
35	2713	6.7	3261	6.7	2
36	3739	6.7	3210	6.7	3
37	2541	6.7	2992	6.7	3
38	2966	6.7	2937	6.7	3
39	0	0.0	0	0.0	4
40	0	0.0	0	0.0	4
41	2575	6.7	2627	6.7	4
42	3553	6.7	3561	6.7	5

43	3404	5.6	3056	5.6	5
44	0	0.0	0	0.0	5
45	0	0.0	0	0.0	6
46	1903	5.6	2104	5.6	6
47	3994	5.6	3575	5.6	6
48	3945	5.6	3493	5.6	7
49	2578	5.6	2557	5.6	7
50	1866	5.6	2253	5.6	7
51	1731	5.6	1869	5.6	8
52	0	0.0	0	0.0	8
53	1459	5.6	1766	5.6	8
Etc.					

Table 15. The Panel Contribution table before it is sorted.

Est st/day	Panel	Rec. Stones	Total Days
403.5	1	21738	53
407.5	2	24679	60
435.5	3	25074	60
377.5	4	9455	27
458.4	5	19824	42
462.1	6	18822	39
454.2	7	23739	50
358.7	8	10178	28
376.9	9	10666	28
351.6	10	7462	22
343.1	11	24035	60
402.8	12	23281	60
471.2	13	27823	60
623.8	14	36911	60
442.8	15	22254	48
399.1	16	20117	44
367.9	17	14769	39
316.2	18	5714	22
401.9	19	19768	44
405.1	20	3636	11
439.4	21	24918	60
413.5	22	23048	60
403.4	23	23572	53
534.0	24	30253	60
500.6	25	24851	53
401.3	26	17020	39
398.3	27	20350	50
409.8	28	17530	44
449.5	29	22126	50
364.2	30	3935	11
411.1	31	22587	53
439.2	32	24192	60
402.6	33	20280	53
488.7	34	27812	60
435.6	35	23722	51

393.5	36	17326	47
364.4	37	11219	32
453.4	38	15520	31
534.5	39	23502	47
683.4	40	23602	31
417.6	41	21673	53
538.2	42	28998	60
517.3	43	34196	60
663.6	44	38255	60
550.1	45	27564	50
435.6	46	20196	45
408.9	47	15051	35
360.6	48	7278	20
398.6	49	15174	40
517.5	50	14943	30
431.6	51	27797	60
465.0	52	29589	60
678.6	53	44980	60
558.2	54	34177	60
462.1	55	16476	38
359.9	56	5170	15
425.2	57	17406	40
361.8	58	15231	40
352.7	59	15457	40
426.2	60	13248	30
335.4	61	9235	27
494.6	62	29633	60
538.0	63	28551	53
367.9	64	19777	53
366.4	65	10747	28
0	66	0	0
452.2	67	4570	10
321.9	68	5808	15
483.8	69	16498	35
543.9	70	16222	30

Table 16. Gross Revenue Calculation.

Year	Days	Stones	Gross Revenue
1	365	228422	525,371,060
2	365	186263	428,403,750
3	330	155743	358,208,217
4	365	163723	376,561,926
5	365	158354	364,213,308
6	330	135812	312,368,151
7	365	149122	342,980,320
8	365	135287	311,160,043

ANNEXURE 2

Table 17. Gross revenue for the various scenarios investigated.

	Year	1	2	3	4	5	6	7	8
Baseline		533,020,201	431,015,197	365,084,654	387,762,839	371,877,100	321,688,079	339,558,634	322,501,778
Predicted Recovey	100% Scenario	515,818,393	417,647,100	346,390,795	386,012,641	365,441,047	307,802,158	330,072,291	325,279,148
	90% Scenario	515,818,393	417,079,743	355,662,119	377,752,478	378,775,220	295,140,904	351,301,096	329,515,677
	80% Scenario	519,408,877	416,019,630	361,552,600	382,099,336	371,239,659	317,823,847	360,953,366	327,972,525
	70% Scenario	528,182,646	425,221,941	378,378,799	382,577,632	388,419,502	333,831,171	349,197,069	0
	60% Scenario	539,720,423	453,038,740	387,558,869	417,631,304	221,328,534	0	0	0
	50% Scenario	543,992,110	497,474,872	233,930,628	0	0	0	0	0
Predicted Rate	30%	527,427,049	408,412,974	352,313,962	363,717,630	366,337,215	318,259,055	337,120,022	334,354,291
	20%	527,427,049	408,412,974	352,313,962	363,717,630	366,337,215	313,091,461	347,623,140	342,593,863
	Rate at C	527,427,049	408,412,974	367,443,937	356,432,610	361,056,223	314,425,282	353,174,571	344,310,677
	10%	527,427,049	408,412,974	373,947,507	351,457,619	362,401,877	313,562,603	359,074,620	347,526,378
	Rate at B	527,427,049	408,412,974	375,273,439	364,726,108	354,014,313	317,186,663	361,576,377	346,797,913
	-10%	529,600,968	424,236,862	377,586,681	386,624,199	369,587,767	315,951,613	370,141,024	0
	Rate at A	529,600,968	424,236,862	377,586,681	381,984,294	370,309,052	327,709,829	370,827,160	0
	-20%	534,298,848	437,485,422	370,200,625	386,029,615	382,829,811	340,406,076	153,350,794	0
	-30%	538,721,985	442,906,573	379,689,693	401,568,558	390,411,373	0	0	0
Nugget Effect Variations	Best Fit	525,371,060	428,403,750	358,208,217	376,561,926	364,213,308	312,368,151	342,980,320	311,160,043
	NE at 0	520,668,207	438,271,253	361,763,964	382,539,404	380,884,199	314,177,273	330,834,180	241,047,250
	NE under by 50%	523,674,511	437,541,765	358,184,972	389,916,508	362,915,831	314,025,383	337,519,733	313,631,542
	NE over by 50%	523,519,368	426,136,285	361,435,850	366,064,623	358,618,685	306,683,538	335,884,061	321,490,749
	Moving Average	514,839,073	438,892,421	357,635,643	347,031,227	357,356,791	300,782,101	322,042,338	331,802,658
Range variations	Range 50% shorter	523,519,368	426,421,725	361,150,411	361,165,344	364,468,630	308,446,004	342,498,578	317,343,113
	Range 25% shorter	527,032,990	426,741,820	358,208,217	376,224,621	363,062,782	298,160,481	356,411,231	307,227,048
	Best Fit	525,371,060	428,403,750	358,208,217	376,561,926	364,213,308	312,368,151	342,980,320	311,160,043
	Range 25% longer	525,371,060	428,403,750	367,646,440	372,105,005	362,814,869	309,382,986	343,131,388	313,102,194
	Range 50% longer	525,371,060	430,833,815	366,827,715	372,158,800	360,500,981	311,552,729	344,991,444	312,268,718

Sample density	25X25	529,332,036	425,584,418	360,775,695	383,267,719	372,941,056	321,859,668	335,261,067	325,467,712
	50X50	525,371,060	428,403,750	358,208,217	376,561,926	364,213,308	312,368,151	342,980,320	311,160,043
	100X100	512,306,010	431,357,847	360,571,728	351,356,017	348,799,518	307,959,406	322,834,642	312,364,806
	200X200	517,628,026	428,316,861	358,290,698	353,418,208	349,037,634	306,530,737	321,004,563	311,785,774
Sample support size	2.5m	513,360,633	432,170,220	353,982,372	354,147,031	344,008,194	316,849,303	320,613,969	328,992,391
	5m	515,879,650	428,230,253	365,039,671	372,826,619	362,967,223	310,561,947	342,020,105	297,768,212
	10m	525,371,060	428,403,750	358,208,217	376,561,926	364,213,308	312,368,151	342,980,320	311,160,043
	20m	524,264,053	430,249,686	361,327,019	379,831,779	367,547,485	314,261,523	339,824,399	323,126,078

Table 18. Gross revenue ratios for the various scenarios investigated. The gross revenue ratio was calculated by dividing the gross revenue in a specific year by the baseline gross revenue for that year.

	Year		1	2	3	4	5	6	7	8
Baseline			1	1	1	1	1	1	1	1
Predicted Recovey	100% Scenario		0.97	0.97	0.95	1.00	0.98	0.96	0.97	1.01
	90% Scenario		0.97	0.97	0.97	0.97	1.02	0.92	1.03	1.02
	80% Scenario		0.97	0.97	0.99	0.99	1.00	0.99	1.06	1.02
	70% Scenario		0.99	0.99	1.04	0.99	1.04	1.04	1.03	0
	60% Scenario		1.01	1.05	1.06	1.08	0.60	0	0	0
	50% Scenario		1.02	1.15	0.64	0	0	0	0	0
Predicted Rate		30%	0.99	0.95	0.97	0.94	0.99	0.99	0.99	1.04
		20%	0.99	0.95	0.97	0.94	0.99	0.97	1.02	1.06
	Rate at C		0.99	0.95	1.01	0.92	0.97	0.98	1.04	1.07
		10%	0.99	0.95	1.02	0.91	0.97	0.97	1.06	1.08
	Rate at B		0.99	0.95	1.03	0.94	0.95	0.99	1.06	1.08
		-10%	0.99	0.98	1.03	1.00	0.99	0.98	1.09	0
	Rate at A		0.99	0.98	1.03	0.99	1.00	1.02	1.09	0
		-20%	1.00	1.02	1.01	1.00	1.03	1.06	0.45	0
		-30%	1.01	1.03	1.04	1.04	1.05	0	0	0

Nugget Effect Variations	Best Fit	0.99	0.99	0.98	0.97	0.98	0.97	1.01	0.96
	NE at 0	0.98	1.02	0.99	0.99	1.02	0.98	0.97	0.75
	NE under by 50%	0.98	1.02	0.98	1.01	0.98	0.98	0.99	0.97
	NE over by 50%	0.98	0.99	0.99	0.94	0.96	0.95	0.99	1.00
	Moving Average	0.97	1.02	0.98	0.89	0.96	0.94	0.95	1.03
Range variations	Range 50% shorter	0.98	0.99	0.99	0.93	0.98	0.96	1.01	0.98
	Range 25% shorter	0.99	0.99	0.98	0.97	0.98	0.93	1.05	0.95
	Best Fit	0.99	0.99	0.98	0.97	0.98	0.97	1.01	0.96
	Range 25% longer	0.99	0.99	1.01	0.96	0.98	0.96	1.01	0.97
	Range 50% longer	0.99	1.00	1.00	0.96	0.97	0.97	1.02	0.97
Sample density	25X25	0.99	0.99	0.99	0.99	1.00	1.00	0.99	1.01
	50X50	0.99	0.99	0.98	0.97	0.98	0.97	1.01	0.96
	100X100	0.96	1.00	0.99	0.91	0.94	0.96	0.95	0.97
	200X200	0.97	0.99	0.98	0.91	0.94	0.95	0.95	0.97
Sample support size	2.5m	0.96	1.00	0.97	0.91	0.93	0.98	0.94	1.02
	5m	0.97	0.99	1.00	0.96	0.98	0.97	1.01	0.92
	10m	0.99	0.99	0.98	0.97	0.98	0.97	1.01	0.96
	20m	0.98	1.00	0.99	0.98	0.99	0.98	1.00	1.00