STATISTICAL BENCHMARK SURVEYING OF PRODUCTION CONCENTRATORS

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DECLARATION

I, Norman Owen Lotter, declare that the work reported in this thesis is my own work, and does represent original application of engineering and statistical principles to the subject matter. Where the lognormal models of Krige, Sichel, and others were developed in the 1950's and 1960's in South Africa for application to the spatial dimension, very limited work has been done to date on their application to the time dimension. Also, extended interpretations of compound distributions in the time dimension have been made, based on their parent models in space, after Clark. These features, in part, form the contribution to knowledge in this work. Earlier concentrator survey models were developed by myself in collaboration with others in South Africa in the early 1990's, and were improved at Falconbridge by the author from 1997 to 2001. These were not published, but have been acknowledged in the text as the heuristic foundations to the present work.

Whereas existing engineering tools have applications within their own disciplines, this work will show that by importing relevant segments of other disciplines into an integrated whole, an improved ability to assess and interpret an existing concentrator's performance is realised. Further work has been identified.

This work is a logical extension from my earlier work based at the University of Cape Town, South Africa, and which focussed on the development and proof of improved sampling and flotation testing methodology **[Lotter, 1995a]**.

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The technicians and technologists of our Technology Centre at Sudbury deserve special mention in this and previous work. The practical application at Falconbridge of High-Confidence Flotation Testing from my earlier work in South Africa, of the heuristic Falconbridge surveying model 1997-2000, and of the new campaign surveying model, is sincerely appreciated.

I wish to acknowledge my dear wife Heather, for her patience and understanding during these four years of part-time studies. She has always recognised and supported the value of further education in our life together.

SUMMARY

The sampling and analysis of sulphide mineral processing plants is addressed in this study. A review of the published literature has shown that the foundations of this topic were laid in the 1970's, but typically a single sampling test was performed, and its representativity accepted provided its metallurgical balance closed without excessive adjustments. There was no mention made of quality control or equivalent tests of representativity of the feed material during sampling tests. No recognition of the effect(s) of ore grade on metallurgical performance was given.

In this study, a quantitative model, called a *statistical benchmark survey,* is presented. Multiple surveys are completed over a limited time; the corresponding stream samples of the surveys deemed acceptable are combined to obtain high confidence composite samples. The head grade of each survey is compared to two distributions to test its acceptability, typically at a 95% confidence level. These distributions are called the Internal Reference Distribution and the External Reference Distribution.

The first test – on the Internal Reference Distribution - uses the Sichel t-estimator, a lognormal model designed for use on small data sets, on the set of six survey unit head grades. The associated confidence limits of this mean grade are equivalent to two standard errors of the distribution, but are skewed about the sample mean. The second test, this time by the External Reference Distribution, also uses a lognormal platform, designed by Krige, but uses larger data sets from 1-3 months of shift sample head grades. The associated confidence limits of this second model are also skewed, but are wider than for the Sichel model, and are equivalent to two standard deviations of the sample mean. This outlier rejection model produces ore grade estimates that are in good agreement with the more robust External Reference Distribution means.

The Raglan Mine case study is used to illustrate that ore grades in situ are highly lognormal; this lognormality is also present in the time domain in head samples (taken at the cyclone overflow), but is less pronounced (i.e. residual).

Two survey models are presented. The *benchmark model* describes typical operations. The *campaign model* specifically chooses ore types that are mined and milled in a specific week of operations for predictive or diagnostic purposes.

The multiple mineral hosting of nickel across three orders of magnitude extends this problem into that of a compound distribution. The construction and use of an External Reference Distribution to estimate the mean and associated skew confidence limits of this compound distribution is shown for both drill core and ore milled (the latter in a case of residual lognormality). A trial decomposition of the spatial External Reference Distribution is discussed. The heterogeneous nickel mineral hosting in ore, after processing, becomes an artificially controlled final concentrate, containing most of the economic nickel sulphides in a normal distribution, and most of the uneconomic nickel minerals in a final tailing with a residually bimodal lognormal distribution.

The presence of bimodal lognormality in final tailing data may have historical or predictive uses: at Raglan, flowsheet improvements and more seasoned operations contributed to the decrease in the mean of both the low-grade and high-grade modes, and increase the contribution of the low-grade mode.

SOMMAIRE

L'échantillonnage et l'analyse des procédés minéralurgiques, et en particulier ceux traitant des minerais de sulfures, font l'objet de la présente étude. Une revue de la littérature indique que le gros des articles publiés dans le domaine de l'échantillonnage de diagnostic en minéralurgie le furent dans les années 70. La méthodologie proposée dans tous les cas était fondée sur un seul échantillonnage, ce qui limitait l'analyse statistique ou la gestion de la qualité des échantillons obtenus. En particulier, aucun test statistique de la représentativité des échantillons n'est proposé, et l'effet potentiel de la teneur d'alimentation sur l'efficacité de la séparation métallurgique n'est pas traité.

Cette thèse présente un modèle d'échantillonnage statistique de référence, appelé "Statistical Benchmark Survey" en anglais, fondée sur une série de six échantillonnages de circuit. La validité de chaque échantillonnage est évaluée statistiquement (à 95%) en comparant la teneur d'alimentation à deux distributions de teneurs servant de référence (une interne, l'autre externe), et les échantillons correspondants des échantillonnages retenus sont combinés.

Le premier test, fondé sur la distribution de teneur des campagnes elles-mêmes, utilise l'estimateur Sichel t, un modèle lognormal conçu pour les échantillons de petite taille. Ce test produit un intervalle de confiance asymétrique pour la teneur moyenne de la population, qui sert de filtre à la teneur d'alimentation de chacun des échantillonnages.

Le deuxième test est fondé sur une distribution des teneurs d'alimentation d'un à trois mois de postes, dite distribution externe. Cette distribution est de taille plus élevée que la distribution interne, ce qui permet d'utiliser l'approche de Krige, qui elle aussi est basée sur des mathématiques lognormales, afin d'obtenir un intervalle de confiance asymétrique des mesures de teneur d'alimentation. Malgré la taille plus élevée de cette distribution, l'approche de Krige produit un

intervalle de confiance plus large que celui de Sichel, puisqu'il est celui d'une mesure individuelle de teneur plutôt que celui de la teneur moyenne de la population.

Nous proposons deux modèles de campagne d'échantillonnage. Le modèle de référence est conçu pour les échantillonnages visant à caractériser le fonctionnement typique de l'usine, tandis que le modèle de campagne, qui dépend davantage de Sichel, s'applique aux échantillonnages ciblant un laps de temps plus court, typiquement une semaine, et un mode de fonctionnement ou une alimentation d'usine inhabituel qu'on cherche à caractériser.

L'étude de cas de la mine Raglan illustre que la lognormalité prononcée des teneurs in situ est toujours présente dans le temps dans la teneur d'alimentation (à la surverse des cyclones), mais de façon résiduelle.

Le nickel se retrouve à Raglan dans trois minerais types de teneur bien différente, de sorte que la distribution de teneur n'est pas seulement lognormale, mais également composée de deux ou trois distributions primaires. Celles-ci peuvent être estimée par déconvolution. Nous illustrons comment l'histogramme de la distribution de référence externe sert à estimer la moyenne et les limites de confiance (asymétriques) de teneur des carottes et de l'alimentation de la flottation. Le circuit de flottation produit un concentré final à teneur strictement contrôlée, qui est normalement distribuée, et un rejet, dont les teneurs de rejet se distribuent de façon bimodale lognormale. Cette distribution évolue dans le temps, à mesure que le schéma de traitement est amélioré et l'opération de l'usine devient plus efficace. La teneur moyenne des deux modes diminue, de même que la proportion du mode de teneur élevée. Cette caractérisation pourrait permettre de mieux baliser ou même anticiper le potentiel d'amélioration du bilan métallurgique.

PUBLICATIONS FROM THIS THESIS

Lotter, NO, and Laplante, AR, *Residual Effects of Lognormality in Concentrator Data*, Canadian Mineral Processors' Conference, Ottawa, January 2005, Paper No. 20.

Lotter, NO, and Laplante, AR, *Statistical Benchmark Surveying of Production Concentrators*, Minerals Engineering Conference on the Processing of Nickel Ores and Concentrates, Cape Town, November 2005 [In Press]

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CHAPTER 1 - INTRODUCTION AND SCOPE

1.1. Problem Statement

Milling and flotation, sometimes accompanied by gravity separation, are well-established unit processes, which have traditionally been used to treat ore that has been mined. This is because these processes liberate, then separate, the valuable minerals, which contain paymetals, from host gangue. This is often done by a combination of crushing and grinding processes to liberate the paymetal minerals, followed by gravity separation and/or flotation, to separate these valuable minerals into a concentrate, or several concentrates. These concentrate(s) have a lower bulk than the parent ore from which they are derived. Further, they contain a high percentage of the valuable minerals. Accordingly, they are more economically treated in subsequent beneficiation processes than the ore from which they were produced.

The choice of concentration process invariably results in a variable paymetal loss to the concentrator tailing. This is discarded and disposed of in a tailings impoundment area, and has no immediate business use. This paymetal loss forms the largest loss of paymetal in the entire flowsheet by which refined, or saleable, metal(s) are produced **[Cramer, 2001]**.

From the foregoing, the sampling and analysis of these production concentrators towards improved metallurgical performance, has logically been the object of technical effort. A history of these efforts exists. These sampling exercises, sometimes called *diagnostic metallurgy*, or *surveys*, have aimed at identifying and diagnosing segments, or areas, of the concentrator flowsheet, which have potential for improved metallurgical performance. Implementation of improved processing in these flowsheet areas reduces the paymetal loss to flotation tailings, and increases the recovery of the paymetal(s) to gravity or flotation concentrate(s). Also, these investigations often aim at reducing gangue dilution, to lower downstream costs, which are often higher than milling and flotation costs on a per-tonne basis. These activities, when successfully performed, enhance the business case of the production concentrator to a significant extent **[Henley, 1983]**. Such recovery or grade gains, if sustainable, generally show very high rates of return for their research and implementation cost.

There are two challenges that develop from these pursuits.

- In order for meaningful diagnosis of whatever nature to be made from the samples extracted from a concentrator survey, these samples have to be representative of the flowsheet streams from which they were taken.
- In order to better understand the actual behaviour of the minerals in the concentrator flowsheet, a quantitative mineral measurement system is needed. Using chemical analysis to determine the paymetal content of the samples does not necessarily describe the behaviour of the minerals.

In this study, the first of the above problems will be addressed. A statistical approach will be taken to improve the representativity, or trueness, of sample material taken from a production concentrator. This goes beyond routine metal accounting: rather a co-ordinated sampling campaign across the entire flowsheet is desired. This will be with the specific purpose of producing a suite of representative, or true, samples taken from an overall flotation flowsheet. This new process will be called *statistical benchmark surveying*, and will relate to the generic features of the ore being mined. It will be shown by a review of the published literature that no fundamental sampling model for this purpose has been developed.

The second of the above two problems has already been solved. The advent of modern quantitative mineral measurement technologies in the 1990's has enabled accurate measurement of minerals in samples taken from production concentrators. There are two known operable technologies. These are Quantitative Analysis of Minerals by Scanning Electron Microscopy (QEM*SEM), [the fourth generation model was commissioned in 1996, has light-element detection capability, and has been renamed Qem*SCAN], which is manufactured by the Commonwealth Scientific and Industrial Research Organisation, Australia, (CSIRO); and the Mineral Liberation Analyser (MLA), which is manufactured by the Julius Kruttschnitt Research Centre, Australia. In these two technologies, overall mineral assemblage and liberation of minerals are described at a sized fraction level. Representative sampling is critical to obtaining meaningful QEM*SEM or MLA data.

An equivalent problem used to exist at the laboratory scale flotation test level. Before one could meaningfully conduct a laboratory scale flotation test, it was first necessary to prove that the sample of ore presented to the flotation test was representative. Further, it was necessary to demonstrate that the flotation test results were reproducible. Earlier work in South Africa addressed these problems. The work showed that a rigorous approach to the ore sampling and sample preparation protocols prior to the tests reduced the fundamental variance of the ore sample to 5%, with quantitative proofs of trueness. A series of replicated flotation tests with quantitative diagnostics for laboratory scale flotation testing of platinum-bearing ores improved the platinum accountability, and thus the reproducibility of results, across the flotation test **[Lotter, 1995a]**. Apart from defining primary sampling protocols, this work measured and minimised the evaluation bias errors of PGE in specialised sample processing ahead of lead collection fire-assay **[Lotter *et.al.*, 2000]**. This work listed further opportunities to be addressed in the field of *statistical benchmark surveying* of production concentrators.

There are several types of information desired from representative sample material. These extend beyond the routine metal accounting of battery limit streams such as final concentrate(s), final flotation tailings and flotation feed. A well-formulated 'plant survey' wherein detailed sampling of internally recirculating or intermediate streams is often highly informative, and requires far more than a one-off 'snapshot' sampling approach. In the case where the type of information desired will be produced from advanced technologies such as Qem*SCAN or the MLA, which are expensive units, the challenge of sample representativity at the polished section level is critical. It is therefore very important to use reliable sampling platforms and quantitative diagnostics in this system. These diagnostics certify the representativity of the plant survey material prior to any advanced analysis being performed. High-quality information from such technologies places the investigator in a position to practice Process Mineralogy on flowsheet optimisation, or in the case of exploration, to model the behaviour of the orebody in a variety of flowsheet options. In the latter case, the modelling of probable metallurgical behaviour is necessary as part of pre-feasibility and feasibility studies, which will guide senior management in their decision to either invest capital and mine the deposit, or not to proceed. These activities generate data that provide the basis for predictive models.

The relatively young discipline of Process Mineralogy was reviewed **[Henley, 1983]**. This involves the merging of the two disciplines, *i.e.* mineral processing and mineralogy (or mineral science). In this review, a flowsheet from orebody exploration to optimal plant operation was proposed. This flowsheet had the objective of developing a predictive process mineralogy model for the orebody and concentrator in question. The early inputs from mineral science, describing the bulk mineral assemblage, possible liberation size ranges, and identifying problematic minerals, were shared with the mineral processing engineer. The latter would develop a flowsheet based partly on this information, and partly on laboratory and pilot plant testing. Following commissioning of the plant, an optimisation programme was pursued by a second and subsequent iterations of this study. These iterations especially extended beyond new samples of drill-core, into samples taken from the operating plant, which were studied by the two disciplines to identify flowsheet limitations. Various mineral science technologies were suggested as suitable means for developing the information. These included X-Ray Diffraction, microprobe analysis, scanning electron microscopy, and classical optical microscopy with point-counting. At the time of writing, the reviewer noted the beginnings of the QEM*SEM development and suggested a size-by-size structure for all of these analyses, in keeping with Trahar's models **[Trahar, 1981]**. Some discussion was offered on the relative errors of point-counting. Henley's review, however, did not address the issue of representative sampling or desired confidence levels for the overall models.

The business value of such practice in Process Mineralogy with quality control is significant. A first prototype of this model, using statistics and quality control diagnostics, has been assembled by the author and is in practice at Falconbridge Technology Centre, Sudbury, Ontario **[Lotter and Whittaker, 1998]**. A schematic of the existing three-stage model for concentrators in operation is shown in Table 1. This heuristic model has produced information, together with QEM*SEM, that has sustained a 92% per annum rate of return on investment between 1997 and 2003, from circuit improvements in production operations in Falconbridge. It is intended here to construct a sampling and associated quality control model, based on the Falconbridge prototype, with the specific purpose of producing a series of reliable, representative samples taken from a production concentrator. Although the first use of this sample material will be a closed mass and value balance, the system will be compatible with subsequent analytical technologies, giving the user a spectrum of choice. For example, size-by-size analysis of mass and values from such sample material is highly useful in describing deportment of values to characterise the flotation mechanisms **[Trahar, 1981]**. The following features of this new approach are:

1. Timing of the sampling campaigns, or surveys;
2. Frequency of sample increments within these surveys;
3. Sample processing;
4. Subsampling;
5. Chemical analysis.

Table 1 - Structure of Process Mineralogy Flowsheet

Stage 1 **Concentrator**	**Stage 2** **Mineral Processing Laboratory**	**Stage 3** **Mineral Science Laboratory**
• Background Geology • Sampling Design • Sampling • External Reference Distribution • Mass Measurement	• Sample Dewatering • Internal Reference Distribution • Quality Control Diagnostics • Sample Blending/Subsampling (Spinning Riffler) • Composite Sample Preparation • Chemical Analysis • Size Fractionation	• Species Identification Programme (SIP) • Polished Section Preparation • QEM*SEM Readings • Reconciliation of Chemical and QEM*SEM Assays • Interpretation of QEM*SEM Data • Building of Process Mineralogy Model(s)

In addition, quantitative analysis of the minerals present on a size-by-size basis further enhances understanding of the flowsheet limitations. This arrangement is set out as follows:

- Design and Implementation of Sampling
- Representative Samples with Quality Certificate
- Closed Mass and Value Balance
- Consideration of Other Options
- Size-by-Size Mass and Value Analysis
- QEM*SEM or Qem*SCAN Analysis

Clearly the desired confidence level will be a key feature of the new model's design. Certain experiences in South Africa have demonstrated that an unqualified standard of sampling and sample preparation negates any use of information from QEM*SEM, because the mineral balances do not close **[Lotter and Stickling, 1995]**. This lack of closure, contrary to earlier practice **[Lotter and Martin, 1993]**, was caused by the complex mineralogy of the Merensky

Reef, which required robust replicate surveying. An earlier heuristic sampling model had already been developed at Amplats, South Africa, to address this issue **[Lotter and Martin, 1993]**. This early model had more limited structure and diagnostics, but addressed to some extent the sampling problem.

The scope of this thesis, therefore, is to address the generic primary sampling and diagnostic designs for a survey of a production concentrator. This will ensure that the final sample material presented for further analysis of whatever kind shall be representative of typical operations at the 95% confidence level, or at some other suitable confidence level to be determined. In other words, it is critical to ensure that the primary samples extracted from the survey truly reflect the mill feed of the operations period to be characterised. A review of Gy's safety line in sampling shows that it is obvious that milled ore is a preferable sample to mill feed **[Gy, 1979a]**. Whether there exists a mineral separation process within the grinding circuit or not, the grinding circuit final product is an appropriate sampling point for tests of 'normal' ore milled.

Although some copper data will be presented in this thesis, the primary focus of this work is on nickel. Thus, unless otherwise stated, all data and calculations will be made on the basis of nickel.

1.2. Objectives and Structure of Thesis

The relevant published literature will show that an abundance of general information on the characteristics and sampling requirements of lognormal distributions exists. Published work on concentrator (survey) practice is far more limited and nowhere demonstrated the necessary statistical rigour. **Krige, 1962**, demonstrated that treatment of the daily mill feed data as a lognormal distribution produced intuitive analysis with logical action limits.

The review will show that the published sampling systems used to present true sample material to whatever laboratory scale technology used for simple size class separation, or mineralogical investigation, have not reported a fundamental method whereby the trueness of such sampling is established, measured or checked. This study must therefore address this specific issue. It is proposed to base the new statistical benchmark survey on the sampling of the Raglan operations, and using the existing Falconbridge model as a basis for further improvement. Once constructed and tested for Raglan, an extension opportunity into another operation will be sought, for example the Strathcona Mill in the Sudbury Igneous Complex, Ontario.

The specific objectives of this study will therefore be as follows:

1. To ensure, by way of thorough review of the published literature, that the apparent absence of a procedure to design, implement and check the sampling of a production concentrator is a real absence.
2. To review and critically examine the existing heuristic Falconbridge model, which is called *statistical benchmark surveying*, and which is presently used for the purpose of presenting QEM*SEM with representative sized sample material and accompanying quality certificates.
3. Based on the strengths and weaknesses found in 2 above, to construct, test and deliver two new quantitative survey models for operational concentrators:
    a. Surveying of typical concentrator operations:
        - *'Statistical Benchmark Survey'*
    b. Surveying of a concentrator during a special mining and milling campaign to describe the performance characteristics of a particular ore mix:
        - *'Campaign Survey'*

    [It is important to explain the difference between 'Statistical Benchmark Survey' and 'Campaign Survey'. The reader is referred to the text in section 1.3 that follows this list of specific objectives.]

4. To examine the historic case studies 1997 – 2002 in Falconbridge which have emanated from the practice of heuristic *statistical benchmark surveying*, in order to test the model.
5. To conduct and test the new type of survey, called a 'Campaign Survey', using a modified form of the Falconbridge model.

In the above discussion, 'quantitative model' shall mean a process of appropriate statistical rigour, or sound statistical evaluation, with quantitative proofs of representativity in samples extracted from the new form of *statistical benchmark surveying*. This approach develops a surrogate population distribution that emulates the unknown true population, and uses replicate surveys for which the Rougher Flotation Feed metal grade must fall within quantifiable limits that are determined by independent means.

The author wishes to apologise to the discipline of applied statistics in advance of this work, for the approach that will be taken. Where the approach will be statistical, it uses the discipline in a different sense to the conventional. Rather than speak of the '95% confidence level' – as is the case in hypothesis testing - terminology along the lines of the 'lognormal skewed confidence limits' will be seen.

The present heuristic model in use at Falconbridge, called 'statistical benchmark surveying' **[Lotter and Whittaker, 1998]**, will be used as an initial platform for this study. The following describes the present practice as far as final composited sample material, which was specifically designed in order to support the new Qem*SCAN unit installed at Falconbridge Technology Centre during 1998.

1.3. Differences Between Survey Types

In the above listing of specific objectives, it was declared that two new survey formats would be developed. These are:

a. Surveying of typical concentrator operations:
   - *'Statistical Benchmark Survey'*

b. Surveying of a concentrator during a special mining and milling campaign to describe the performance characteristics of a particular ore mix:
   - *'Campaign Survey'*

It is of key importance that the reader now understands the differences between these two survey formats. These formats will be developed during this thesis:

Statistical Benchmark Survey

A Statistical Benchmark Survey will be performed when an existing concentrator operation is to be audited for performance under normal, typical conditions. These conditions will be appropriately defined for individual survey campaigns. The benefits from this type of survey will be:

1. To establish a benchmark of metallurgical performance for typical ore being treated under typical conditions in the existing flowsheet.
2. To identify section(s) of the existing flowsheet that present performance opportunities.

The above information will be obtained from a series of replicate survey units, each survey unit being a two-hour sampling campaign through the flotation circuit.

This type of survey will rely on comparison of paymetal grade measurements in ore milled so as to satisfy acceptance criteria. These criteria are paymetal grade measurements that will take the form of reference distributions. There will be two reference distributions. These will be the External Reference Distribution, and the Internal Reference Distribution.

The External Reference Distribution will measure the grade of ore typically milled at that concentrator from a large, robust sample of operations data that are typically gathered in the course of routine metal accounting for that concentrator. The data for the External Reference Distribution will be taken from routine samples of Rougher Flotation Feed, generally across a period of at least one month. The term 'Rougher Flotation Feed' means the final milled product from the grinding circuit which is about to be processed by the flotation plant.

The Internal Reference Distribution will measure the grade of ore milled during the six two-hour survey units. These data will be obtained from the Rougher Flotation Feed samples that are taken during each of the two-hour survey units. These two-hour survey units will be spaced across a period of two weeks of operations.

From the above outline, it should be clear that the External Reference Distribution in the Statistical Benchmark Survey is independently derived from the Internal Reference Distribution; and that the External Reference Distribution is a large, robust data base for which parameter estimation is relatively simple. It should also be clear that the Statistical Benchmark Survey will characterise typical metallurgical performance for that concentrator.

Campaign Survey

The Campaign Survey will be designed to meet a different objective. This will be to mine and mill a specific ore mixture, often a problematic ore mixture, or sometimes a simulated future ore mixture, to understand the reasons for problematic performance or to predict future metallurgical behaviour. In such a case, the tonnage of ore mixture is limited and the concentrator operation in treating that limited tonnage will be within one week. This format calls upon very high levels of planning and co-operation between disciplines on the mine.

In this format, the two reference distributions, *viz.* the External Reference Distribution and the Internal Reference Distribution, remain in place - but the External Reference Distribution takes a different form. In this *Campaign Survey*, the External Reference Distribution uses drill-core data and waste dilution (the latter obtained by actual (theodolite) surveying of the mined areas after the fact). Accordingly, this External Reference Distribution works in the full lognormality of space and must be handled as such. The Internal Reference Distribution is gathered in a manner similar to its counterpart in the *Statistical Benchmark Survey*.

Common Purpose

The common purpose of both of these survey formats will be to provide a suite of representative flotation circuit samples that will have been taken during times of representative or known concentrator feed conditions. The determining basis of acceptability will be whether those sample suites correspond to agreement between the means of the External and Internal Reference Distributions of circuit feed.

CHAPTER 2 - EXISTING MODEL AND APPLICATION TO THE FALCONBRIDGE OPERATIONS

2.1. History of Past Concentrator Survey Practice

Very little published material which describes the knowledge and best practice of surveying concentrators was found. This section will summarise what has been published on the subject, to conclude that no fundamental concentrator survey model has been published. Rather, the past practice has been heuristic. The reader is referred in section 1.2. to the earlier discussion on '*appropriate statistical rigour*' and '*quantitative statistical proofs*'.

Based on the endeavours reported, it is clear that some metallurgical engineers have long recognised the valuable potential of a well-formulated and executed concentrator survey. However, because these endeavours failed to connect with fundamental sampling models, the reliability of the interpretation was questionable. This led to a hit-and-miss track record in flowsheet optimisation programmes, which used these surveys for information.

Commonly a flowsheet is functionally described, showing the logical flow of streams in sequence. A list of streams to be sampled is prepared. A team of staff samples the plant during its normal operation according to the list of specified samples. The sampled streams are often also checked for solids and water flow as necessary inputs to mass balancing. After dewatering, the samples are used for various purposes such as overall chemical analysis, or size-by-size analysis.

The duration of such survey sampling varies. Commonly a survey samples the plant operation across a few hours. One of the earlier references to this approach described an audit of the Broken Hill South concentrator, Australia **[Cameron *et. al.,* 1971]**. This project used a sampling duration of four hours with sample increments taken every 20 minutes. The samples were sized, and a size-by-size performance model developed for lead and zinc. The authors stated that unambiguous conclusions had been derived from the approach. A subsequent reference to the subject of concentrator surveying described the structure and procedure in logical detail **[Restarick, 1976]**. In this practical work, the importance of proper planning, preparation, and execution of steady state campaign for metallurgical plants was emphasised.

The general rules emanating from this work were:

1. Construction of the actual flowsheet by physical inspection during the planning phase of the survey,
2. A survey duration of at least four times the residence time of the process,
3. No shift change at the time of the survey,
4. A total number of 'cuts' for any stream between 20 and 50 to reduce the effect of any faulty cut,
5. Restricting, stopping or diverting all minor or continuous flows such as floor washings ('spillage'),
6. Selection of personnel for the sampling with adequate and relevant training and expertise,
7. Diligent attention paid to the integrity and labelling of each sample bucket, including the recording of nett mass before sampling.

The specification of sample cutters was described in detail. In particular, reference to Taggart's rule of the cutter gap being at least four times the size of the largest particle in the stream was noted. Further, the slot length had to be at least five centimetres longer than the maximum stream width relative to the direction of cut **[Taggart, 1944]**. The author correctly concludes that care and attention to detail in the preparation and execution of a sampling campaign is essential to the attainment of accurate and significant data that would emanate from such sampling.

An example of diagnostic metallurgy based on a plant survey system for the Mount Lyell concentrator, Australia, was reported in 1977. This approach used a five-hour survey consisting of sample cuts every 30 minutes **[Hartley *et. al.*, 1977]**. This amounted to 10 sample cuts per survey, which could be regarded as insufficient. The mass balance was based on direct measurements of the fresh feed rate of dry ore to all primary grinding circuits. The flotation analysis used a cyclosized size-by-size format. The key conclusions were that recoverable chalcopyrite losses in tailings occurred as locks or middling particles, and that the chief pyrite loss occurred as fines in the desliming cyclone overflows. These conclusions were first established by detailed size-by-size mineral recovery modelling, followed by more specific investigation of suspected streams.

In the analysis of a tin concentrator using a method comparable to that of Restarick, whose inputs to the plant survey design were acknowledged, it was shown that useful and robust mathematical models of the tin behaviour across the shaking table could be developed, correctly diagnosing the basic cause of poor tin recovery **[Weller and Sterns, 1984]**. In this work, samples were accumulated over a four-hour period with cuts taken every twenty minutes. This would amount to 12 sample cuts per survey, which could be regarded as insufficient. Sample management included the capture of a wet mass, then a dry mass after dewatering. Again a size-by-size approach was used to model the behaviour of the mineral of interest, in this case cassiterite. Various methods of mass and volume measurement were used. These ranged from timed volume measurements to an ultrasonic flowmeter. Other flow measurements, although made, were disregarded because of large or uneven flows. A mass and value balance smoothing exercise was performed on the raw data. The analysis was further supported by laboratory scale testing and mineralogical study on polished sections. Flowsheet changes were recommended from this work.

The role of quality control, and fundamental statistical analysis in achieving the ‘clear and unambiguous conclusions’ from these foregoing four surveys was not mentioned or used. Neither did these publications fundamentally address the issue of calculating the minimum sample mass required for any given stream from a suitably-designed sampling experiment. It appears that although the approaches were basically correct, they were heuristic, and probably exceeded Gy's minimum sample mass model requirements. A summary of these approaches is shown in Table 2.

Table 2 - Summary of Plant Survey Approaches

Aspect	**Cameron *et. al.*, 1971**	**Restarick, 1976**	**Hartley *et.al.*, 1977**	**Weller *et. al.* 1984**
Survey Duration, h	4	4	5	4
Sampling Increment, min	20	20	30	20
Sizing and Assays by Size	Yes	Yes	Yes	Yes
Mass Balancing	Yes	Yes	Yes	Yes
External Checking	No	No	No	No
Quality Control	No	No	No	No
Confidence Level	No	No	No	No

In this study, the new *statistical benchmark surveying* model will provide quantitative models to adjudicate measurements of feed grade to the concentrator. These were found to be absent in the previously published (heuristic) surveying models. Statistical models such as these are probabilistic, not deterministic. Therefore, the new model will estimate the likelihood that *'it has failed to identify non-representativeness'*, rather than *'it has proven representativeness'.*

It is concluded from this review of past concentrator survey practice that:

1. Four genuine attempts were made to establish a reliable surveying method. These were all in the period 1971-1984, and produced heuristic rule-based structures without quality control diagnostics.

2. No consistent approach was taken between these four methods.
3. Nowhere in these methods was any mention made of checking the paymetal grade of ore milled, either internally or externally.
4. In none of this work was there any mention of replicate survey units; all of the surveys were performed in singleton.

2.2. Statistical Benchmark Surveying – History at Falconbridge

The business value of the practice of concentrator surveying in Process Mineralogy with quality control is significant. Data generated by sampling surveys form the basis for sound recommendations for concentrator flowsheet improvement. These improvements often require significant capital expenditures. The investment risk must be minimised. Conversely, the full potential of flowsheet improvements must be identified and tapped. Hence it is critical to generate very high-quality data with appropriate confidence intervals. To this effect, a first prototype of this model using statistics and quality control diagnostics has been assembled by the author and is in practice at Falconbridge Technology Centre, Sudbury, Ontario **[Lotter and Whittaker, 1998]**. A schematic of the existing three-stage model for concentrators in operation is shown in Table 1. The arrangement of the three major areas of sampling is shown in Figure 1. Information to hand at the time of writing strongly suggests that the Central Limit Theorem and associated use of reference distributions **[Box *et. al.*, 1978, Chaps. 2-3]** may form the basis of the plant survey model structure. This is because singleton surveys will not gain the advantages of the Central Limit Theorem. Replicate surveys, when taken through an averaging process, bring about normally-distributed errors that are lower in value than their singleton counterparts. Also, when these replicate surveys are appropriately spaced so as to break the effects of auto-correlation in time, a more effective outcome results. The reference distribution approach addresses the sampling of composite distributions, and breaks the potential auto-correlation in time problem. This aspect of auto-corelation in time will be explained in section 3.7.

At this stage it is sufficient to state that the breaking of auto-correlation in time is key to randomising the sampling activities.

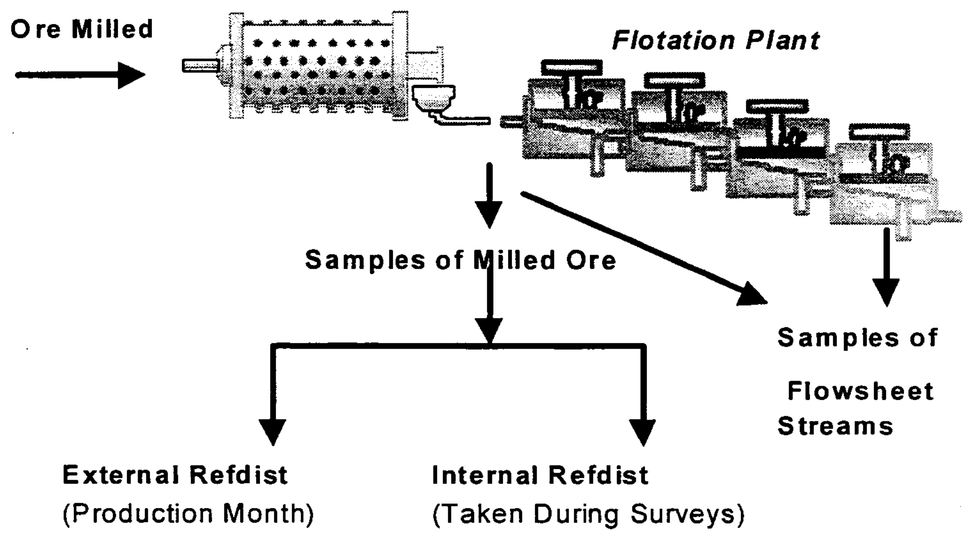


Figure 1 - Major Sampling Areas - Falconbridge Model

In the construction of this Falconbridge model, Lotter formulated the sampling and quality control components, whilst Whittaker formulated the mineral science components, such as specification of polished section preparation, species identification programmes, etc. The battery limits, or end products, therefore, of the sampling and quality control components were from ore milled to a suite of final composite samples ready for further analysis of any kind.

Flowsheet Planning

In planning a particular survey of a nominated mill or concentrator, staff construct an 'as-is' flowsheet of the production operation by physically walking through the process. In this process, the actual connections between unit operations are empirically logged. The flowsheet is then 'frozen' until the survey has been completed. Discussions are held, together with a review of actual production records, to establish milling rate benchmarks. A proposed list of streams to be sampled is prepared. Each proposed sampling site is physically inspected and engineering modifications specified for the safe and unbiased sampling access to that stream. The engineering modifications are attended to prior to any surveying activities.

A schedule of surveying is proposed to the production staff. It is understood that steady-state operations with minimal process changes during surveying are a requirement, and that no spillage is recycled during the surveys. Especially, there must be no flowsheet changes during the surveys.

In this prototype, six replicate surveys are taken. Their respective feed grades are used to assess their acceptability into composite samples. This decision is made in two stages. First, the six replicate feed grades are tested as a small data set, which is called the Internal Reference Distribution. Second, those samples not rejected in the first stage are tested again using a larger data set of feed grades representing routine operations for the month surrounding the surveys. This is called the External Reference Distribution. The samples that fail to conform to these trials are rejected. The accepted set of samples are prepared into a composite that is a robust representation of the process, as the Central Limit Theorem would predict **[Box *et. al.* 1978, Chaps. 2-3]**.

External Reference Distribution

The month of milling production data surrounding the actual survey is accessed. The grade of ore milled is collected as a variable on a shift frequency level. An external reference distribution is constructed from these data. The sample mean and standard deviation are calculated. The 95% acceptance limits of the distribution are calculated. Let the following variables be assigned:

n_{ext} = the number of observations used in the external reference distribution

s_{ext} = the sample standard deviation of the external reference distribution

$\bar{g}_{ext}$ = the sample mean grade of the external reference distribution

The Two-Hour Survey Unit

Prior to the two-hourly survey unit being performed, an engineer works on shift for a period of 8 hours with the plant operators to ensure steady milling and flotation operations, and to ensure that reagent dosage respects the standard arrangement. Further, all sample buckets are labelled and tared. The survey unit consists of four 30-minute sample increments (also called cuts) taken at each listed stream. The increments are added progressively to a labelled sample bucket. The same sequence of sampling is used in each 30-minute increment period. Should a mill or other major piece of process equipment trip out during this sampling, the survey unit is automatically disqualified. This two-hour survey unit is based on earlier work on the surveying of grinding circuits and reflects the need for the operating process to undergo several retention time units during the survey **[McIvor, 2004]**.

After the two-hourly survey unit is completed, the samples are delivered to the mineral processing laboratory where they are dewatered, then weighed again. This provides vital information on pulp density. [*Four cuts - each taken at 30-minute intervals in the two-hour survey unit - present an opportunity for improvement. This will be discussed under 'Probable Effects' on page 38*].

Number of Replicate Two-Hour Survey Units

Generally six replicate two-hour survey units are conducted across a 7- to 14-day period of operations. This provides for the separation of each two-hour survey unit by at least one day in order for basic sample management to be addressed. This also allows for a separation of each survey unit into a random-stratified arrangement, which is likely to break or minimise the effects of auto-correlation in time. (On occasion, new staff who have insufficient experience in this matter and do not readily appreciate the subject of random stratification, sometimes take an opportunistic extra survey on a back-to-back basis from the previous survey unit when operating conditions are steady, in the belief that they are being cost-efficient. [*This practice will later be shown to be of no value. Instead, it will produce extra survey unit(s) with Rougher Flotation Feed grades that are unlikely to add as much information as the other surveys, thus negating the perceived benefits of 'finishing up the surveys in as short a time as possible'.*] This in part amounts to the importance of both training and experience in the subject) The sample material from each two-hourly survey unit is separately stored.

A representation of 'random stratification' follows. In a two-week survey period, let six two-hour survey units be taken. No two surveys should be back-to-back in time. All survey units should be separated by a variable amount of non-sampling time. Some survey units will be taken on day shift. Some will be taken on afternoon shift. Some will be taken on night shift. In this way a randomised pattern of sampling in time is created, breaking auto-correlation in time **[Napier-Munn, 1995]**. The training of site operations staff in this technique is strategic to the overall resource and cost implications of the survey project. Where a core team of specialists will be on site for the survey project, most of the mill stabilisation and sampling should be done by the operations staff, who readily train for the necessary skills.

Internal Reference Distribution

The sample labelled 'ore milled' for each two-hourly survey unit is dewatered. After each sample is dried, a specially-designed blending procedure called 'odds-and-evens blending', using a spinning riffler **[Lotter, 1995a]**, is used to internally blend each ore sample. Thereafter, a subsample of each 'ore milled' sample from their respective two-hourly survey unit is taken, again using the spinning riffler in a procedure called 'subsampling' **[Lotter, 1995a]**, and is chemically analysed. This analysis is performed in replicate format, generally using 6 determinations. The arithmetic mean grade is calculated.

The six mean grades are pooled to form an internal reference distribution, with a sample mean and sample standard deviation.

n_{int}	=	the number of observations (two-hourly survey units) used in the internal reference distribution
s_{int}	=	the sample standard deviation of the internal reference distribution
$\overline{g}_{int}$	=	the sample mean grade of the internal reference distribution

Internal Inspection of the Internal Reference Distribution

Individual mean grades of each two-hourly survey unit are compared with the 95% acceptance limits of the internal reference distribution. The limits are constructed from the sample mean plus or minus two sample standard deviations. Any observations falling outside these limits are deleted, and the two-hourly survey unit concerned, including all of its sample material, is disqualified from further use. The relevant sample material is discarded. If such disqualification occurs, a shortened set of parameters is recalculated for the internal reference distribution. [*In the eight years of documented practice 1997-2005, this procedure has never identified an outlying observation. This is because the present form of the Internal Reference Distribution is not statistically robust, i.e. is not a powerful filter. It will be shown later in this study that the Sichel t-Estimator is far more appropriate for this purpose, but is fundamentally different in terms of associated confidence intervals.*]

External Inspection of the Internal Reference Distribution

The remaining two-hourly survey ore grades are overlaid on the external reference distribution. Any observations falling outside of the external reference distribution limits at the 95% acceptance level are discarded in similar manner to the inspection performed for the internal reference distribution. If such disqualification occurs, a shortened set of parameters for the internal reference distribution is recalculated.

Finally, the average grades of the external and internal reference distribution are arithmetically compared for likeness. At this stage, if the two means are 'sufficiently close', approval is given for a composite to be prepared from the successful two-hourly survey units, provided that a minimum of three units pass these tests. [*An opportunity for improvement is recognised here. Appropriate significance testing should be developed for the comparison of these two means.*]

A diagnostic report is prepared, in which the aforementioned measurements, disqualifications and deletions (as the case may be) have been performed. All raw data are attached to this report.

Composite Preparation

The final composite sample for each sampled stream is then prepared on an equal-mass increment basis, again using the spinning riffler odds-and-evens preparatory blending and the subsampling procedures mentioned earlier **[Lotter, 1995a]**. The blended composites are then immediately subsampled into two subsamples, one of which is stored in a secure place for backup purposes. The front half of this composite is then presented to various procedures, the first of which is overall chemical analysis to the >99% composition level.

Heuristic Grounds

The foregoing brief description of the Falconbridge model indicates that emphasis on inspection is on the grade of ore milled only. Earlier work performed in this model **[Lotter and Tuzun, 1998]** empirically found that equivalent checking of the concentrate and tailings streams using the same logic of external and internal reference distributions made no difference to the 'accept' or 'reject' decisions on the two-hourly survey units. Further reading indicated the likelihood that, since all the minerals present in final concentrate and tailings are derived from the full mineral assemblage of the mill feed, or ore milled **[Clark, 1993b]**, the only variable which would cause changes in the makeup of these streams during the survey would be the 'process treatment', 'flowsheet effects', or 'nurture'. Since the arrangement of the Falconbridge model is within a time frame of a month of operations, and since rules of defined, steady milling rate and of frozen flowsheet are in place, the 'process treatment' could be regarded as a constant or as an insignificant variable in that time. Therefore, the main source of variation would be expected to be within the grade of ore milled.

Probable Effects

The probable effects of this sampling model draw on averaging processes such as are known in the Central Limit Theorem. The external reference distribution of ore milled produces a sample mean. Since the sample mean is derived from a robust sampling size, that estimate of the mean grade is likely also to be robust. The internal reference distribution, in producing a composite sample of ore milled, will be using 4 primary sample increments every two-hour survey unit. The process of adding the slurry sample increments to one sample bucket physically averages this. When between 3 and 6 two-hourly survey samples of ore are composited (after quality adjudication), a total of between (4 x 3) = 12 and (4 x 6) = 24 measurements are being used to formulate the sample mean of the internal reference distribution. This is also in keeping with the above Central Limit Theorem effect.

Assuming for simplicity that there is no autocorrelation in the errors, an exercise follows on the basis of the sampling standard deviation and standard error. To illustrate, taking the sampling standard deviation at an arbitrary 100%, the number of cuts in the two-hour survey unit affect the standard error S_ε according to $S_\varepsilon = 100/(n)^{0.5}$. At n=4 cuts, the standard error is 50%. With increasing number of cuts, the standard error reduces in proportion to $n^{-0.5}$. It is now a balance between the number of cuts that a technician can handle practically in a two-hour period, opposite the theoretically idealised minimisation of standard error by very large n.

Often each technician has to handle several different sampling points. It is possible to perform cuts at a 10-minute interval. This would provide 12 cuts per survey, and would reduce the standard error to $100/(12)^{0.5}$ = 29%, nearly halving the original standard error of 50% for the 4-cut option.

It is proposed, therefore, in the new survey model, that 10-minute cuts be used, producing a total of 12 cuts per 2-hour survey unit. This would be consistent with the reported past practice reviewed in Table 2, page 30.

This model has been under a process of continuous improvement since 1997. At the time of writing, the following problems have been identified which need to be addressed:

1. The Raglan geological system presents some complex distributions of the nickel paymetal. This is addressed in part by End-Member studies, which identify discrete geological entities and conduct sampling accordingly. The milling and flotation characteristics of each End-Member are specific, or individual.

2. At the level of overall ore milled, the various End-Members combine to present a complex distribution of nickel grades. This distribution has to be adequately represented.
3. Auto-correlation exists in the dimension of time, represented as Rougher Float Feed. The sampling of this stream has to take this factor into account.
4. The number of cuts per 2-hour survey unit should be increased to 12 from 4.

2.3. End-Member Characterisation at Raglan

Generic Features – The End Member Study

As a mill treats an orebody in its entirety across a period of several years, variation in day-to-day performance may be described as the sum total variation of the individual variations, which are independent of each other. The major sources of these variations are: -

- Variation within the orebody,
- Variation of the process(es) treating that orebody.

Both of these variations could have correlations that span many days, weeks or months of plant operation. In the practice of Process Mineralogy, the main aim is to identify the generic features of the orebody (its *nature*), and to establish whatever deficiencies exist in the mineral processing flowsheet treating that orebody (the *nurture*). Given that it is impossible to change the nature of an orebody, the only leverage available by which process performance may be optimised is by improving the *nurture*, or treatment, in a series of flowsheet changes that align with the modelled nature of that orebody. In other words, the flowsheet must match the process mineralogy requirements of that orebody if optimum mineral processing performance is sought **[Kinloch, 1991; Wilkie,**

1999]. This approach was also reported in the earlier heuristic survey work **[Cameron *et. al.,* 1971]**.

At a later stage of this work, it will be shown that it is not sufficient to simply perform representative sampling on a production mill/concentrator in order to make sense of the sample material so extracted. Three more key requirements are indicated. These are closely linked:

1. It is very essentially also a matter of taking individual ore type samples from underground or from drill core and studying these in order to develop a fundamental picture of the generic makeup of the orebody. This is called an *end-member study* **[Whittaker, 2002; Lotter *et. al.,* 2003]**. By describing an orebody at the end-member level, a component of the mill feed variation in the time dimension becomes described and quantified. Predictive properties develop as a result.
2. Beyond the primary sampling, it is also necessary to implement judicious use of sample blending and subsampling. The choice of sample preparation equipment and method defines the error level **[Allen, 1990]**. The spinning riffler is the equipment of choice because it produces the lowest error level at less than 0.5% for base metal ores. The sampling mass reduction protocol must therefore respect this technology since no size reduction is performed in the treatment of the survey samples.

The end-member model addresses the orebody at a generic level. The orebody generally has different geological components in terms of, for example, sulphide mineral texture. These textures correlate with paymetal grade. In the case of Raglan, the grain size of pentlandite also correlates with texture. Coarser grain size is associated with the massive sulphide group, intermediate grain size with the net-textured group, and fine grain size with the disseminated group. This arrangement fundamentally sets the liberation patterns, which develop from a common grinding circuit, thus defining the basic recovery profile across the flotation circuit. At least a part of the day-to-day variation in paymetal grade at the mill feed level, and paymetal recovery at the final concentrate level, are thus accounted for by variation in the relative tonnages of the various textures mined and delivered to the mill. The variation in these tonnages mined depends, in their turn, on the mining plan.

A further predictive feature emerges from the end-member model approach. This, in the case of ultramafic orebodies, consists of variation in either the primary silicates between feldspars and pyroxenes, or between the primary to secondary (altered) ferro-magnesium silicates from pyroxene to talc and chlorite. Examples of these are seen in the Raglan mine, an ultramafic base and precious metal deposit located in the north of the Ungava peninsula, northern Québec; and in the various orebodies found and mined in the Sudbury Igneous Complex, (also called the Sudbury Basin), Ontario. Earlier work in the South African Bushveld Complex on the Merensky ore type detailed the individual mineral processing characteristics of the three End Members Footwall, Reef and Hanging Wall **[Lotter, 1985]**. This feature has a profound effect on flotation characteristics, in that the ferro-magnesium silicates interfere with the pulp electrochemistry, adsorb on to sulphides, and adsorb flotation reagents. This slows down the flotation kinetics of the paymetal mineral(s) (pentlandite in this case) **[Lotter *et. al.*, 2003, Pietrobon, 1996; Pietrobon *et. al.*, 1997]**. In the case of the Sudbury Basin end-members, it was shown that a common grind produced similar (satisfactory) liberation patterns across the three end-members, yet the end-member with the higher proportion of ferro-magnesium silicates demonstrated slower flotation kinetics **[Lotter *et. al.*, 2003]**. Accordingly, variation in the degrees of alteration, or between feldspar and ferromagnesium gangue present, could cause significant variation in flotation performance. Knowledge of these features prior to, and during, a survey of a concentrator, therefore accounts for components of the metal grade variation in mill feed, and for components of 'process variation' attributable to ore mix.

This predictive property is one of the key features of Process Mineralogy. The typical deliverables from an end-member study are thus:

1. Definition and sampling of end-members
2. Description of end-members in terms of:
    a. Paymetal(s) grade
    b. Bulk and minor mineral assemblage
    c. Host mineral grain sizes
    d. Host gangue
    e. Milling and Flotation behaviours (see (3))
3. High-Confidence Flotation Tests of representative samples of each end-member, simulating the standard production operations conditions
4. High-Confidence Flotation Tests on probable Run-of-Mine ore mixtures to describe an overall result, including whatever interaction the end-members may have.

As indicated earlier, the development of appropriate sampling and flotation test protocols for laboratory scale flotation testing was shown to have a key effect on minimising sampling and laboratory scale flotation testing errors **[Lotter, 1995a]**. This work took an unconventional approach in studying, measuring, and testing appropriate *acceptance limits* for replicate observations of sampled ore grade, rougher flotation test replicate concentrate masses, overall mass balances, and reconciled feed grade calculations. These acceptance limits were based on the characterisation of the spatial and temporal distributions of the PGE paymetals. After minimising the total errors in the sampling, flotation testing and lead-collection fire-assaying errors, the PGE metal balance could be closed to within ± 3.3% total error. Prior to this work, typical flotation testing practice did not always have the prerequisite of appropriate ore sampling protocols, nor was there any structured, quantitative quality control system in place which could be used to assure the investigator that the reproducibility of the work was suitable, and that the metal balance across the flotation test(s) was at an appropriate level of closure. This work concluded that a 95% level of data distribution confidence limits was a suitable standard, and at this level produced an economic optimum of quality information for the research resource expended in sampling and flotation testing. This system was accepted as industrial practice in Rustenburg Platinum Mines, South Africa, for the support of grinding and flotation optimisation project work to production concentrators in the 1990's; and has been accepted as the standard in Falconbridge Ltd., in Canada since 1997. The reader is invited to read the appropriate reference in order to understand how this system was developed, tested and justified **[Lotter, 1995a]**.

With the above list of deliverables to hand, the investigator is placed in a strong position to plan a *statistical benchmark survey*. Such preparatory work is a definite recommendation prior to any surveys. At Falconbridge, detailed end-member studies have been performed for Raglan, Strathcona, and Kidd Creek **[Lotter *et. al.*, 2003; Fragomeni and Boyd, 2003]**.

At AMPLATS in South Africa, for example, detailed end-member studies of the Bushveld Merensky ore type were performed **[Lotter, 1985; Lotter, 1995a]**. These studies described and quantified individual assemblages, grain sizes and PGE mineral host assemblies, together with individual mineral processing needs and flotation circuit behaviours for Footwall, Reef and Hanging Wall. These end-member studies formed a foundation of information, which was subsequently used for major capital projects in AMPLATS for the attainment of sustainable improved metallurgical performance.

Raglan End-Members

The Raglan operations of Falconbridge Ltd are located in the Ungava peninsula of Nouveau Québec north of the 55^{th} parallel. The area is also known as Nunavik. The mine was commissioned in December 1998, and treats approximately 1 million tonnes of mined ore per annum, to produce a bulk sulphide concentrate which is despatched to the Sudbury operations of Falconbridge for smelting and further processing **[Lotter *et. al.*, 2002]**. It is intended to base most of the experimental work for this thesis on the Raglan operations. The ore mineralogy and textures of the critical zone of the Raglan system was proposed to be a textural classification described by sulphide abundance and grain size **[Lesher *et. al.*, 1999]**. The current generic model involves derivation of the paymetals from the magma and sulphur from underlying sulphidic sediments, upgrading of sulphides during transport as immiscible layers and/or droplets in turbulently flowing lava channels or channelised sheet flows, and deposition in a complex network of multiple, overlapping lava channels.

The sulphide mineral assemblage of most Raglan ores contains monoclinic pyrrhotite, pentlandite, chalcopyrite, and pyrite. Trace amounts of sphalerite, arsenopyrite, and platinum-group minerals such as sperrylite, merenskyite and sudburyite are noted. The relative abundances of pyrrhotite, pentlandite and chalcopyrite vary from sample to sample, zone to zone, and deposit to deposit. This reflects original variations in iron, nickel and copper contents and modifications during metamorphism and deformation. The gangue mineralogy is pervasively serpentinised and also contains variable amounts of magnetite. Lesher *et. al.* proposed six sulphide textures as an end-member set, *viz:*

1. Cloudy disseminated: 1-50% sulphide, microscopic (<0.1 mm) interstitial to serpentinised olivine grains, typically pyrrhotite-rich.
2. Interstitial disseminated: 1-30% sulphide, fine-grained (1-3 mm) interstitial to serpentinised olivine grains.
3. Blebby disseminated: 1-5% sulphide, fine to coarse (0.2-0.3 cm) pentlandite partially enclosed by pyrrhotite.
4. Patchy net-textured: 20-40% sulphide, patches (0.5-2.0 cm), enclosing olivine grains, commonly associated with interstitial disseminated sulphides.

5. Patchy reverse-net-textured: 20-40% sulphide, patches (0.5-2.0 cm), enclosing and partially replacing serpentinised olivine grains, commonly associated with patchy net-textured sulphides.
6. Net-textured: 40-70% sulphide; enclosing serpentinised olivine grains.

This, in terms of metallurgical behaviour, simplifies into three end-members because of the grain size grouping. The generic stratigraphy of the economic zone is shown in Figure 2.

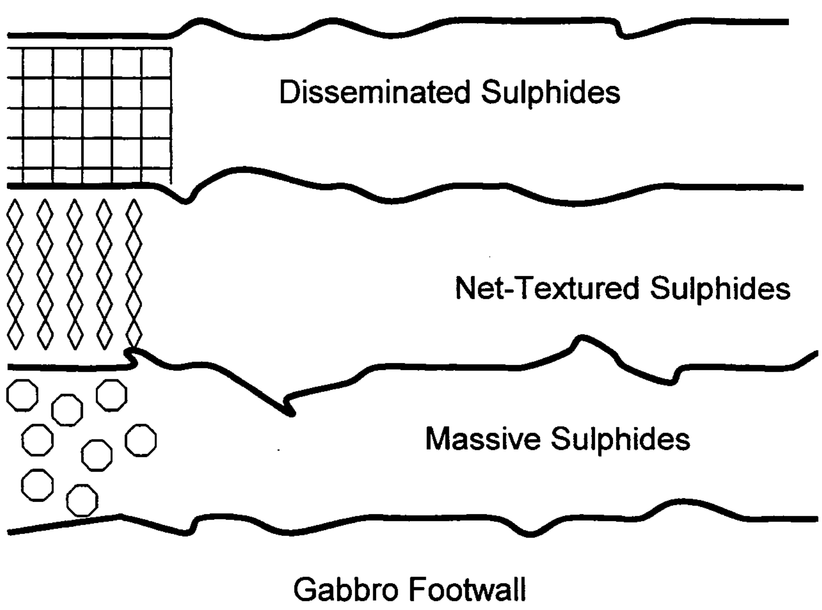
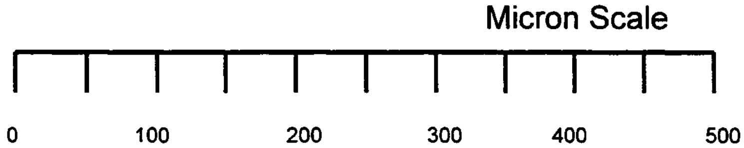


Figure 2 - Generic Stratigraphy of the Raglan Orebody

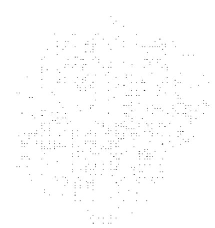
Three generic end-members are overlain on a gabbro footwall. The higher-grade massive sulphides rest on this footwall. Above the massive sulphides, the net-textured sulphides form the majority (approximately 70-75% by mass) of the ore mined, and are a lower grade than the massives. Minor (10-15% by mass) disseminated sulphides complete the structure and rest on the net-textured end-member. The generic trend in metal grade is thus highest at the base, in the massive sulphides, and lowest at the upper end, in the disseminated sulphides

[Whittaker and Kormos, 2001]. Examples of typical particles in Rougher Flotation Feed, described by QEM*SEM from the 1998 statistical benchmark survey, follow:

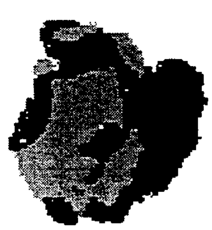


Red: Pentlandite
Green : Ferro-Magnesium Silicates
Yellow : Chalcopyrite
Purple : Pyrrhotite

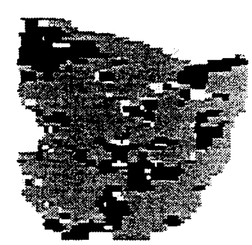
Massive Sulphides:



Net-Textured Sulphides:



Disseminated Sulphides:



The reader is referred to the foregoing discussion on the Raglan End-Members. (Figure 2). This discussion suggests that there are three groups of nickel grade in the mineral hosting by End-Member. It is expected, therefore, that a trimodal distribution of nickel grades will be a key feature of the distribution of values in ore. The concentrator operations which grind and classify this ore to a size of typically d_{80} = 65 microns, allow the imperfect separation of sulphide minerals into a concentrate of approximately 15% mass relative to ore milled. The bulk of the waste minerals such as silicates or oxides is discarded as tailings.

Primary Composition File

The identification and measurement of all minerals present in Raglan ore and concentrator products was performed using Energy-Dispersive X-ray technology. Characteristic spectra of key and minor minerals have been captured using conventional Scanning Electron Microscopy (SEM). The list of minerals for the Raglan sample measurement on QEM*SEM, called the primary composition file, shows that the '*chemically assayed nickel*' is hosted in a variety of minerals in three orders of magnitude of nickel concentration **[Kormos and Whittaker, 1998 and 2001]**. These are summarised in Table 3.

Table 3 - Raglan Primary Composition File: Qem*SCAN
Nickel Host Minerals Grouped by Order of Magnitude
[Whittaker and Kormos, 2001]

Order of Magnitude	**10-100% Ni**		**1-10% Ni**		**0.1-1.0% Ni**	
Nickel Grade %	Altered Pentlandite (Rare)	18.4	Nickel Mg Fe Silicate (Rare)	1.0	Serpentine (Common)	0.1
	Pentlandite (Common)	34.0	Serpentine-Pentlandite Texture (Common)	5.1	Magnetite (Common)	0.2
	Nickel Arsenides (Rare)	35.4	Carbonate-Nickel Iron Sulphide Texture (Trace)	9.0	Nickel in Pyrrhotite (Common)	0.5
	Violarite (Trace)	39.0				
	Non-Ferrous Nickel Sulphides (Rare)	64.7				

The 'pseudo-minerals' listed, as for example 'serpentine-pentlandite texture', are actually fine-grained textural intergrowths of two or more minerals. These account for a significant feature in the metallurgical response of Raglan ore in the concentrator. This feature is focussed on liberation requirements prior to flotation, and is the main reason that the regrinding of the cleaner tailings in the flotation circuit achieved the sustainable recovery gains reported [**Lotter *et. al.,* 2002**].

The relative abundance in overall ore milled of the first group of minerals reported in Table 3 varies according to the relative proportions of the three End-Members in that overall mixture of ore milled. This variation drives the overall nickel assay when measured as Rougher Flotation Feed. Later in this text (Table 8) the nickel sulphide grain sizes of the three End-Members are reported, and are different according to End-Member. This drives the degree of nickel sulphide liberation in Rougher Flotation Feed.

The range of nickel in the above Primary Composition File for Raglan ore is thus from 0.10 % (Serpentine) to 64.7% (Non-Ferrous Nickel Sulphides). Within each mineral type, the range is much narrower. These two observations lead to the following conclusions. Firstly, effective sampling of the plant feed, which exhibits a balanced blend of the full range of minerals, will be the most challenging, and will require multiple assaying. Secondly, the final concentrate and tailing products, and to a lesser extent intermediate concentrate and tailing products, will preferentially contain a narrower range of dominant minerals, and as a result have a lower variance and are easier to sample.

This type of problem was studied in the form of random stratified sampling and was compared to random sampling **[Cochran, 1946]**. The type of population discussed was that of a compound distribution. In such a compound distribution, there are several subdistributions, each of which has its own mean and variance. The overall variance of the compound distribution is larger than the largest variance in any of the subdistributions, because such compound distributions span a wider range of data. In other words, heterogeneous ore samples require a larger sample size than does a simpler unimodal, homogeneous distribution. In mineral processing terms, this means that the 'stratified' – or 'homogenised' sampling of the ore as the concentrator products final concentrate and tailings, when reconciled to a theoretical head grade of ore milled, is a more accurate estimate of that grade when sampled as an ore milled (which is a heterogeneous sample). As a result of the first survey of the Raglan Concentrator after commissioning, the following mineral balance was found in the Rougher Flotation Feed composite sample.

Table 4 - Minerals in Raglan Rougher Flotation Feed
June 1998 Survey – Measurement by QEM*SEM (Sorted by Nickel Grade)

Mineral and Texture Type	Mineral/Texture	Quantity in Raglan Rougher Flotation Feed %
High-Grade Sulphides 9.21%	Pentlandite	8.72
	Violarite	0.47
	Nickel Sulphides/Arsenides	0
Low-Grade Sulphides and Sulphide-Silicate Textures 14.3%	Serpentine-Chalcopyrite	0.97
	Pyrite	0.60
	Pyrrhotite	7.77
	Chalcopyrite	1.82
	Other Sulphides	0.04
	Serpentine-Pentlandite	1.21
	Serpentine-Pyrrhotite	1.87
Silicates/Oxides 76.5%	Serpentine	46.54
	Pyroxenes	0.05
	Amphibole	14.39
	Talc/Chlorite	3.22
	Other Silicates	6.16
	Carbonates	1.45
	Magnetite/Ilmenite/Chert	3.33
	Other	1.39

The above data represent an overall measurement of all minerals present in Raglan Rougher Flotation Feed during the June 1998 survey.

High-Confidence Flotation Testwork

As described earlier in this manuscript, the End-Member Characterisation Study is useful as a preliminary exercise prior to any operations survey. At Falconbridge, this exercise was conducted and reported **[Fragomeni and Boyd, 2003]**. The method of sampling, sample preparation and flotation testing followed the high-confidence approach by **Lotter, 1995**. The aims and objectives of this work were, for each End-Member of the Raglan set, starting with a standard grind typical of the operations (a d_{80} size of 68 microns):

1. To produce high-confidence reference distributions of ore grade,
2. To perform high-confidence replicate rougher flotation tests,
3. To characterise the grade/recovery performance,
4. To test the sensitivity of grade and recovery to different grinds,
5. To confirm the End-Member categorisation by way of the above three attributes.

The parts of this End-Member study relevant to the present work pertain to the flotation tests conducted at the standard grind of d_{80} size 68 microns. Summarised results of these reference distributions are shown in Table 4. At the time of this investigation, the Massive Sulphide group was subdivided into Massive and Semi-Massive sulphides. For the purposes of simplicity only the nickel and copper assays are shown.

Table 5 - Summary of Ore Grades: Raglan End-Member Set

End Member	**Massives**	**Semi-Massives**	**Net-Textured**	**Disseminated**
Nickel				
Mean Grade %	13.08	9.18	4.42	1.89
Sample Std. Dev. %	0.32	0.06	0.03	0.02
Relative Standard Deviation %	2.41	0.69	0.66	1.22
No. of samples, n	10	10	10	10
Copper				
Mean Grade %	2.16	1.37	1.11	0.56
Sample Std. Dev. %	0.08	0.02	0.01	0.01
Relative Standard Deviation %	3.90	1.18	0.90	1.13
No. of samples, n	10	10	10	10

A clear trend of increasing grade from disseminated to massive end-members is seen for both nickel and copper. The nickel grade/recovery curves for nickel are shown in Figure 3.

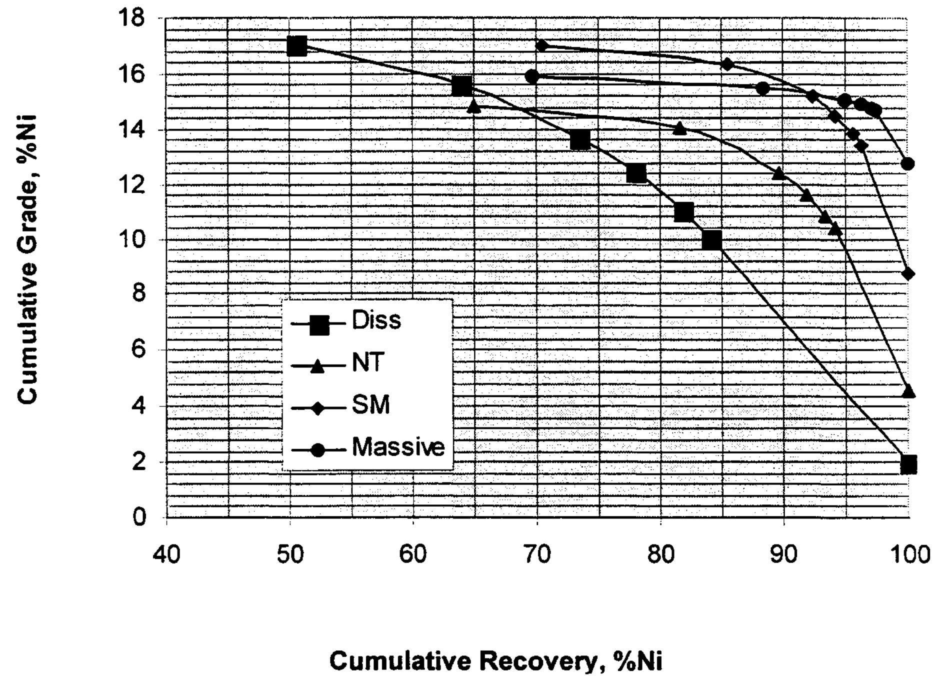


Figure 3 - Grade/Recovery Curves for Nickel for Individual End-Members of Raglan Ore: Rougher Flotation at High-Confidence

The rougher recoveries, measured from a grade/recovery graph at a constant grade of 11% Ni grade, were as individual. These are shown in Table 6.

Table 6 - Summary of Ni Recoveries: Raglan End-Member Set

End Member	Massives	Semi-Massives	Net-Textured	Disseminated
Nickel				
Recovery %	95.7	93.0	93.1	81.8

The equivalent copper data follow.

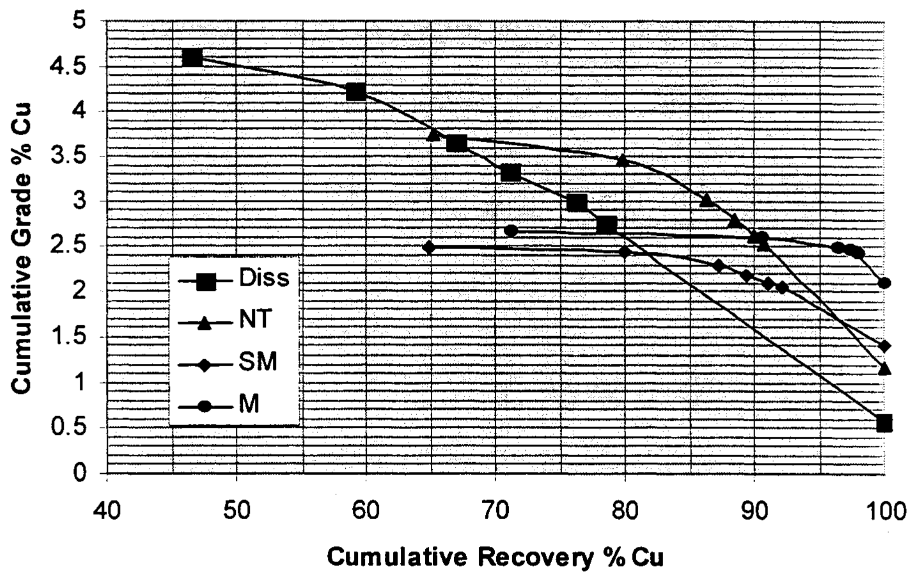


Figure 4 - Copper Grade/Recovery Curves for Individual End-Members

Table 7 - Summary of Cu Recoveries: Raglan End-Member Set

End Member	Massives	Semi-Massives	Net-Textured	Disseminated
Copper				
Recovery %	97.5	92.5	82.3	73.3

These nickel and copper recoveries show a clear trend of higher recoveries towards the massive sulphides and higher ore grades. It was subsequently shown that this is largely driven by differences in mineral grain size and abundance of sulphides. Grain size measurements of pentlandite and chalcopyrite made by mineralogical studies showed the following results **[Kormos, 2004]**:

Table 8 - Mean Grain Size Measurements: Raglan End Member Set

End Member	Pentlandite microns	Chalcopyrite Microns
Massive Sulphides	291	106
Net-Textured Sulphides	78	43
Disseminated Sulphides	68	36

Given the differences in paymetal grade between these end-members, the composite nature of the distribution of nickel assays in Rougher Flotation Feed becomes obvious (Figure 8). It was further concluded from this End-Member study that the grade/recovery curves of the Massive and Semi-Massive End-Members were similar enough to warrant regrouping them as one End-Member, now called Massive Sulphides. It was also concluded from this and the ensuing survey work that the grades and recoveries of the concentrator operations are largely driven by the daily variations in the End-Member mix of ore milled.

2.4. Summary – Shortcomings of Existing Model

One of the specific objectives of this thesis is to critically review the existing Falconbridge survey model, and develop improvements thereto. The following initial observations are made:

1. The Internal Reference Distribution performs a check on itself by inspecting the individual flotation feed grades against the upper and lower confidence limits at two sample standard deviations from the sample mean. A more robust check could be made using either the Grubbs T model or Sichel's t-estimator.

2. There is no statistical testing of the significance of any difference between the External and Internal Reference Distributions, for example, average, variance.

3. The residual lognormality of the distribution of rougher float feed grades has not been recognised or accommodated. It is acknowledged that the degree of lognormality in this dimension is not as high as is typically found in the spatial dimension, however diagnostic tests should be constructed to investigate this issue.

4. The possibility of auto-correlated time effects in the two reference distributions should be investigated and managed.

5. More robust formulation of the size of the Internal and External Reference Distributions is suggested. The 'economic optimum' number of survey units, and the appropriate selection and sizing of the External Reference Distribution, should take the characteristics of these distributions into account.

In order to address these shortcomings, some relevant tools will be presented in the next chapter.

CHAPTER 3 - STATISTICAL REVIEW

3.1. Summary of Existing Model Shortcomings and Proposed New Tools

From the list of shortcomings of the existing Falconbridge survey model, the following table offers probable compensations for the listed shortcomings. At this stage, Table 9 lists suggestions rather than concrete solutions. Only the process of experimentation and validation will later confirm which of these probable compensations will be appropriate for the survey model.

Table 9 - Probable Compensations for the Falconbridge Survey Model

Identified Shortcoming	Proposed Compensation
Internal Reference Distribution Performs a Check on Itself – too forgiving	Grubbs Outlier Test
No Statistical Testing of Difference In Sample Means of Internal and External Reference Distributions	Use the appropriate t-test
Residual Lognormality in Time not Recognised	1. Transform Reference Distributions to Lognormal 2. Use Models of Krige and Sichel
Auto-Correlated Time Effects not Recognised	3. Examine Option of Basing Measurement on Differences rather than Raw Observations: Formal Reference Distribution Method 4. Semi-Variogram
More Robust Reference Distributions Required	5. Consider Larger n for External Reference Distribution 6. Use Sichel t-Estimator for Internal Reference Distribution

3.2. The Grubbs Outlier Test

Grubbs, 1969, reported a model for discriminating replicate observations for outliers. An outlier may be merely an extreme manifestation of the random variability inherent in the data. If this is true, the extreme observation should be retained within the accepted data. Conversely, an outlier may be the result of gross deviation from a standard procedure, or an error in calculating or recording the numerical value. In the latter case, it is desirable to conduct an investigation into the reason(s) for the aberrant value. The observation may or may not be rejected as a result. The Grubbs outlier model arranges the replicate data in ascending order. The sample mean and standard deviation are then calculated. In other words the set of x_i are arranged

$$x_1 > x_2 > x_3 \ldots\ldots\ldots\ldots > x_n \qquad [1]$$

for n observations, thus x_n is the smallest, and x_1, the largest, observation. The Grubbs T parameter is then calculated for all data points as

$$T_i = \frac{(x_i - \overline{x})}{s} \qquad [2]$$

where $\overline{x}$ and s are the sample mean and standard deviation. The units of T are thus the same as those of the standard deviation. Grubbs derived the critical values of T for small data sets up to n=25 for various significance levels. An extract of some of these values for the 5% significance level is shown in Table 10. These tables are available for values of n up to 149. An example of this model was given by **Grubbs** in his **1972** paper, in the context of measurements of muzzle velocity of a batch of match grade ammunition of 7.62 mm NATO grade cartridges.

Table 10 - Critical Values of Grubbs T

n	T crit	n	T crit
3	1.15	9	2.11
4	1.46	10	2.18
5	1.67	11	2.23
6	1.82	12	2.29
7	1.94	13	2.33
8	2.03	14	2.37

The results are summarised in Table 11. Two measurements of velocity were made: one by the ECI counter, the other by the Oehler counter. The data are in the form of differences in muzzle velocity (in feet per second) between the two measurements.

Table 11 - Sorted Data - Grubbs 1972

Obs. #	Difference Between Methods f/sec	Multiple of Standard Deviations from Mean Difference	Obs. #	Difference Between Methods f/sec	Multiple of Standard Deviations from Mean Difference
23	-1.00	1.81	34	0.00	0.29
38	-0.78	1.47	46	0.00	0.29
17	-0.64	1.26	39	0.15	0.06
25	-0.64	1.26	44	0.15	0.06
36	-0.63	1.24	49	0.46	-0.42
37	-0.63	1.24	11	0.47	-0.43
47	-0.49	1.03	14	0.47	-0.43
21	-0.48	1.02	40	0.47	-0.43
30	-0.48	1.02	13	0.62	-0.66
45	-0.48	1.02	43	0.63	-0.67
41	-0.47	1.00	35	0.67	-0.74
16	-0.32	0.77	12	0.77	-0.89
22	-0.32	0.77	6	0.78	-0.90
24	-0.31	0.76	15	0.78	-0.90
26	-0.31	0.76	7	0.92	-1.12
31	-0.31	0.76	10	0.95	-1.16
27	-0.17	0.54	19	0.95	-1.16
29	-0.16	0.53	20	0.96	-1.18
32	-0.16	0.53	3	1.09	-1.38
42	-0.16	0.53	9	1.10	-1.39
48	-0.16	0.53	4	1.27	-1.65
33	-0.15	0.51	5	1.27	-1.65
1	0.00	0.29	2	1.38	-1.82
18	0.00	0.29	8	1.44	-1.91

The discrimination of any particular datum as a potential outlier is thus referred to a table of critical values of Grubbs' T value. For example, the apparently extreme difference in measurements for observation 8 (1.44 feet/second) produces a T value of –1.91. For n=49 the Grubbs T at the 10% level is 2.76, so even the largest apparent difference in measurement methods is not significant. The two methods of measurement of muzzle velocity can thus be regarded as comparable. This approach is based on the Normal Distribution.

3.3. Comparison of Means

The comparison of two mean values comprises a very basic and standard segment of elementary statistics. The following review is presented for those readers who may not be familiar with this procedure.

For the case where two data sets were obtained from the random sampling of an unknown true population, a method of pooling the variance estimates to obtain an overall standard error for the difference of observed sample means m_1 and m_2 was derived, and referred to a t-test **[Box. *et. al.*, 1978, Chap. 3, pp.74-77]**. The salient features of this approach are reviewed.

The variance, V, of the difference in means m_1 and m_2 with n_1 and n_2 observations, is

$$V_{(m_1-m_2)} = \sigma^2.(\frac{1}{n_1}+\frac{1}{n_2}) \qquad [3]$$

with σ^2 being the unknown true population variance. The standard error of the difference, $s_{\varepsilon(\Delta)}$, is then

$$s_{\varepsilon_\Delta} = \sigma.(\frac{1}{n_1}+\frac{1}{n_2})^{0.5} \qquad [4]$$

Even if the distributions of the original observations had been moderately non-normal, the distribution of the difference m_1 - m_2 between sample means drawn from (in this case) 10 samples for each mean, would be expected to be normal because of the Central Limit Theorem. An estimate of a z score (from the Normal Distribution) could then be made using

$$z = \frac{[(m_1 - m_2) - (\eta_1 - \eta_2)]}{[\sigma.(\frac{1}{n_1} + \frac{1}{n_2})^{0.5}]} \qquad [5]$$

where η_1 and η_2 are the true unknown means of the populations.

The value of σ may be estimated from a robust sample standard deviation, which is drawn from a larger parent distribution of the same system being sampled.

3.4. The Normal Distribution

The distribution of measurements about some true mean value has properties that either assist the investigator or cause a hindrance to the accurate estimation of the mean. Starting with the symmetrical distribution called the 'Normal' or 'Gaussian' distribution, the values of y are distributed about a mean η with a standard deviation σ according to

$$f(y) = \frac{k}{\sigma}.e^{-\frac{(y-\eta)^2}{2\sigma^2}} \qquad [6]$$

where

k	=	a constant (often shown as $1/\sqrt{2\pi}$)
σ	=	the population standard deviation
η	=	the population mean
y	=	the distributed variable

This distribution has the property that the logarithm of its probability density is a quadratic function of the standardised error (y-η)/σ **[Box *et. al.*, 1978, p.43]**. Two main features of the normal distribution are: -

- The central limit effect, which produces a tendency for real error distributions to have 'normal' characteristics,
- The robustness of many popular statistical test procedures to deviations from theoretical normality.

Especially when sampling in replicate format is performed, such as in a scientific experiment where several measurements of a parameter are made, the errors of measurement of the sample mean tend to be normally distributed about the mean error. In particular, when a minimum of 4-5 observations are taken and averaged, this desirable state of affairs develops **[Grant and Leavenworth, 1988]**. This is a direct consequence of the Central Limit Theorem. Defining the overall error of measurement as ε, where

$$\varepsilon = (y-\eta) \qquad [7]$$

This overall error ε will be the sum of the individual, or component, errors according to

$$\varepsilon = \sum_{i=1}^{n} a_i . \varepsilon_i \qquad [8]$$

where

a_i = constants
ε_i = the set of independent component errors,
n = the number of error sources.

The Central Limit Theorem states that for real data sets the distribution of such a linear function of errors will tend to normality as the number of error sources (components) becomes large, almost regardless of the individual distributions of the components. It is assumed in stating this important relationship that the component errors ε_i make important contributions to the overall error ε, and that no single error dominates over the others.

It is most desirable that in a sampling arrangement, the total error of measurement be normally distributed. In other words, this means that the errors of sampling are symmetrically arranged about their mean. This arrangement places the investigator in a position to use simple diagnostic statistical tests to audit the data. A further feature of this arrangement is that the estimates of sample average tend to be normally distributed, even though the observations upon which these estimates have been made are not. Therefore, statistical methods that depend, not directly on the distribution of individual observations, but on the distribution of one or more averages of observations, tend to be robust to non-normality.

The squared term in Eq. 6 creates symmetry. Exclusion, or tail, areas are shown in Table 12. The symmetry of the Normal Distribution allows for simple use. Table 12 is only a subset of the full distribution, which is given for only half the sample space, *i.e.* $0 \leq (y-\eta)/\sigma \leq 3.5$ (which describes in excess of 49.99% of the population). Confidence intervals, by extension, are symmetrical, as are exclusion intervals. This function is often demarcated in multiples of the population standard deviation σ.

Table 12 - Selected Tail Areas of the Normal Distribution

Multiple of Standard Deviation σ	**One-tailed Area**
$x \geq \eta+\sigma$	0.1587
$x \geq \eta+2\sigma$	0.0228
$X \geq \eta+3\sigma$	0.0013

This infers that for the two-tailed areas, *i.e.* for a deviation to occur in either direction of the mean, will be twice these amounts, i.e. 0.3174 for σ, 0.0456 for 2σ, and 0.0026 for 3σ.

3.5. The Lognormal Distribution

The reality of paymetal values in an orebody being distributed on a lognormal basis was first identified and modelled by Krige in 1951. His initial work was based on the gold ore deposits of the Witwatersrand and Free State, in South Africa, and showed that by transforming the raw gold grade data with the Naperian Logarithm, the typical right-hand skew (lognormal) distribution of gold values assumed a symmetrical, Normal Distribution shape. The importance of this transformation lies in the reliable estimation of the mean. When estimated from the transformed distribution, the mean is more reliably obtained **[Krige, 1951a, 1981]**.

In an exercise performed on Free State borehole data, **Krige** showed in **1952** that the lognormal frequency curve could be applied to the observed distribution of values in the Main Sector of the Free State goldfields. The model was used to estimate average recovery grade and milling tonnage. He stated that random sampling from this type of distribution would be biased. The basis of this transformation changes the raw scores of metal grade (*otherwise understood as unaltered measurements of paymetal concentration in the ore samples, as determined by a chemical analysis process*) according to the following probability density function:

$$\psi(x) = \frac{1}{\sigma.(2\pi)^{0.5}}.e^{-\frac{(x-\eta)^2}{2\sigma^2}} \qquad [9]$$

where

x	=	ln(z+a]
z	=	raw score grade (or measured and uncorrected grade)
a	=	a constant
η	=	mean of ln(z + a)
σ^2	=	logvariance

Two-Parameter Distribution

For the two-parameter distribution, a=0, else it has a positive value, in which case the distribution is three-parameter. At the time of publication, which was in the 1950's, it was understood that the third term 'a' was a bias. Several decades later, it is more simply interpreted as a necessary parameter to be used in cases where a curvature appears in the probability plot, and which is necessary for correct estimation of the mean. A generic example of a lognormal probability plot of a two-parameter lognormal distribution of metal grades in ore is shown in Figure 5.

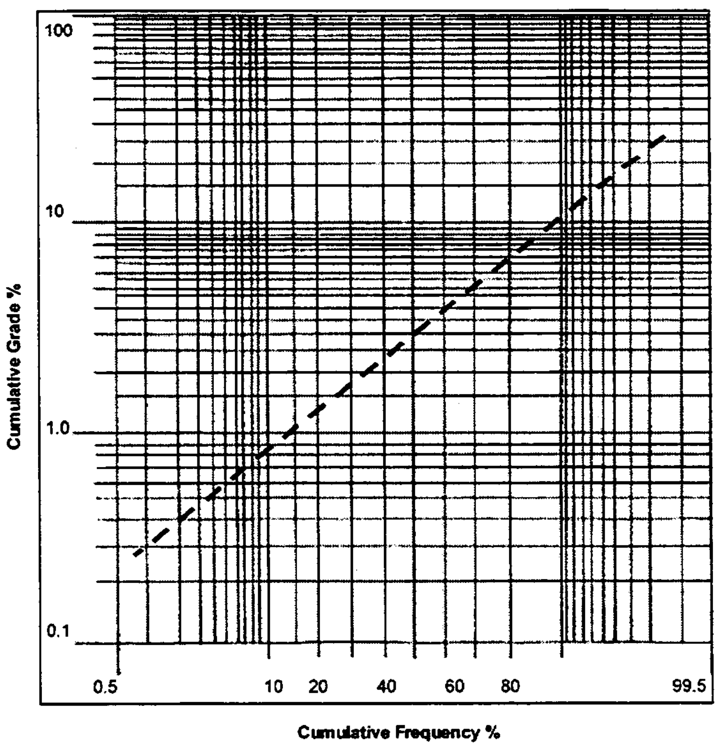


Figure 5 - Two-Parameter Unimodal Lognormal Distribution Probability Plot
Generic Example of Probability Plot

Diagnostic modelling of a lognormal distribution is advisable prior to the abovementioned estimation of the mean. **Clark and Garnett** showed in **1974** that this was a necessary first step for the ore reserve estimation for low-grade mineral concentrations. In these, the estimation was usually made on the assumption that the paymetal grades follow the lognormal distribution. Multiple paymetal mineral hosting causes a mixture of lognormal distributions because of the different tenors of paymetal concentration in each host. Clark and Garnett showed that by decomposition of the overall mixed distribution, a more reliable estimate of the mean grade is derived. A lognormal probability plot method was used for this purpose, in which the Naperian logarithm transform linearises the cumulative plot of metal grade and frequency. For a unimodal logarithmic distribution, one single straight line is obtained (Figure 5), provided that the frequency is two-parameter.

Three-Parameter Distribution

For a three-parameter distribution, the linearity may be lost towards the lower grade end of the distribution, giving over to a curve. In such a case, Equation 10 is used to estimate the value of the third parameter 'a' by using estimates of f_1 and f_2 from Fig. 6 **[Krige, 1960; Rendu, 1981]**, according to

$$a = \frac{(m^2 - f_1.f_2)}{(f_1 + f_2 - 2m)} \qquad [10]$$

where

m = median of the distribution

a = the third parameter

f_1 = grade corresponding to p

f_2 = grade corresponding to (1-p)

p = a cumulative frequency selected between 5 and 20% frequency of the distribution.

The choice of p is empirically determined. Various values of $5 < p < 20$ are chosen, and by a process of trial and error, in which Figure 6 is redrawn as a result of the correction by the trial value of p, a straight line is produced. At this point, the final value of p is accepted. The importance of this third parameter lies in linearising the lognormal probability plot. The checking of lognormal distributions for the third parameter is thus obviously necessary. It is more often a real effect produced by the geological history of the deposit **[Sichel *et. al.*, 1992]**. In such cases, mobilization of the paymetal (in the case of Sichel's study, gold) could only transport certain sizes of sediments at different stages as the river systems gradually reduced their velocities downstream. One would expect a relationship between the gold grade (or size of gold particle) and the distance of the site from the origin of the deposit. Sampling within a defined geographical range would then run the risk of not sampling the entire distribution, leading to the third-parameter effect.

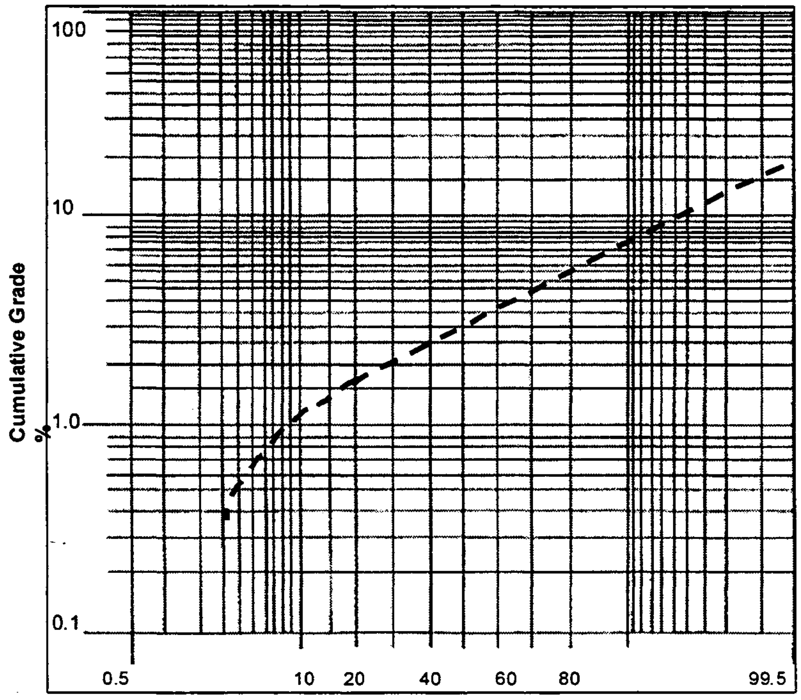


Figure 6 - Three-Parameter Unimodal Lognormal Distribution
Generic Example of a Probability Plot (Before Correction for the Third Parameter)

It will be shown at a later stage of this manuscript that the Raglan system is two-parameter lognormal, not three-parameter lognormal. Therefore, in the case of

Raglan, the third parameter is not an issue, and detailed discussion of the derivation, role and use of the third parameter 'a' is outside the scope of this thesis. The characterisation of distributions of Platinum Group Elements (PGE) was extensively used and discussed for South African Bushveld Merensky ore **[Lotter, 1995a]**. This work characterised the multiple mineral hosting of platinum group elements (PGE) for chip sample data taken from stoping operations of hanging wall, footwall and reef at Rustenburg Platinum Mines. It was shown that specialised sample comminution procedures ahead of fire-assaying physically normalised the compound distribution by producing an artificial one-component mixture of analytical grade silica and sample, called '*pulverite*'. Logarithmic probability plots of replicate assays form pulverite were shown to be unimodal, two-parameter lognormal; whereas replicate assays of untreated sample showed three-parameter, trimodal lognormal. This work further showed that the third parameter could be estimated from Rendu's formula. Application of the estimated value of 'a' straightened out that part of the multiple distribution which was three-parameter. The impact of this treatment had a significant effect in the value of mean grade estimated from the replicate assays. PGE metal balances across laboratory scale flotation tests thus demonstrated lower metal balance errors.

3.6. Sampling the Lognormal Distribution – the Sichel t-Estimator

The estimation of mean grade from the lognormal distribution using very large samples (large n) encounters little difficulty regardless of the shape of the distribution. **Sichel** held in **1966** that this had by that time been demonstrated for orebodies on a world-wide basis. In this context, 'large n' means in the order of several thousand observations **[Hunt, 1994]**. The work of **Krige**, **1962**, showed that for a mill treating a gold-bearing ore, where sampling was performed on an 'ore milled' basis in the time dimension, smaller data sets could be used. In such an application, Krige used 99 consecutive data points. This will be used as a platform for the construction of the External Reference Distribution in the new survey model.

A particular problem arises for the estimation of the mean of these distributions from small data sets, with n generally less than 10 (for example, with drill-core data for a newly discovered mineral resource where less than 10 drill-holes have been completed, and where a decision to further the programme into a mining project has to be made). The arithmetic average is an unbiased but noisy estimator, whereas the geometric average has a lower variance but suffers from bias.

In his earlier publication **Sichel, 1952**, showed that, where the estimation of mean paymetal grade in ore from mining operations was reliably found by the arithmetic mean with very large n, his new t-estimator (definition follows), designed around the correction of the geometric mean by the logvariance, was more efficient. The corrected geometric mean is a more efficient estimator of the mean, as illustrated in Table 13.

Table 13 - Efficiency of Sichel t versus Arithmetic Mean

[For given number of observations in Sichel's t, the table shows the required number of observations to achieve equivalent confidence by the arithmetic mean]

n(t) by Sichel t	**5**	**10**	**20**	**30**	**40**	**50**
σ^2						
0.5	5	10	21	31	41	52
1.0	5	11	22	34	45	57
1.5	6	12	25	39	52	65
2.0	6	14	30	46	62	78
2.5	7	16	36	56	75	95
3.0	7	19	44	69	94	120

Table 12 shows that for low logvariance, the arithmetic mean is as efficient as Sichel's t-Estimator. On the platform of two- or three- parameter lognormal distributions, a reliable estimation of the mean and associated (skewed) upper and lower confidence limits at the 90% level is obtained by the Sichel t-estimator **[Sichel, 1966]** as:

Robust Mean:

$$t = e^{\xi}.\gamma_n(V) \qquad [11]$$

where

t	=	the corrected mean of the distribution
ξ	=	the Naperian logarithmic mean of the distribution
V	=	the logvariance
$\gamma_n(V)$	=	the correction coefficient , derived from the sampled logvariance (V), according to

$$\gamma_n(V) = 1 + \sum_{r=1}^{\infty} \frac{(n-1)^r.V^r}{(2^r.r!)(n-1)(n+1)......(n+2r-3)} \qquad [12]$$

Table 14 gives a limited extract of the values of $\gamma_n(V)$ at the 90% confidence level. The value of the term $\gamma_n(V)$ increases with increasing logvariance, and to a much lesser extent, with increasing numbers of observations. This is a key point in understanding the Sichel t estimator.

Table 14 - Selected Values of the Multiplier $\gamma_n(V)$ for the Sichel t Estimator

Logvariance V	**Value of n**					
	2	4	6	8	10	20
0	1.00	1.00	1.00	1.00	1.00	1.00
0.10	1.05	1.051	1.051	1.051	1.051	1.051
0.20	1.102	1.103	1.104	1.104	1.104	1.105
0.50	1.260	1.269	1.273	1.276	1.277	1.280
1.00	1.543	1.58	1.598	1.608	1.615	1.630
1.50	1.848	1.938	1.981	2.007	2.025	2.065

Confidence Limits:

The upper confidence limit UCL is obtained as

$$UCL = t.e^{\frac{\sigma_t^2(V)}{2} + T.\sigma_t(V)} \qquad [13]$$

The lower confidence limit LCL is given by

$$LCL = t.e^{\frac{\sigma_t^2(V)}{2} - T.\sigma_t(V)} \qquad [14]$$

The parameter T, expressed either as T_p or T_{1-p}, is a new standard deviate defined as

$$T = \frac{(\ln(t) - \ln(\theta_\alpha))}{\sigma_t(V)} + \frac{\sigma_t(V)}{2} \qquad [15]$$

The standard deviate T, estimated by a complex integral, is robust to change in the parameter $\sigma_t^{\ 2}$, and is difficult to estimate, since it is not readily algebraically isolated. For this, **Sichel, 1966**, and **Clark, 1987**, proposed a convergence method. This is done by the software called TRIPOD [**Clark, 2000**]. Neither can one use the Student t in the Naperian domain to address this issue, as it produces incorrect confidence limits. Application of this narrower confidence interval to individual measurements will result in a rejection rate much larger than 5% (*i.e.* 1-alpha).

Whereas the limits of the Naperian logarithm of mean grade are symmetrical, the limits of the mean grade itself are not. These limits are actually skewed about the grade. This appropriately reflects the character of the lognormal distribution. The foregoing review of the normal and lognormal distribution models suggests that, for the estimation of the mean of a lognormal distribution,

1. The arithmetic mean is unbiased but noisy,

2. The geometric mean is less noisy but biased,
3. The corrected geometric mean, per Sichel's t-Estimator, is more appropriate, since it uses correction terms for the geometric mean according to n and the logvariance,
4. The Sichel t-estimator corrects the bias of the uncorrected geometric mean.

3.7. The Semi-Variogram

The formal semivariogram, located in the field of geostatistics, was developed by Kolmogorov, and later improved by Matheron in 1960, to identify in spatial distance the area of influence in which paymetal grades in ore were correlated **[Clark, 1993a, Lotter, 1995b]**. The model relies on the auto-correlation of grade data in space and in time.

To prepare a semivariogram, spatially logged paymetal grades, for example in a North-South direction, are assembled in a sequence. The semivariance is calculated as

$$\gamma_h = \frac{1}{2N} \cdot \sum_{1=1}^{N} (g_i - g_j)^2 \qquad [16]$$

where

g_i = grade at position i

g_j = grade at position j

N = the number of data

The settings of i and j are systematically varied, viz. set at 1 distance unit apart to start with, then by two distance units, then 3, and so on. The semivariance for each spacing is thus derived from a summation of the individual pairs of data that correspond to the chosen spacings. The spherical Matheron semivariogram (Figure 7) thus shows a three-component model. The spacing is represented on the x-axis by the term **'h'**. The semi-variance is represented by the term $\gamma(h)$. The first component is for zero spacing, and is C_0, otherwise called the 'nugget effect'. At zero spacing, this means measurements such as the two halves of split drill-core. From this point to $C+C_0$, an area from zero spacing to **'a'** in the x-axis shows an increase in the y-axis semivariance $\gamma(h)$ for every successive increase in spacing. After the short-term spacing, **'h'**, exceeds **'a' (i.e. h>a)**, the semivariance no longer increases. This is termed the **'sill'** of the semivariogram, and is found in units of semivariance $\gamma(h)$ on the y-axis.

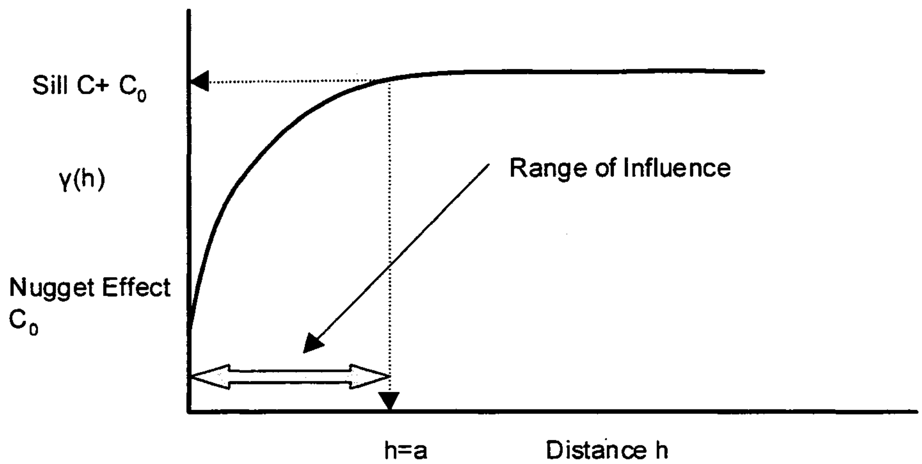


Figure 7 - The Semivariogram

The algebraic expression of this semivariogram is a three-component function, *viz.*

$\gamma(0)=0$	h=0	[17]
$\gamma(h)=C[\frac{3h}{2a}-\frac{0.5h^3}{a^3}]+C_0$	0<h<a	[18]
$\gamma(h)=C+C_0$	h>a	[19]

Other forms of the semivariogram exist. The foregoing by Matheron is the simplest.

Since time is a subset of space, the semivariogram applies as logically to the variation of paymetal grade in mill feed in time. As the spatially distributed metal values of ore are mined in stopes, the blasting, ore transport, crushing and grinding processes treat the same ore in the dimension of time as mill feed. The logvariance, however, is lower, since the ore has been mined from several stopes, crushed and blended prior to presentation to the primary mill. Thereafter, when sampled as Rougher Flotation Feed after the primary mill, it has been further mixed and reduced in size. This process, however, increases the level of autocorrelation in time in Rougher Flotation Feed.

3.8. Assessment of Operating Plant Performance

Assessment of Mill Feed Grade in Time

Krige, 1962, showed that the problem of controlling mill feed grade for a gold mine could be compensated for by a statistical approach with a control graph of daily mill feed values in time across a production month. In this case, 'mill feed' was the Run-of-Mine ore prior to milling. The lognormal distribution of the daily values was dealt with by a Naperian log transform. He concluded that the appropriate level for this type of monitoring was to calculate the skewed upper and lower confidence limits at the 2.5% tail area. This provided data limits against which daily measurements could be compared. This work aimed at the provision of a timely measurement of daily gold input to the 'reduction works', or concentrator, so as to ensure daily production of refined gold within a desired range of targets. In this work, Krige used 99 consecutive daily observations of 'daily mill yield plus pulp' grade, equivalent to a 99-day continuous milling period. *['Daily Mill Yield Plus Pulp'* grade means the calculated head grade of gold based on dissolved gold (or refined gold bullion) plus gold found in cyanidation residues]. The calculations were all performed in logarithmic transform, and the inverse transform values obtained thereafter. He claimed that this model, when set out in a daily plot of grades, and overlain with the upper and lower confidence limits of daily gold grade data at the 95% acceptance level, provided the mine management with a practical daily quality control on operations. The limits set by this system were not symmetrical about the mean ore grade, since they were derived from the Naperian logarithm base. Krige stated that, from the two data sources (either ore milled or as mill yield plus pulp), the acceptance limits were wider for the ore milled than for the mill yield plus pulp. This reflects the higher variance in unprocessed ore and is consistent with the theory of **Cochran, 1946.** No account was offered in this work for any time-correlated effects.

This work demonstrated that the dimension of time was appropriately treated by a lognormal approach for grade of ore milled, and was later confirmed by **Clark, 1993b.**

This piece of work by Krige provided three key findings:

1. That the (spatial) lognormal distribution of paymetal grades in ore mined was applicable in the time dimension as ore milled.
2. The control limits so established at the mill, in terms of 'ore milled', were asymmetrical about the mean.
3. That the 95% level of quality control was recommended in this application.

A the time of writing this thesis, *i.e.* 2005, it is recognised that the sampling of ore milled for gold (at several grammes per tonne) is far more difficult than the sampling of high-grade base metal ores in Canada (at several percent). The position of the Internal Reference Distribution in *Statistical Benchmark Surveying* **[Lotter and Whittaker, 1998]**, relative to this model is, that the Krige distribution here described provides a robust examination of a small data set.

Correlation Between Mill Feed Grade and Plant Recovery

Napier-Munn, 1998, reported that an auto-correlation existed between mill feed grade and plant recovery of a concentrator. This correlation was used to analyse plant trials where two operating conditions are compared, for example reagent regimes or flotation circuit configuration. Where the shape of the overall grade/recovery curve is not linear, Napier-Munn proposed that, for the limited domain of plant operation in which close control is exercised on grade and recovery within desired limits, a linear approximation of the grade-recovery curve is adequate. This method was suggested in cases where plant historical data were being examined, or where the formal design of experimentation, such as a factorial, was not possible.

The relevance of this work to the present study is that higher feed grades generally correlate with higher flotation recoveries. In the existing heuristic Falconbridge statistical benchmark surveying model, the mill feed grade is used as a reference point to establish that the survey(s) were performed during a period which showed a typical mill feed grade. With the publication by Napier-Munn, 1998, in which the autocorrelation between mill feed grade and recovery was studied, the usefulness of the mill feed grade parameter is demonstrated. At a later stage of this thesis, it will be shown that it is simpler and more effective to obtain samples of typical ore grade than to try and correct for it. This is because we would otherwise have to correct accurate Qem*SCAN measurements, which would be a complicated procedure requiring a long list of assumptions about particle composition and liberation, introducing even larger errors. It will be a specific objective of this study, therefore, to obtain the correct, or typical, head grade.

Detecting Small Performance Improvements in a Concentrator

Napier-Munn, 1995, addressed the problem of assessing a concentrator performance change in time, where distinct time-based trends were present in the data. This approach was intended to break the relationships and effects of auto-time correlation by performing the evaluation on the basis of differences rather than of actual observations. Although his focus was on detecting small changes in recovery of paymetal across the concentrator operation, much of the discussion is relevant to the subject of sampling mill feed in time. He observed that the mill feed grade significantly influenced the recoveries, and stated that three assumptions had to be made before this type of test was applied:

1. That the ore milled metal grade data were normally distributed,
2. That each data point is a random independent sample of the population of experimental outcomes, under those conditions,
3. That the two sample variances, i.e. of the 'A' (treatment on) or the 'B' (treatment off) data blocks are estimates of the same population variance.

His case studies dealt with samples in the range of n ~ 50-60. By random on-off switching of the process condition which was being evaluated, for example the operation of a regrind mill, the time-correlated effects in mill feed grade were removed, and the true difference in recovery attributable to, for example, the operation of the regrind mill, could be measured and tested for by the Analysis of Variance (ANOVA). Mean differences in recovery of paymetal(s) attributable to the process change, once proved significant, were elaborated with appropriate confidence limits using

$$CL = \frac{t_x s}{n^{0.5}} \qquad [20]$$

where

CL	=	the confidence limit of the mean difference
t_x	=	the critical value of t for the x confidence level
s	=	the standard deviation of the difference
n	=	the number of observations.

The confidence limit was then either added to or subtracted from the mean difference in recovery to find the upper and lower confidence limits of the mean difference.

It will be shown at a later stage of this manuscript that the first assumption of Napier-Munn, *viz.* that the ore milled grade data are normally-distributed, is not a safe assumption, and needs to be tested for prior to the investigator undertaking the plant trials. Where Napier-Munn is correct, however, is that the distribution of differences is normally-distributed and independent of any time-correlated effects or trends.

CHAPTER 4 - APPLICATION OF NEW TOOLS TO THE RAGLAN OPERATIONS

4.1. The Case Of Raglan

The Raglan operations, located in the Ungava peninsula of northern Québec, which were commissioned in December 1997 and play a key role in present nickel production for the Canadian Integrated Nickel Unit (INO) of Falconbridge Limited, were chosen to be the site where most of the present study would be conducted. This is because the potential economic impact at this site is the greatest within existing Falconbridge operations. Earlier work had already shown that first-generation Process Mineralogy in Falconbridge, using the prototype surveying model, produced meaningful information that was successfully used to improve flowsheet performance **[Lotter *et.al.*, 2002]**.

During the five-year period 2004-2009, the Raglan operations will encounter a change of ore characteristics that will potentially cause nickel production to be reduced. *[The first six years of operations deliberately chose to mine higher-grade ores so as to deliver an early cash flow into the project revenues].* The life-of-mine grade of ore is lower than has been mined and milled to date, and has a higher SAG Power Index (SPI) **[Starkey *et. al.*, 2003; Starkey, 2001; Langlois and Holmes, 2001]**. This change in grade and ore hardness will result in either coarser grinds, with attendant reduction in paymetals recovery, or in reduced milling rates, unless grinding capacity is increased. The Process Mineralogy Group at the Metallurgical Technology Centre of Falconbridge, based in Sudbury, Ontario, was appointed to survey, sample, test and interpret the first six years of operations, so as to develop a clear picture of future processing requirements with a flowsheet retrofit. This milling operation was designed with conventional Mineral Processing. It was recognised, therefore, that further performance potential existed, and could be identified and realised, by using Process Mineralogy - for the reasons enumerated earlier in this text.

This was done by two statistical benchmark surveys, one in June 1998, the other in February 2000 **[Lotter and Whittaker, 1998]**, and was supported by the definition and sampling of the three end-members. High-Confidence Flotation Testing was also extensively performed on these ore samples to establish flotation characteristics and optimisation opportunities **[Lotter, 1995a, Fragomeni and Boyd, 2003]**.

Two capital projects are presently active as a result. These are:

- A Semi-Autogenous Grinding (SAG) Mill retrofit, to convert the present fully autogenous grinding mill to semi-autogenous format. This will include the installation of more grinding power and improved classification to handle the increased ore hardness.
- A flotation circuit retrofit, wherein improvements to the primary and cleaner flotation sections will be installed.

Both of these projects are the direct result of the information obtained from the heuristic Falconbridge 'statistical benchmark survey' model, which was reviewed in an earlier section of this document. It was a requirement that, prior to confirming final designs in both projects, a campaign of specially selected 'future' ore was mined and milled during a seven-day period in November 2003. The definition of the 'future' ore was extracted from medium to long-term mining plans. The life-of-mine is approximately 20 years. The first new 'statistical benchmark survey', ie the product being developed in this thesis, would be put to the test. The ore mixture had to mimic the following blend of end-members to produce a blend as follows:

- Massive Sulphides : 9%
- Net-Textured Sulphides : 75%
- Disseminated Sulphides : 16%

Detailed co-ordination of the plans for these activities were made, involving several disciplines and teams at both the Raglan operations site and at the Metallurgical Technology Group in Sudbury. These plans commenced in January 2003 in anticipation of the survey being performed in November 2003. The sourcing and tonnages/grades of the various mining (working) faces in the operations is summarised in Table 15.

Table 15 - Sourcing of Raglan Campaign Ore
[Data Source : Mine Planning]

Net-Textured Sulphides (75% : 13 500 tonnes)		
Source	**Tonnes**	**Estimated Grade % Ni**
11515	2000	2.5
CLH	2750	2.2
Champagne	2750	2.1
Q1350	2000	3.3
W1265	2000	3.8
LL-1330	2000	2.7
Total Net-Textured Sulphides	13500	2.7
Disseminated Sulphides (16% : 2880 tonnes)		
I pit	2880	1.7
Massive Sulphides (9% : 1620 tonnes)		
S1370	1620	6.0
Overall Blend (100% : 18 000 tonnes)		
Blend	18 000	2.83

4.2. Raglan Drill Core Database 2000-2008 : Lognormality in Space

Characterisation of the Spatial Distribution

The ore reserve drill core for Raglan was sampled in order to provide a robust set of samples for SAG Power Index testing at Minnovex Ltd in Toronto. The initial purpose was to develop a predictive data base for milling rate estimation at the mill operations. These samples were also analysed for nickel. A distribution of nickel grades was constructed from 205 drill core assays, spatially representing the ore reserve for mining operations 2000-2008. The distribution results are summarised in Table 16 and Figure 8.

Table 16 - Distribution of Nickel Grades - Raglan Ore Reserve Drill Core

Grade Bin % Ni	Frequency %	Cum. Frequency %
0.5	0.0	0.0
1.0	0.5	0.5
1.5	2.9	3.4
2.0	5.9	9.3
2.5	12.7	22.0
3.0	21.5	43.4
3.5	11.7	55.1
4.0	10.7	65.9
4.5	9.8	75.6
5.0	5.4	81.0
5.5	4.4	85.4
6.0	4.4	89.8
6.5	2.9	92.7
7.0	2.4	95.1
7.5	0.0	95.1
8.0	1.0	96.1
8.5	0.5	96.6
9.0	0.5	97.1
9.5	1.0	98.0
10.0	0.5	98.5
10.5	0.0	98.5
11.0	0.0	98.5
11.5	1.0	99.5
12.0	0.0	99.5

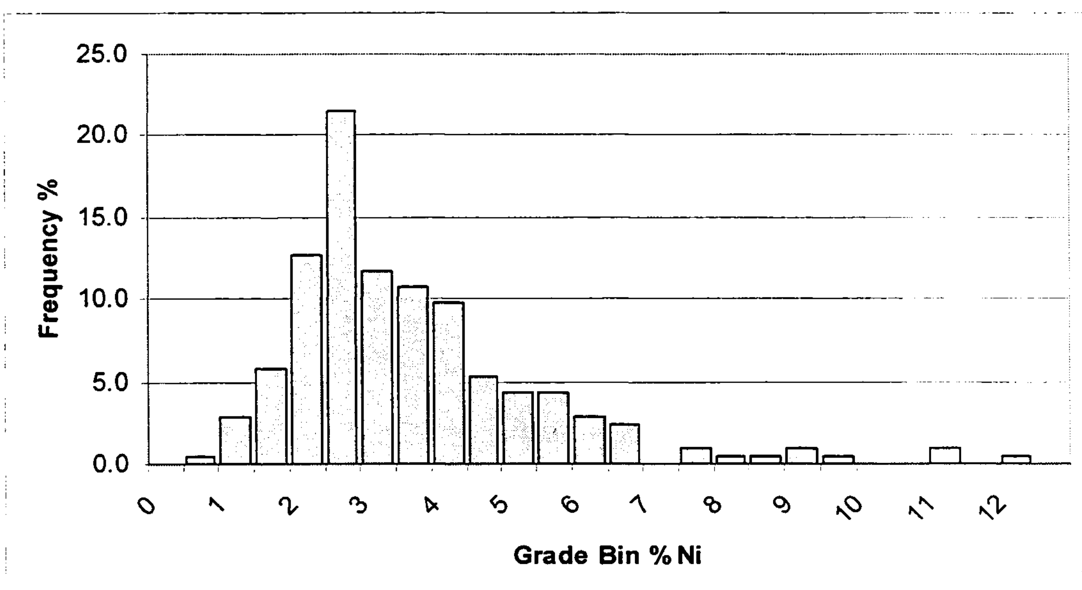


Figure 8 - Distribution of Nickel Grades : Raglan Ore Reserve Drill Core

When the above drill core data are transformed into the Naperian logarithm base, a more symmetrical distribution is apparent. This transformed distribution is plotted in Figure 9.

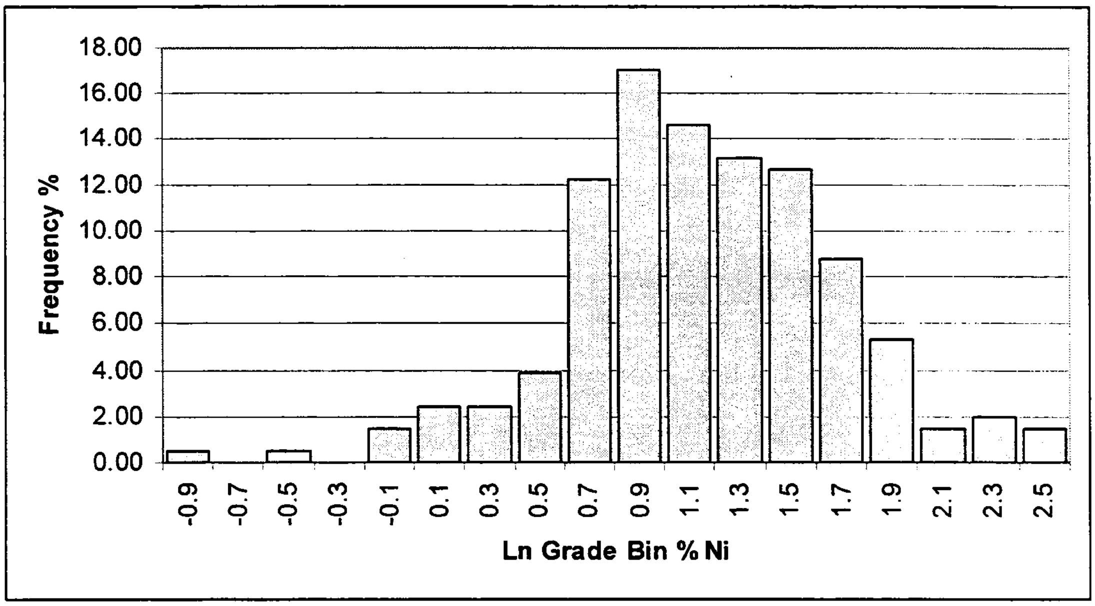


Figure 9 - Naperian Log Transform Distribution of Raglan Drill Core : Ni

When the cumulative grades are plotted against cumulative frequency on a lognormal probability plot, as in Figure 10, it is clear that the nickel grades are lognormally distributed. This observation is in keeping with an abundance of literature dating from the 1950's onwards, in which workers such as Krige and Sichel laid the foundations of the lognormal distribution of paymetals in the spatial dimension. What is further evident is that the grades in Figure 10 follow a two-parameter, not three-parameter, lognormal model. (No evidence of curvature on the left-hand side of the plot is seen). It is also clear that there are three distributions, separated by two points of inflection A and B, at 9.5 and 95.5% of the distribution respectively. In other words, this spatial distribution might be described as compound lognormal. This interpretation is consistent with the work of **Clark and Garnett, 1974**, in which multiple mineral phases carrying the same paymetal were shown to plot as straight lines of different gradient in the overall plot. Using the Raglan End-Member model, this plot suggests an obvious correlation to the three End-Members of the Raglan system. Further work on this topic should be identified and pursued, however this falls outside the scope of this thesis. The first distribution (0-9.5% frequency) has a sampling space from 0.5 to 1.5% Ni. The second distribution (9.5 to 95.5%) has a sampling space of 1.5 to 6.5% Ni. The third distribution (95.5 to 100%) has a sampling space of 6.5 % Ni or higher.

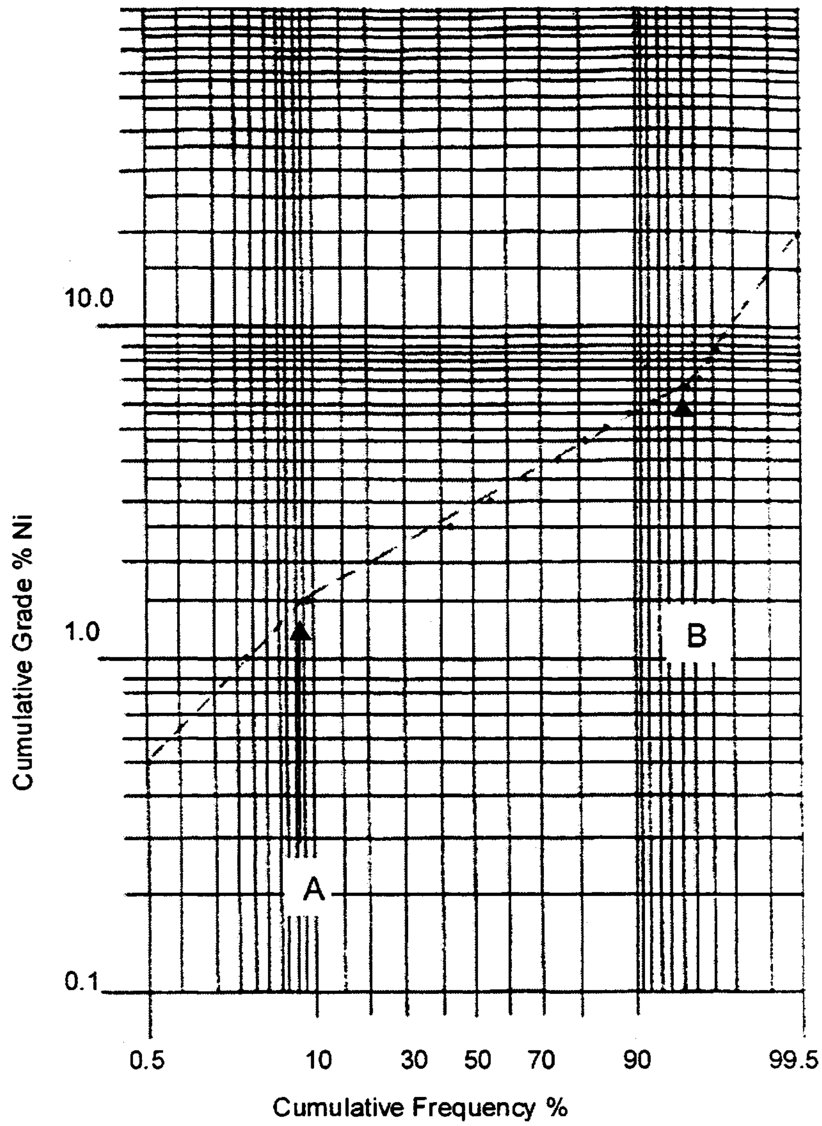


Figure 10 - Lognormal Probability Plot of Raglan Drill Core Nickel Assays

The problem of mixtures of distributions is one encountered in many fields, including mining **[Sichel, 1972]**. The situation arises if a specimen may have derived from one of several populations possessing similar distributions, but perhaps different means and standard deviations. In mining or geology the characteristics of a deposit may be modified by reworking or secondary mineralisation phases. Specimens drawn from separate phases of mineralisation may exhibit different statistical behaviour, but may not be distinguishable (or distinguished) by geological or chemical analysis **[Clark, 1977]**.

Decomposition of the Compound Distribution

With the computer program entitled 'ROKE', Clark prepared appropriate software which would decompose alleged compound distributions into their subordinate distributions **[Clark, 1977]**. The approach relies on the linearisation of the probability plot through the transform of cumulative observed frequency as x scale, with cumulative observed paymetal grade as the y scale (see Figure 5).

ROKE is now available as part of the overall software package called 'TRIPOD', which is marketed by Geostokos Ltd. **[Clark, 2004]**. It uses a modified type of Excel data file, or comma – delimited (CSV) data file. From this software, it is possible to decompose a mixed, or compound, distribution into its component parts, with estimates of subordinate means, frequencies and standard deviations. This software was purchased and used to examine the data pertaining to Figure 10 **[Lotter, 2004]**. The decomposition of the compound distribution shown in Figure 10 by Tripod is summarised in Table 17.

Table 17 - Decomposition of Raglan Drill Core Distribution by TRIPOD

Nickel Data

Mineral Phase	**1**	**2**	**3**
Frequency %	9.12	85.36	5.51
Naperian Transform			
Mean	0.917	1.051	1.803
Standard deviation	0.60	0.486	1.022
Upper Confidence Limit of Subdistribution (95% Level)	2.117	2.023	3.847
Lower Confidence Limit of Subdistribution (95% Level)	-0.283	0.079	-0.241
Exponential Inverse Transform			
Mean % Ni	2.50	2.86	6.07
Upper Confidence Limit of Subdistribution (95% Level) % Ni	8.31	7.56	46.85
Lower Confidence Limit of Subdistribution (95% Level) % Ni	0.75	1.08	0.79

One is tempted to draw the conclusion that, since there are three components to the drill core distribution, these 'must align' with the three End-Members as set out from geological definition. Comparison of the data in Tables 5 and 17 refutes this tempting conclusion. The sampled mean grades in Table 5 (drill-core) do not agree well with the estimated (modelled) mean grades produced by TRIPOD. There are two factors that restrict such a conclusion. One, this is a small data set of 205 observations; two, the distinction between the components may be blurred by sections of drill-core that contain more than one End Member.

4.3. Residual Lognormality in Time

The concentrator operations data for Raglan for the period December 2002-November 2003 have been used for this section of the work. The data are shown in Appendices 1-3, pages 194-206.

Rougher Flotation Feed

In this section of the present work, the distribution characteristics of nickel and copper grades of rougher float feed were studied in a year of continuous operations data from December 2002 to November 2003. The level of information was per twelve-hour shift, producing two observations of grade per day. Exclusions were restricted to days or shifts when the mill was off-line because of breakdown or planned maintenance, or shifts where unusually high grades were measured and inspection of the sampling area determined that the feed sample was contaminated. In such cases, two sequential shifts were disqualified because the operations staff had noted a leak in a concentrate slurry pipe, causing contamination of the rougher float feed stream and thus the rougher float feed samples for those two shifts. All other data were accepted. A total of n=665 data points was gathered in this manner. The data are shown in Appendix 1. This is a suitable number of observations for assessment of the distribution characteristics. It will be demonstrated that despite the blending that results from mining, storage and comminution unit processes, the flotation feed demonstrates clear multimodal residual lognormality.

The data were downloaded from operations files into Excel, and represent automatic sampling of this stream with acid dissolution/atomic adsorption spectrometer measurement of nickel and copper. The method of construction of a histogram, or frequency distribution, of data, is described adequately elsewhere **[Box et al, 1978, pp. 24-28]**. This produced the untransformed, or natural distributions of nickel and copper. The same data were transformed into the Naperian lognormal domain by the Naperian transform (Eq. 21). The lognormal distribution was then plotted using the same procedure as before. The untransformed and transformed data sets each produced a histogram if nickel values sorted by nickel grade bins.

$$z = Ln(x) \qquad [21]$$

A control graph in the form proposed by **Krige, 1962**, was constructed. This took the form of a lognormal probability plot. The control limits were calculated at the 95% confidence level. A 99-day segment of the year's data was selected and used for demonstration purposes. A comparison of parameters was made between the original untransformed data, and their equivalent in the Naperian log transform. A lognormal probability plot, using the cumulative values of nickel, was made from these data, and referring to the method of **Krige, 1981**. In this plot, the cumulative paymetal grade is plotted on the vertical (y) axis in logarithmic scale, opposite the correspondent x values of cumulative probability. The x scale is limited to a range of 0.5 to 99.5%. From the method proposed by **Clark and Garnett, 1974**, the plot was used to semi-quantitatively decompose the distribution into the three components.

The raw data from the 665 observations of Rougher Float Feed grade at the Raglan operations, from November 2002 to November 2003, are shown in Appendix 1. The untransformed distributions are shown in Tables 18 and 19, and in Figures 5 and 6.

Table 18 - Distribution of Nickel in Raglan Rougher Flotation Feed
December 2002-November 2003

Grade Bin, % Ni	Frequency %	Cum. Frequency %	Grade Bin, % Ni	Frequency %	Cum. Frequency %
2.6	0.60	0.60	4.8	1.95	96.69
2.8	1.50	2.11	5.0	1.35	98.05
3.0	5.86	7.97	5.2	0.75	98.80
3.2	7.07	15.04	5.4	0.45	99.25
3.4	16.09	31.13	5.6	0.15	99.40
3.6	14.29	45.41	5.8	0.45	99.85
3.8	18.50	63.91	6.0	0	99.85
4.0	12.33	76.24	6.2	0	99.85
4.2	9.02	85.26	6.4	0	99.85
4.4	6.47	91.73	6.6	0.15	100.00
4.6	3.01	94.74			

Table 19 - Distribution of Copper in Raglan Rougher Flotation Feed
December 2002-November 2003

Grade Bin, % Ni	Frequency %	Cum. Frequency %	Grade Bin, % Ni	Frequency %	Cum. Frequency %
0.7	1.65	1.65	1.3	4.66	95.19
0.8	8.12	9.77	1.4	2.56	97.74
0.9	23.01	32.78	1.5	1.20	98.95
1.0	27.07	59.85	1.6	0.90	99.85
1.1	18.95	78.80	1.7	0.15	100.00
1.2	11.73	90.53			

The relevant parameters for the untransformed and transformed distributions are shown in Table 20.

Table 20 - Distribution Parameters : Raglan Rougher Flotation Feed
November 2002-2003

Parameter	Nickel	Copper
Untransformed Data		
Sample Mean	3.49	0.988
Sample Standard Deviation	0.525	0.166
N	665	665
Upper Confidence Limit (95%)	4.54	1.320
Lower Confidence Limit (95%)	2.44	0.656
Transformed Data		
Sample Mean	3.45	0.975
Sample Standard Deviation	1.157	1.177
N	665	665
Upper Confidence Limit (95%)	4.62	1.345
Lower Confidence Limit (95%)	2.58	0.704

The arithmetic sample mean values agree with their uncorrected geometric equivalents (compare 3.49 % Ni with 3.45% Ni and 0.988 % Cu with 0.975% Cu). In other words, the arithmetic mean has little noise. This is because of the low level of logvariance in these data. The geomean (or uncorrected geometric mean) is slightly lower than the arithmetic mean. The Sichel correction of the geomean (t = 3.493) is successful. The remaining difference that should be noted is that the associated confidence limits from the lognormal base are skewed about the mean. These limits are appropriate because of the residual lognormality.

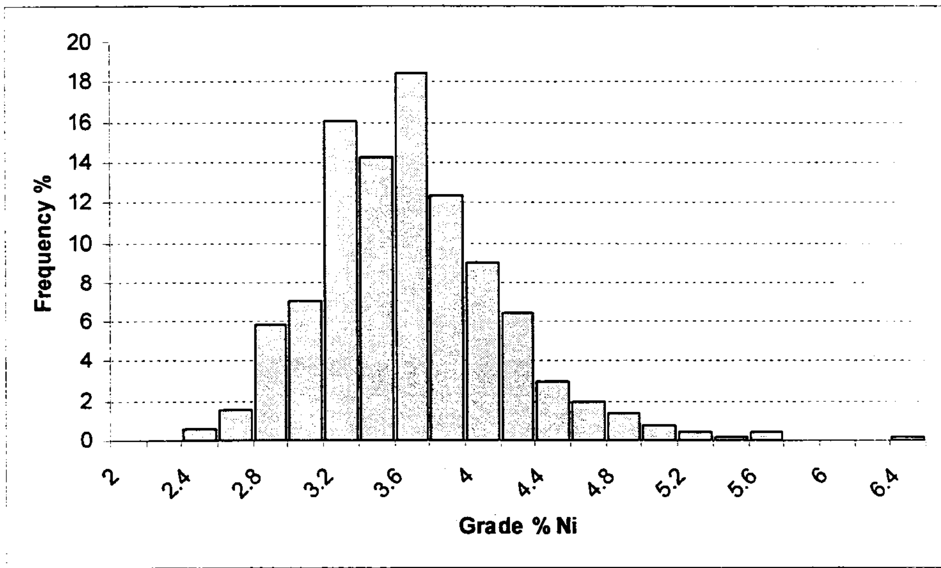


Figure 11 - Untransformed Nickel Values in Raglan Flotation Feed

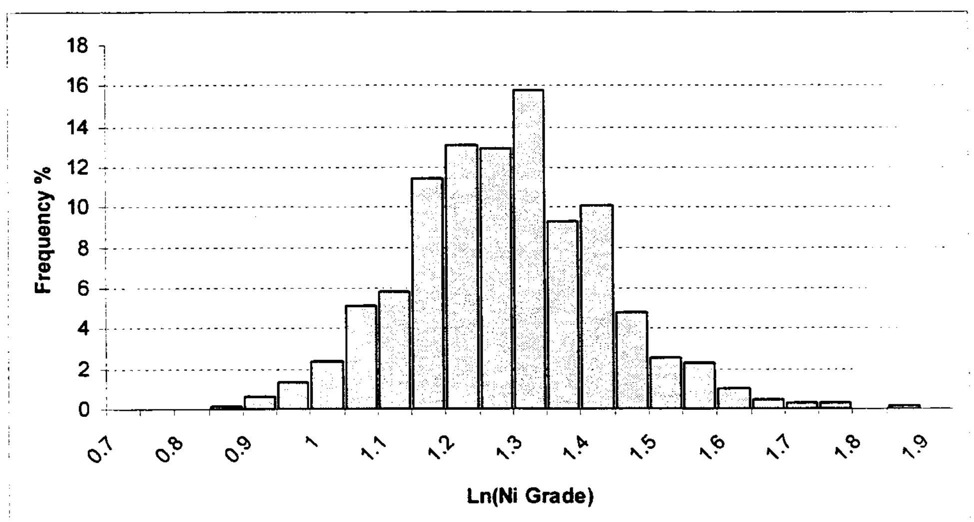


Figure 12 - Transformed Nickel Values in Raglan Flotation Feed

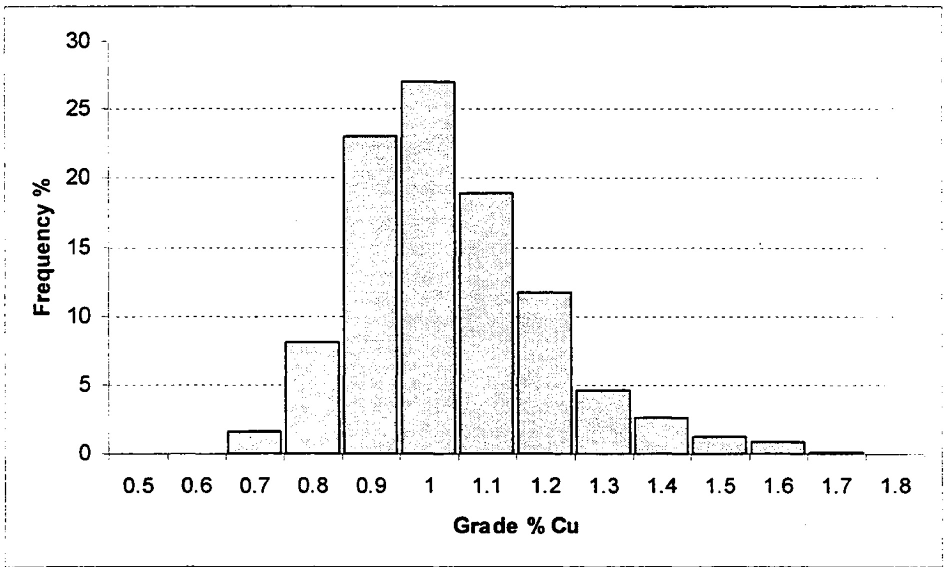


Figure 13 - Untransformed Copper Values in Raglan Rougher Flotation Feed

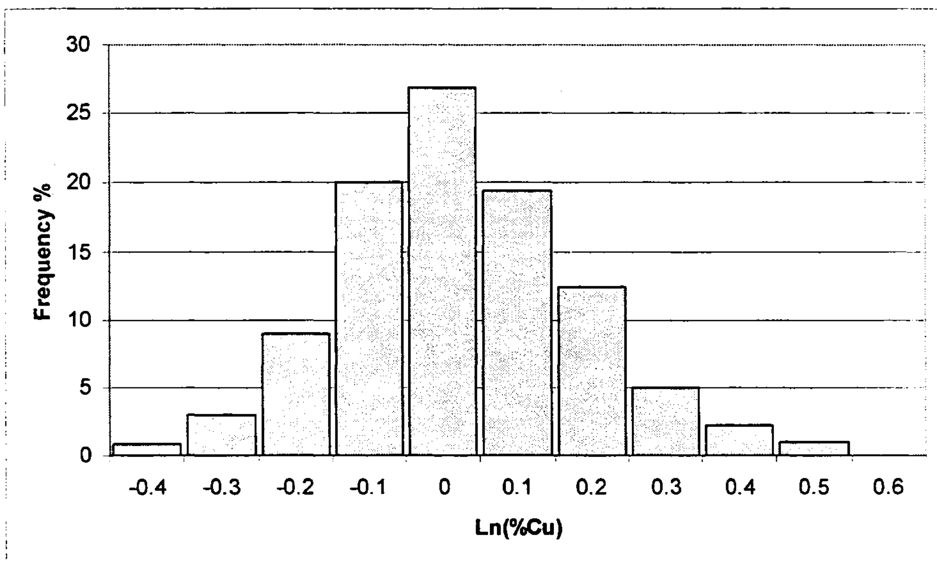


Figure 14 - Transformed Copper Values in Raglan Rougher Float Feed

Probability Plot of Nickel in Raglan Rougher Float Feed

Inspection of Figures 11 and 12 (Nickel) and 13 and 14 (Copper) shows that the Naperian log transform has evened out the distribution of values to a more symmetrical arrangement (as was proposed by **Krige, 1951a,b**). A probability plot of the untransformed cumulative values on lognormal probability paper is shown in Figure 15.

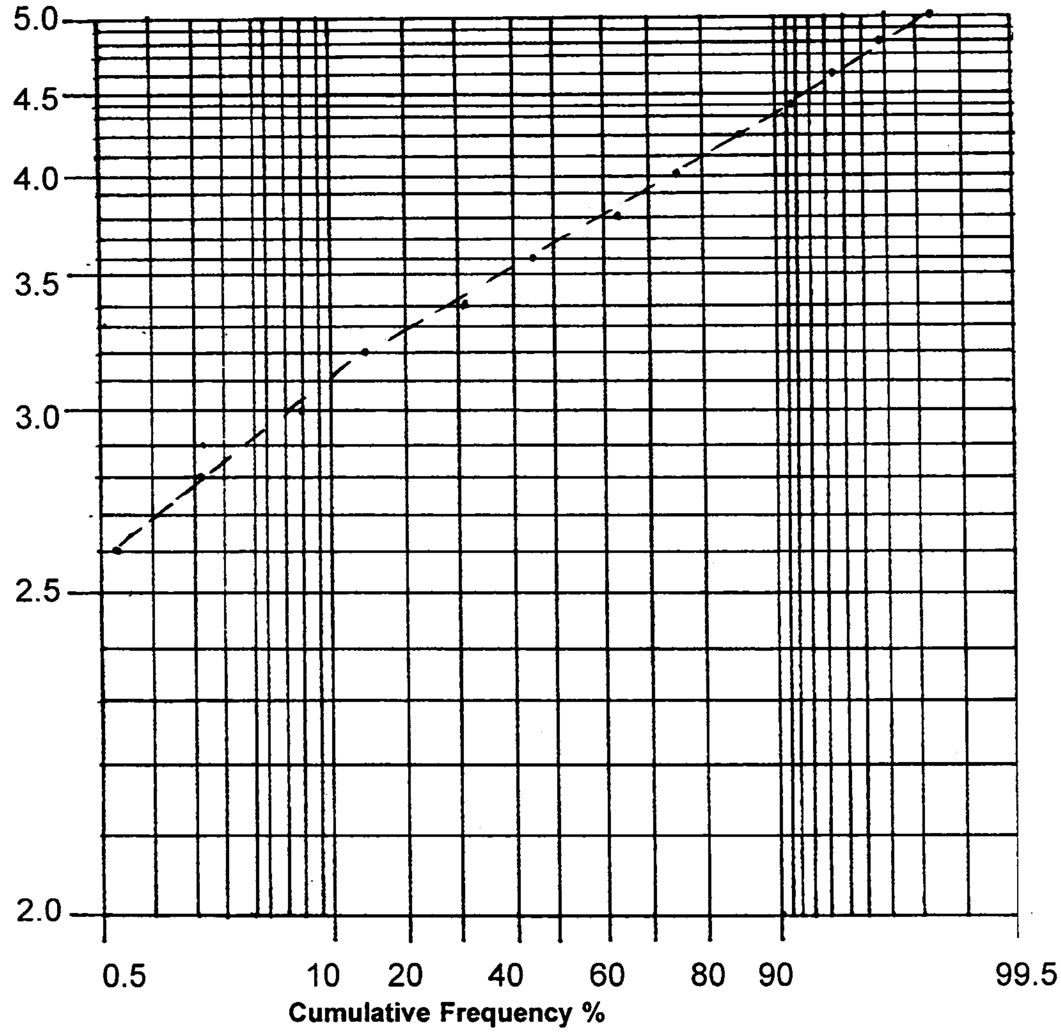


Figure 15 - Lognormal Probability Plot of Nickel in Raglan Rougher Float Feed Data from a 99-Day Segment of the Period Nov 2002/December 2003.

Inspection of Figure 15 shows a three-phase plot. This shows that the distribution is three-component, two parameter lognormal. The ' low-grade' component, to the left of 15% frequency, is clear, whereas the 'high-grade' component, to the right of 91% frequency, is almost undetectable. Functionally it is more practical to regard this distribution as two-component rather than three-component. Decomposition of this distribution by TRIPOD software, as described previously for the drill-core example, produced the following results. The two component means are very close. Neither of the two subdistribution means seem to correspond to the sampled grades of any of the End-Members.

Table 21 - Decomposition of Raglan Rougher Float Feed by TRIPOD
Nickel Data

Component	**1**	**2**
Frequency %	13.06	86.94
Naperian Transform		
Mean	1.229	1.240
Standard deviation	0.135	0.151
Upper Confidence Limit (95% Level)	1.499	1.542
Lower Confidence Limit (95% Level)	0.959	0.938
Arithmetic Inverse Transform		
Mean % Ni	3.42	3.46
Upper Confidence Limit (95% Level) % Ni	4.47	4.67
Lower Confidence Limit (95% Level) % Ni	2.61	2.56

Final Concentrate

The same production period, December 2002 - November 2003, was used as for Rougher Flotation Feed, using again the same rules of exclusion, *viz.* mill being off-line for maintenance, etc. The raw data are shown in the Appendix, and are summarised in Tables 22 and 23.

Table 22 - Raglan Final Concentrate
Summary of Nickel Distribution

Grade Bin % Ni	**Frequency %**	**Grade Bin % Ni**	**Frequency %**
15	0.00	20.5	14.48
15.5	0.00	21	9.10
16	0.15	21.5	4.93
16.5	0.15	22.0	3.28
17	1.79	22.5	1.04
17.5	1.64	23.0	0.75
18	5.97	23.5	0.15
18.5	11.79	24.0	0.00
19	11.64	24.5	0.00
19.5	17.91	25.0	0.00
20	15.22	25.5	0.00

Table 23 - Raglan Final Concentrate

Summary of Copper Distribution

Grade Bin % Cu	Frequency %	Grade Bin % Cu	Frequency %
3	0.00	6	21.49
3.5	0.15	6.5	9.25
4	0.30	7	4.48
4.5	4.03	7.5	0.60
5	22.99	8	0.30
5.5	36.42		

Table 24 - Summary of Distribution Parameters : Raglan Final Concentrate

December 2002-November 2003

Parameter	Nickel	Copper
Untransformed Data		
Sample Mean	19.52	5.37
Sample Standard Deviation	1.19	0.60
N	670	670
Upper Confidence Limit (95%)	21.90	6.58
Lower Confidence Limit (95%)	17.14	4.17
Transformed Data		
Sample Mean	19.48	5.34
Sample Standard Deviation	3.29	1.12
N	670	670
Upper Confidence Limit (95%)	22.02	6.65
Lower Confidence Limit (95%)	17.25	4.29

The arithmetic sample mean values agree with their uncorrected geometric equivalents (compare 19.52 % Ni with 19.48 % Ni and 5.37 % Cu with 5.34 % Cu). This agreement reflects the lower logvariance of the concentrate grade. As before with the case of Rougher Float Feed, the control limits differ but this difference is negligible. These data will be treated as a Normal Distribution.

Table 25 - Distribution of Nickel in Raglan Final Concentrate
December 2002-November 2003

Grade Bin, % Ni	Frequency %	Cum. Frequency %	Grade Bin, % Ni	Frequency %	Cum. Frequency %
16	0.15	0.15	20	15.22	66.27
16.5	0.15	0.30	20.5	14.48	80.75
17	1.79	2.09	21	9.10	89.85
17.5	1.64	3.73	21.5	4.93	94.78
18	5.97	9.70	22.0	3.28	98.06
18.5	11.79	21.49	22.5	1.04	99.10
19	11.64	33.13	23.0	0.75	99.85
19.5	17.91	51.04	23.5	0.15	100.0

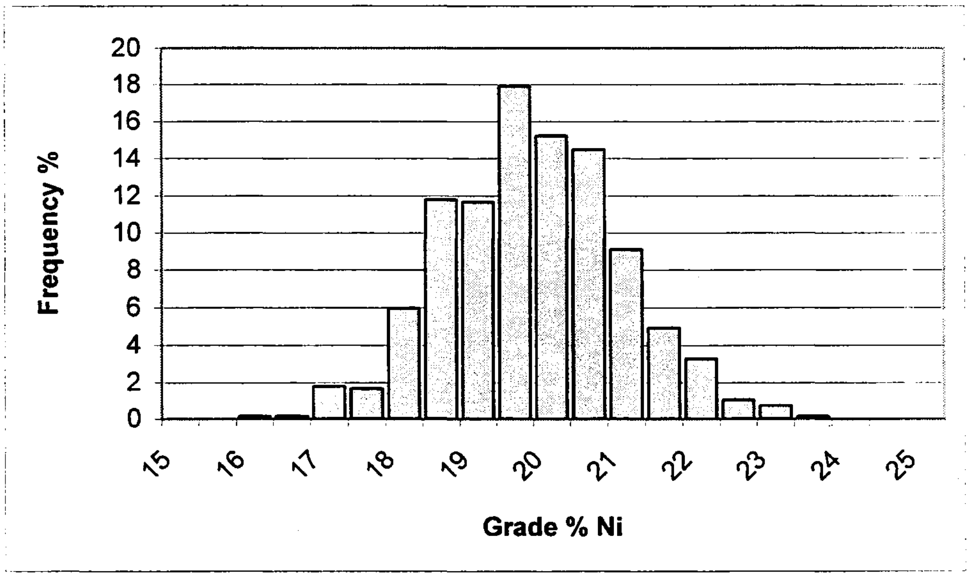


Figure 16 - Distribution of Nickel Values in Final Concentrate
(Raglan)

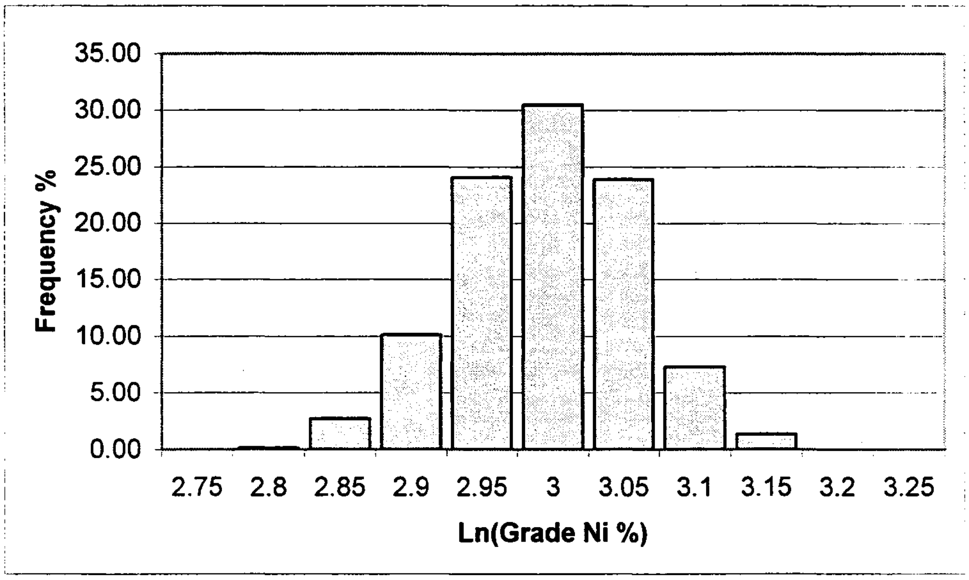


Figure 17 - Distribution of Transformed Nickel Values in Final Concentrate (Raglan)

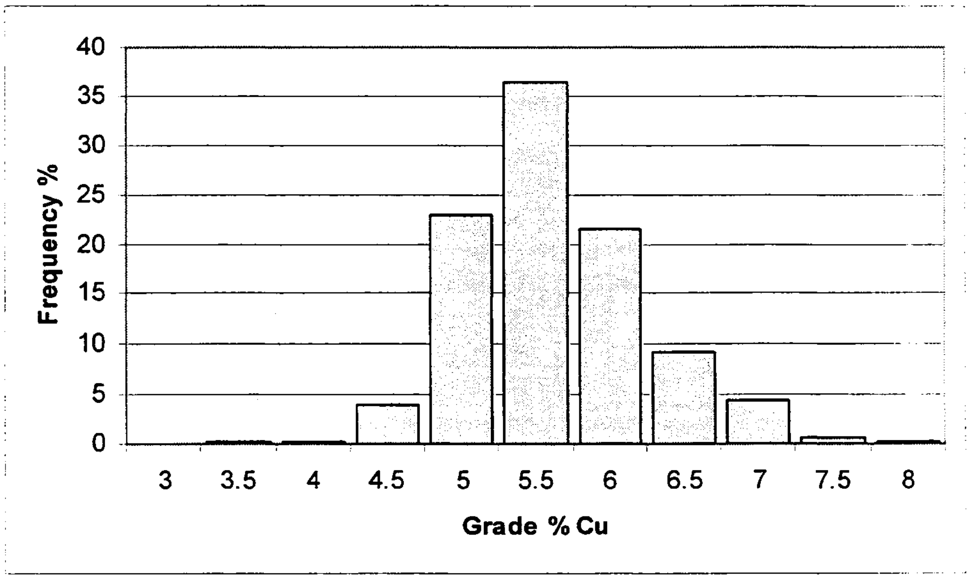


Figure 18 - Distribution of Copper Values in Final Concentrate (Raglan)

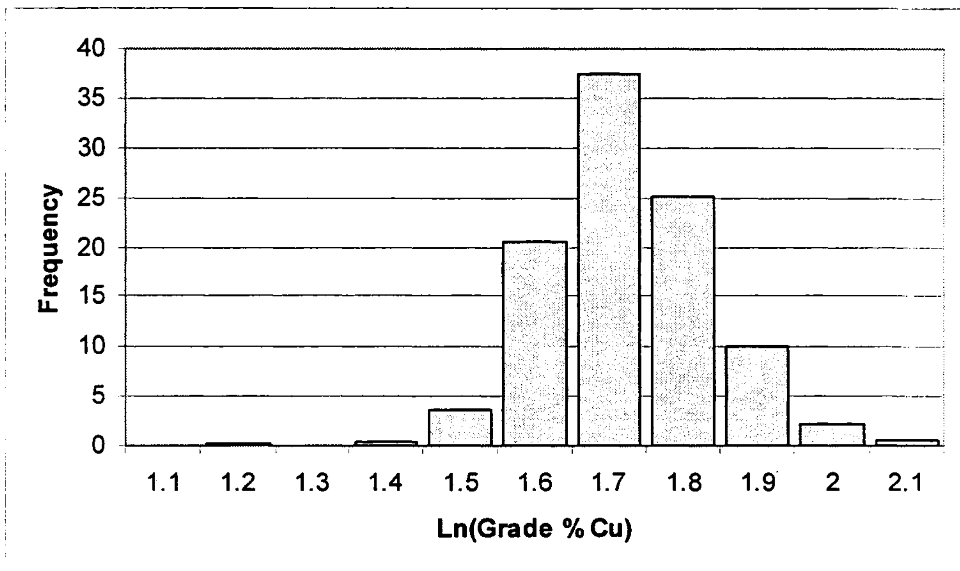


Figure 19 - Distribution of Transformed Copper Values in Final Concentrate (Raglan)

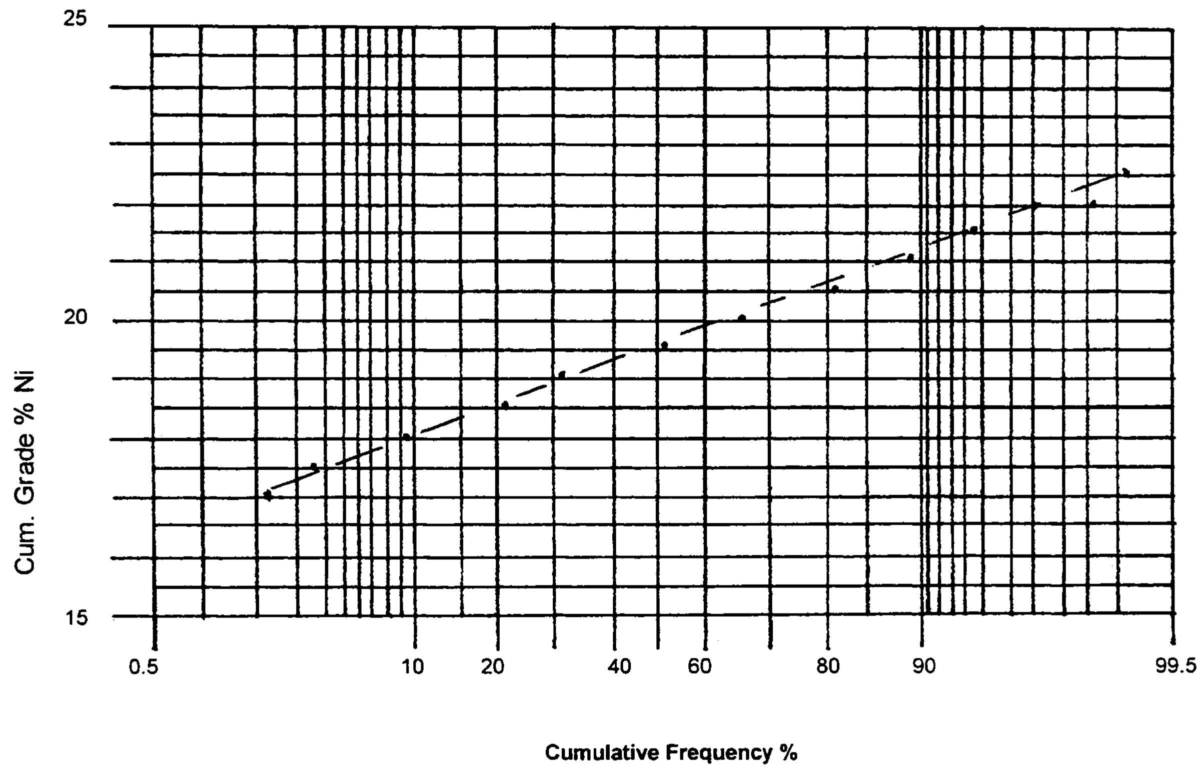


Figure 20 - Normal Probability Plot of Nickel Values in Final Concentrate (Raglan)

Figure 20 suggests that the nickel values in the time dimension align with a symmetrical, or normal, distribution **[Clark, 1993a]**. This finding is a probable outcome of the engineering process by which the concentrate is produced. In other words, the processes of liberation (grinding) and selective flotation (separation) have extracted principally most of the sulphide nickel mineral hosts into a concentrate which is tightly controlled in terms of a range of assayed nickel grade. The concentrate, therefore, carries mostly nickel in mineral hosts that are in a similar range of nickel content.

Final Tailings

Untransformed and Transformed Distributions of Nickel and Copper in Raglan Final Tailings

The same production period, December 2002 - November 2003, was used as for Final Tailings, using again the same rules of exclusion, viz. mill being off-line for maintenance, etc. The raw data are shown in the Appendix, and are summarised in tables 26 and 27.

Table 26 - Raglan Final Tailings
Summary of Nickel Distribution

Grade Bin % Ni	Frequency %	Grade Bin % Ni	Frequency %
0.1	0.00	0.55	7.90
0.15	0.00	0.6	3.43
0.2	0.15	0.65	1.64
0.25	0.30	0.7	1.19
0.3	4.02	0.75	1.04
0.35	17.59	0.8	1.04
0.4	30.55	0.85	0.45
0.45	18.63	0.9	0.15
0.5	11.62	0.95	0.30

Table 27 - Raglan Final Tailings
Summary of Copper Distribution

Grade Bin % Cu	Frequency %	Grade Bin % Cu	Frequency %
0	0.00	0.25	8.06
0.05	0.00	0.3	1.94
0.1	1.19	0.35	1.04
0.15	40.90	0.4	0.60
0.2	46.12	0.45	0.15

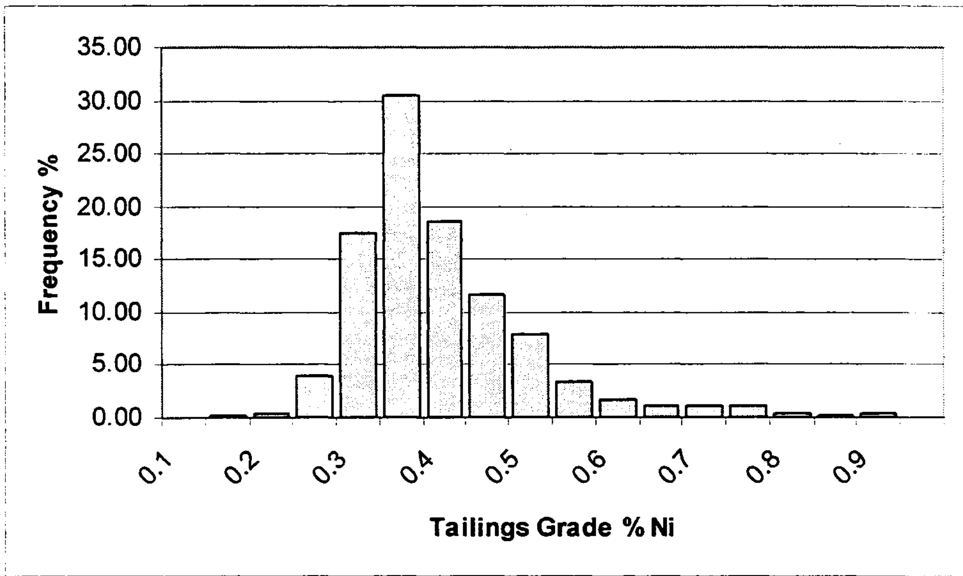


Figure 21 - Distribution of Untransformed Nickel Grades in Final Tailings (Raglan)

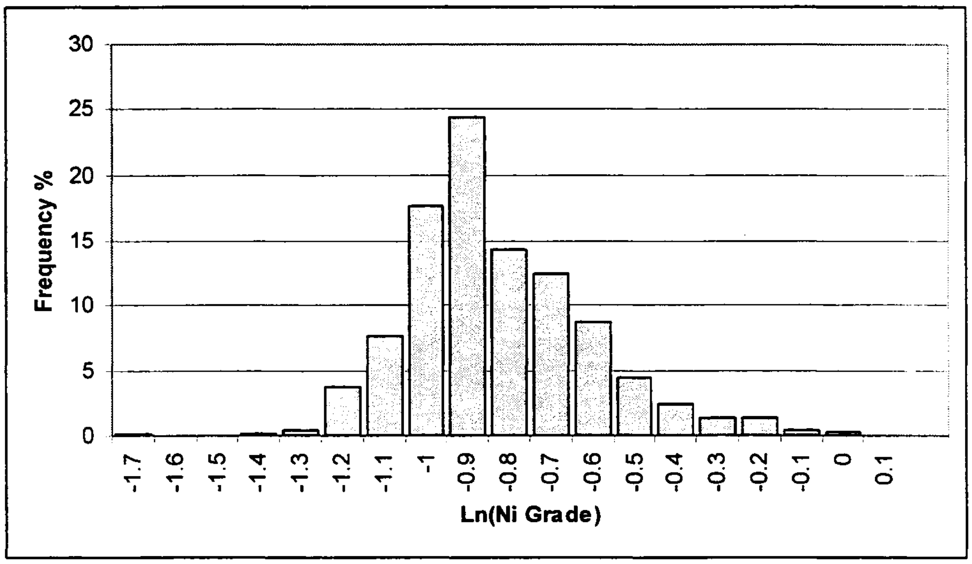


Figure 22 - Distribution of Transformed Nickel Grades in Final Tailings (Raglan)

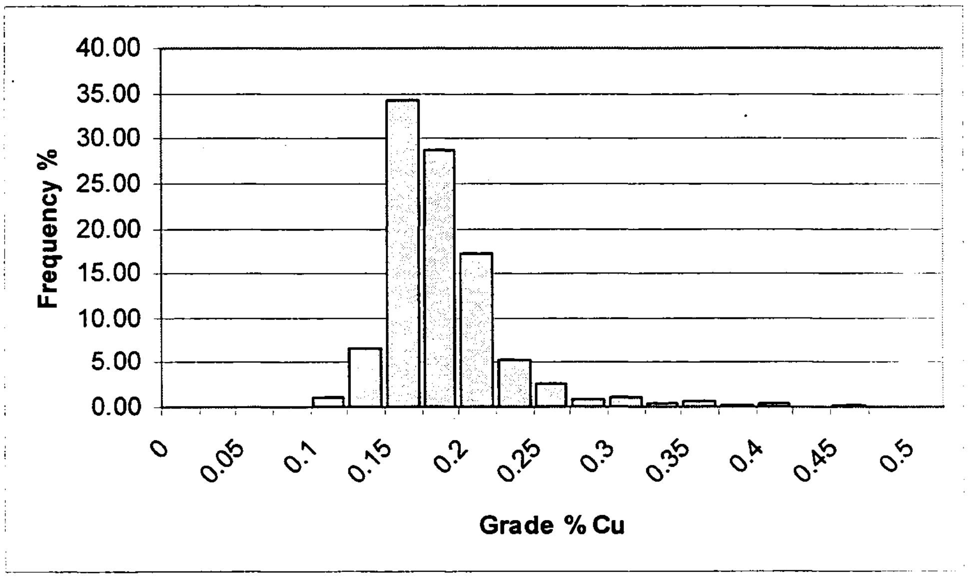


Figure 23 - Distribution of Untransformed Copper Grades in Final Tailings (Raglan)

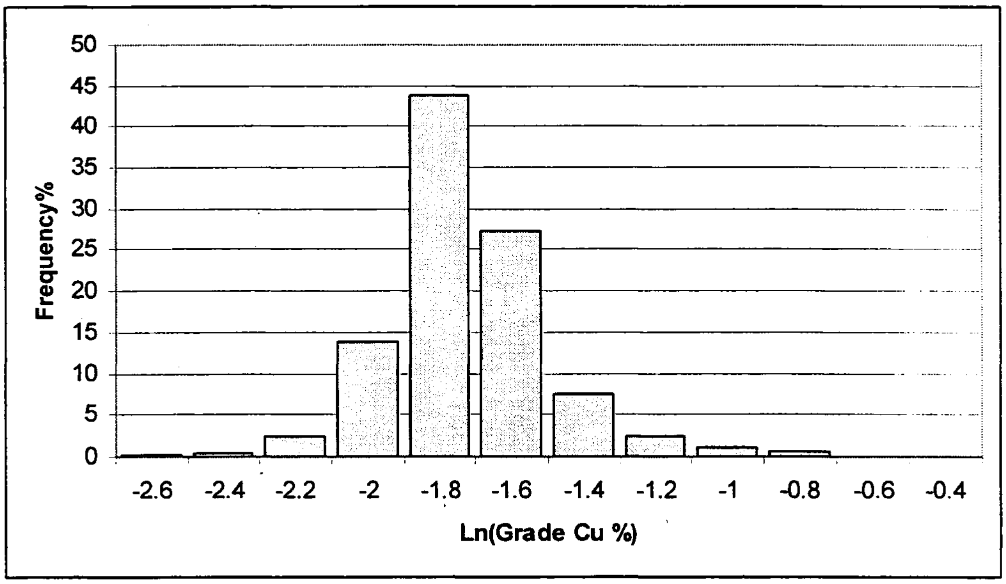


Figure 24 - Distribution of Transformed Copper Grades in Final Tailings (Raglan)

Table 28 - Summary of Distribution Parameters : Raglan Final Tailings December 2002-November 2003

Parameter	Nickel	Copper
Untransformed Data		
Sample Mean	0.425	0.166
Sample Standard Deviation	0.103	
N	671	671
Upper Confidence Limit (95%)	0.632	0.245
Lower Confidence Limit (95%)	0.218	0.087
Transformed Data		
Sample Mean	0.418	0.161
Sample Standard Deviation		
N	671	671
Upper Confidence Limit (95%)	0.643	0.276
Lower Confidence Limit (95%)	0.268	0.094

The arithmetic sample mean values agree with their uncorrected geometric equivalents (compare 0.425 % Ni with 0.418 % Ni and 0.166 % Cu with 0.161 % Cu). The value of Sichel's t-estimator for the nickel grades in these tailings data (separately calculated) was t = 0.425% Ni. The link between the arithmetic mean, the uncorrected geomean, and the corrected geomean follows the discussion of Table 20.

Probability Plot of Nickel in Final Tailings

Table 29 - Distribution of Nickel in Raglan Final Tailings

November 2002-November 2003

Grade Bin, % Ni	Frequency %	Cum. Frequency %	Grade Bin, % Ni	Frequency %	Cum. Frequency %
0.2	0.15	0.15	0.6	3.43	94.19
0.25	0.30	0.45	0.65	1.64	95.83
0.3	4.02	4.47	0.7	1.19	97.02
0.35	17.59	22.06	0.75	1.04	98.06
0.4	30.55	52.61	0.8	1.04	99.11
0.45	18.63	71.24	0.85	0.45	99.55
0.5	11.62	82.86	0.9	0.15	99.70
0.55	7.90	90.76	0.95	0.30	100.00

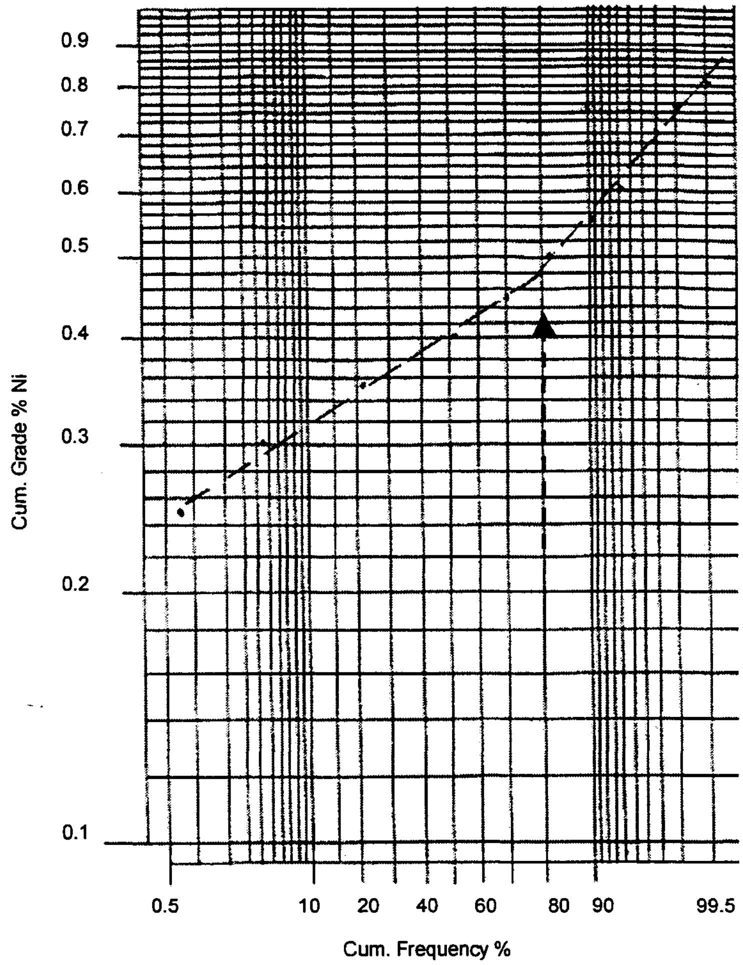


Figure 25 - Lognormal Probability Plot of Nickel Values In Final Tailings (Raglan)

The probability plot for Raglan Final Tailings suggests an alignment with lognormality, with two main component phases present **[Clark, 1993a]**. In other words, this is a bimodal distribution. The point of division between the two subdistributions is at approximately 80% of the composite distribution. This will now be explored. The existence of two phases in a compound (bimodal) distribution in the Final Tailings suggests that either nature (*e.g.* End-Members) or nurture (*e.g.* processing or treatment), or both, are contributing to this situation.

In terms of nature, or End-Members, reference to the pentlandite grain sizes in Table 7 suggests that the disseminated sulphides have the finest pentlandite grain sizes of all three End-Members. Imperfections in the grinding and classification processes treating the ore, which is a variable mixture of these End-Members, produce a range of liberation classes of pentlandite. Each of these would respond differently to the flotation process (with the liberated pentlandite floating more efficiently than the unliberated pentlandite). This would yield a segment of pentlandite losses in the Final Tailings which are incompletely liberated. This has been confirmed by Qem*SCAN measurements made of Raglan tailings samples, in which 70-75% of the nickel iron sulphides present were locked or middling particles **[Lotter et.al., 2002]**

In terms of nurture, imperfections or variation in key unit processes would result in recoverable losses of liberated pentlandite. Examples are: the grinding circuit, (which affects size distribution), control of the flotation process, or improper level or air control in the flotation cells, or inadequate reagent dosage. Again, Qem*SCAN measurements of Raglan tailings show liberated pentlandite to be present [**Lotter *et. al.*, 2002**].

This End-Member and processing perspective might consolidate into a situation where the paymetal values are distributed between two distinct phases. One might be the unrecoverable form, hosted in silicates either as finely disseminated unliberated pentlandite, a feature which has become well-known as the 'textural dominance' of pentlandite in the Raglan ore. Also, solid solution nickel in altered ferromagnesium silicates such as serpentine is known. The other might be recoverable, such as ultrafine liberated pentlandite, or nickel either locked as pentlandite flames in slow-floating pyrrhotite. The latter group is known to be much higher in nickel grade than the former, this would explain the bimodality. Put differently, whatever weight ratio and mineral composition exists in these two components, the overall mixture is lognormally bimodal because of the mineral hosting carrying nickel in different orders of magnitude. The overall tailing seems to be a blend of these various components, with a strong emphasis on the first component. The reader is referred to Table 2: Raglan Primary Composition File.

The Final Tailings distribution was decomposed into its subordinate distributions using Tripod **[Clark, 2004; Clark, 1977]**. The following decomposition resulted after 8 iterations of the non-linear least squares decomposition, after which TRIPOD diagnostics indicated that there was no further optimisation possible:

Table 30 - Decomposition of Raglan Final Tailings by TRIPOD

Mineral Phase	1	2
Frequency %	82.76	17.24
Naperian Transform		
Mean	-0.944	-0.623
Standard deviation	0.170	0.201
Upper Confidence Limit (95% Level)	-0.605	-0.221
Lower Confidence Limit (95% Level)	-1.283	-1.025
Exponential Inverse Transform		
Mean % Ni	0.389	0.536
Upper Confidence Limit (95% Level) % Ni	0.55	0.80
Lower Confidence Limit (95% Level) % Ni	0.28	0.36

The above diagnostics were used together with linear extensions of the two visible lines in Figure 15 as a guide, to model the subordinate distributions. These subordinate distributions are shown in the following figures 26 and 27:

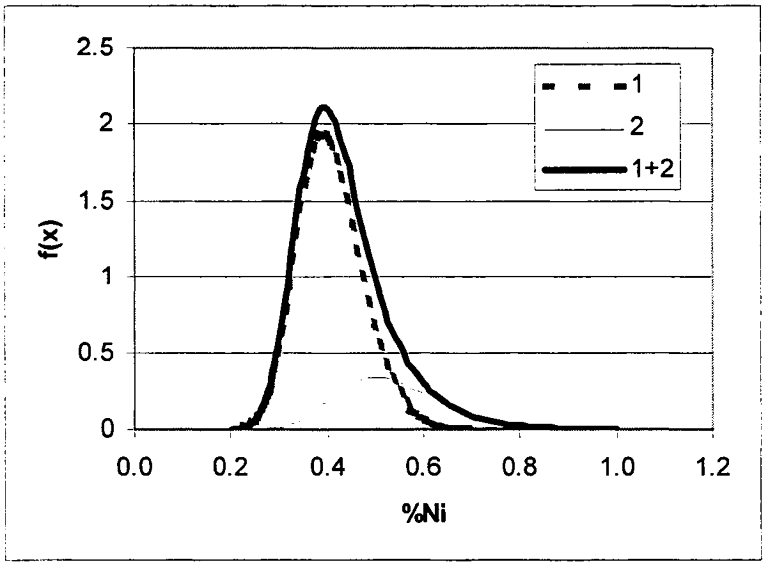


Figure 26 - Compound (1+2) and Subordinate Distributions of Tailings
Phase 1 (80% weight) and Phase 2 (20% weight) in Raglan Final Tailings

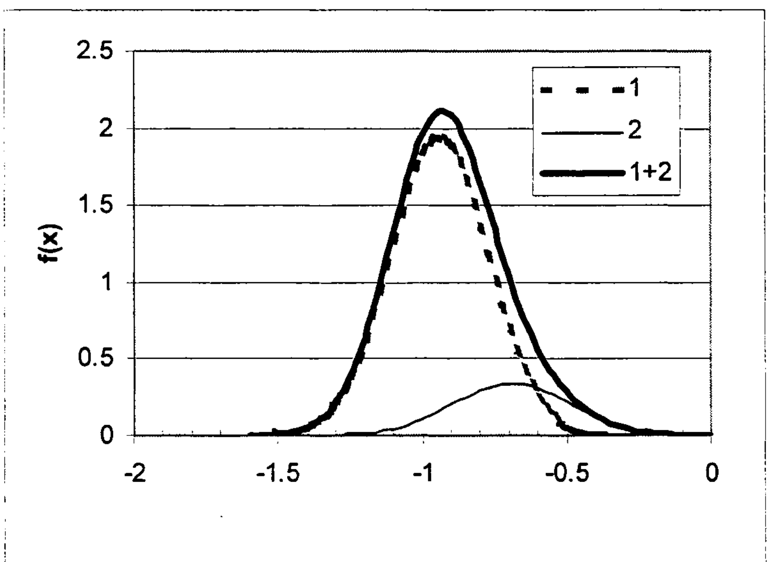


Figure 27 - Compound (1+2) and Subordinate Distributions of Tailings
Phase 1 (80% weight) and Phase 2 (20% weight) in Raglan Final Tailings – Logarithmic Transform

It is obvious from the above decomposition and from the Tripod diagnostics that the minor distribution (Phase 2) has a higher variance and range of data than Phase 1. This is consistent with discussion on the general interpretation of lognormal probability plots **[Rendu, 1981]**. What is also obvious is that the minor Phase 2 carries the higher-grade nickel values. As a cross-check, using some visible and intuitive points in Figure 25, visual estimates may be made of the decomposition of this distribution. Assuming that the 80/20% weight of the two subdistributions is correct, and that the centre of the composite distribution carries too much overlap of the subdistributions to allow visual use of that domain, the two subdistribution means may be estimated as:

Subdistribution 1 : 80% Weight

Using the range 0-40% cumulative frequency, we may ignore the effects of overlap from the minor (high-grade) distribution. Taking the intercept on the overall plot at 40% cumulative frequency, and dividing by 0.8, we obtain 50%, which would represent the mean of this subdistribution. The estimated grade (y-scale) corresponding to this point is 0.385% Ni.

Subdistribution 2 : 20% Weight

Using the 90% + cumulative frequency, we may ignore the effects of overlap from the major (low-grade) distribution. Taking the intercept on the overall plot at 90% frequency (which for the high-grade distribution is a frequency of 10%), and dividing that frequency by the weight of 20%, i.e. 0.2, we obtain 50%, which would represent the mean of this subdistribution. The estimated grade (y-scale) corresponding to this point is 0.55% Ni.

Comparison of the means of this simplified decomposition with the nonlinear least squares method of TRIPOD (Table 30) shows a good agreement. Taking the simplified decomposition results into an interpretation of probable recoverable values in the final tailings, simple proportion suggests that (0.8*0.385) +(0.2*0.55) = 0.418 (which is the observed overall distribution mean). If we assume that the second (minor) phase of 20% can be eliminated by better 'nurture', or process operation, then the final tailings grade would be 0.385% Ni. The decrease in tailings loss, assuming the same weight recovery to tailings, would equal 10.19% x ((0.418-0.385)/0.418) = 0.80% recovery.

Cross-Check on the Interpretation of the Bimodality of the Tailings Distribution

The bimodality of the Raglan Final Tailings described above in Figure 25, if correct, could be of significant generic diagnostic value. For this reason, an extra exercise was conducted for the Raglan operations to verify by independent means that the upper subdistribution correlated with some particular mineral(s) that were paymetal-bearing and could be recoverable. Since the Raglan concentrator was surveyed in June 1998 and the samples were measured by Qem*SCAN at Falconbridge, actual mineral measurement data are available for the Final Tailings sample for that survey. A summary of the actual minerals measured, using the same primary composition file as described earlier in this text, follows.

Table 31 - Bulk Modal Analysis of Raglan Final Tailings : Survey June 1998
QEM*SEM Measurements (Bulk Modal Analysis, or BMA)

Mineral Type	**Mineral/Texture**	**Quantity in Raglan Rougher Final Tailings %**
Sulphides	Pentlandite	0.78
	Violarite	0.06
	Nickel Sulphides/Arsenides	0.00
	Pyrite	0.15
	Pyrrhotite	6.63
	Chalcopyrite	0.22
	Other Sulphides	0.04
Sulphide-Silicate Textures	Serpentine-Chalcopyrite	0.94
	Serpentine-Pentlandite	1.03
	Serpentine-Pyrrhotite	2.03
Silicates/Oxides	Serpentine	54.0
	Pyroxenes	0.05
	Amphibole	16.87
	Talc/Chlorite	3.74
	Other Silicates	7.07
	Carbonates	1.69
	Magnetite/Ilmenite/Chert	3.63
	Other	1.07

Liberation analysis of the above data by Qem*SCAN using the Particle Measurement Analysis, or PMA, provided the following analysis.

Table 32 - Liberation Analysis of Raglan Final Tailings : Survey June 1998

QEM*SEM Measurements (Liberation Analysis, or PMA) : Nickel Distribution

Mineral/Texture	Quantity in Raglan Rougher Final Tailings %
Liberated Nickel-Iron Sulphides	**10.64**
Middling Nickel-Iron Sulphides	**14.97**
Locked Nickel-Iron Sulphides	45.56
Pyrrhotite	6.24
Serpentine	19.20
Magnetite/Ilmenite/Chert	3.26
Other gangue	0.14

The reader is referred to the *primary composition file* of the Raglan system (Table 3). The '*Nickel-Iron-Sulphides*' group (Ni-Fe-S) is collectively pentlandite, violarite, nickel arsenides, *etc*., which are to be found in the 10-100% order of magnitude in nickel content; the remainder, mostly silicates and sulphide gangue, are to be found in the 0.1-1.0 % order of magnitude in nickel content. Liberated Ni-Fe-S are defined in Qem*SCAN as those Ni-Fe-S particles of any size which contain more than 90% Ni-Fe-S. Middling Ni-Fe-S are defined as those particles containing between 30 and 90% Ni-Fe-S. Locked Ni-Fe-S are defined as those particles containing between slightly more than zero but less than 30% NiFeS. In other words, this means that the tailings stream of the Raglan Concentrator contains a mixture of nickel mineral hosts, ranging from pure pentlandite (at 34% Ni) to worthless serpentine (at 0.1% Ni). The mineral mixture being described in Table 32 quantitatively represents the nickel mineral hosts in the Raglan Final Tailings. This mixture contains some very high grade nickel species, such as liberated pentlandite. It also contains some silicate particles which contain only nickel in solid solution.

Tables 31 and 32 present clear pictures of the average distribution of nickel losses in tailings. The first two classes, *viz*. liberated and middling, should be recoverable. It is not clear how this distribution changes between the two modes in Figure 25. Further weight would be provided if similar QEM*SEM data were available for the two modes.

If nurture is the major cause of the high tailings end mode, then its recoverable nickel component should be higher than that of the lower tailing grade mode. If nature is the major cause, then the opposite should be observed[1].

In the absence of more specific modal analysis, the role of nurture can be further investigated by comparing the data of Figure 25 to earlier data from 1998, corresponding to the less efficient flowsheet used at commissioning.

A distribution of nickel grades was constructed from a continuous three-month data base for May, June and July 1998 operations. The distribution of nickel grades is shown in Table 32. The purpose of this exercise is to demonstrate that the cumulative logarithmic probability plot of nickel grades in Final Tailings at the Raglan Concentrator can be used as a semi-quantitative estimator, or benchmark, of the recoverable nickel grade(s) being lost to those Final Tailings. Specifically, across a span of several years of operations during which several flowsheet improvements have been made, and which have reduced that amount of recoverable nickel in Final Tailings.

Table 33 - Distribution of Nickel in Raglan Final Tailings: May-July 1998

Grade Bin, % Ni	Frequency %	Cum. Frequency %	Grade Bin, % Ni	Frequency %	Cum. Frequency %
0.2	0	0	0.6	6.5	71.4
0.25	0	0	0.65	1.8	76.8
0.3	0.6	0.6	0.7	1.2	82.1
0.35	6.0	6.5	0.75	1.2	88.7
0.4	8.9	15.5	0.8	2.4	90.5
0.45	14.9	30.4	0.85	0.6	91.7
0.5	23.2	53.6	0.9	6.5	92.9
0.55	17.9	71.4	0.95	1.8	95.2

[1] Unfortunately at the time of writing, such information is not available, but such an exercise has been planned for 2005/6.

This distribution is shown in Figure 28. The probability plot of this distribution, formatted in Lognormal transform, is shown in Figure 29.

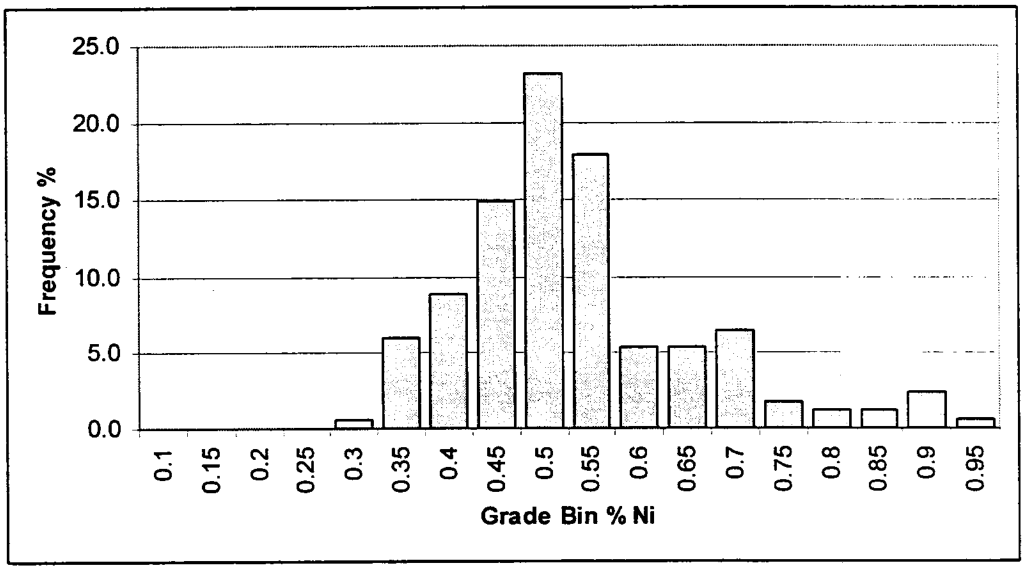


Figure 28 - Distribution of Raglan Final Tailings Nickel Grades

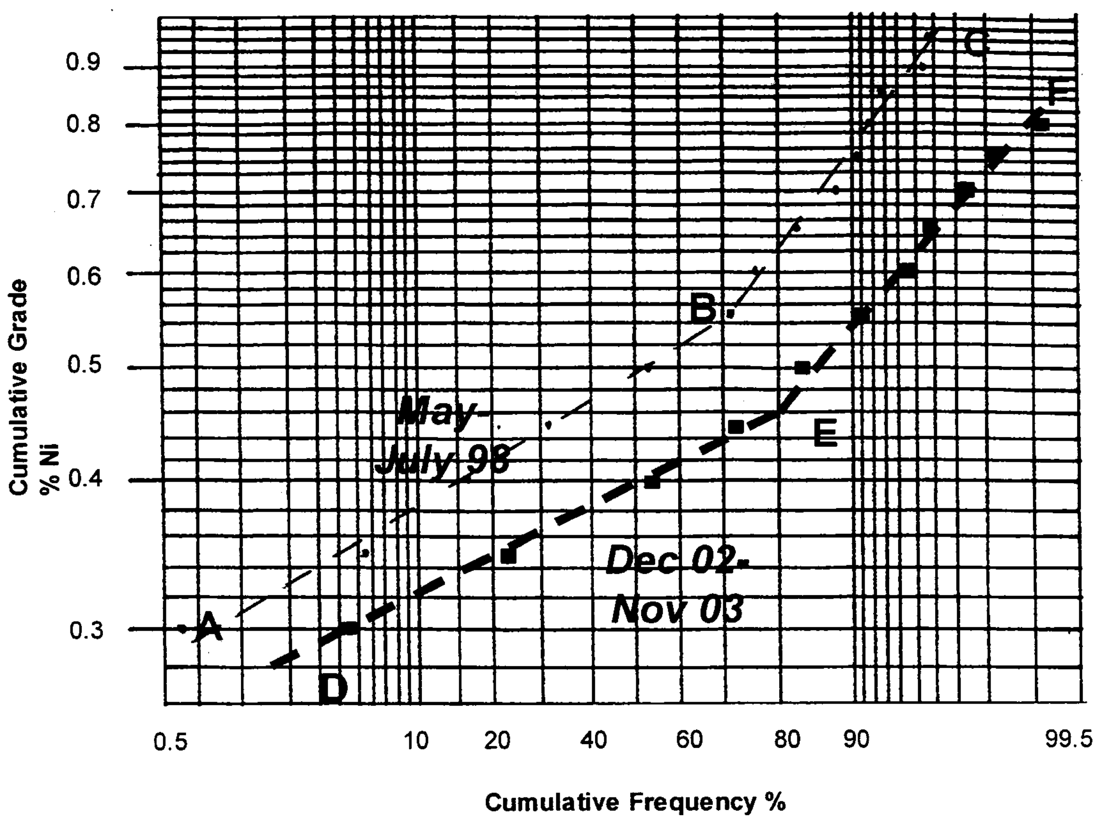


Figure 29 - Probability Plot of Raglan Final Tailings Nickel Grades

Figure 29 shows a lognormal bimodal distribution ABC, with a clear inflection point at approximately 71% cumulative frequency. Line ABC represents the probability plot of nickel grades for the period May-July 1998. This infers that the two subdistributions account for 71 and 29% of the composite distribution, respectively. The form of this plot is generically consistent with that of the Raglan Final Tailings 2002-3 (line DEF). One difference is the location of the point of inflection. This is to be expected, as concentrator performance varies (or improves from commissioning date onwards into the future). It is logical to describe the domains of Figure 29 as a means of comparing continuous improvement in the concentrator operation. First, the point of inflection has moved from 71% to 80% between 1998 and 2002/3. In 2002/3, the final tailings assayed 0.418% Ni. In the May-July 1998 exercise, they assayed 0.543% Ni. Therefore, as a concentrator improves performance by way of several factors (cumulative gain in operator experience, flowsheet improvement, improved reagent suite etc.), the probability plot of final tailings will monitor this improvement, showing steady movement of the plot from the top left to the bottom right of the graph shown in Figure 29.

In 1998, the surveyed mean grade of ore milled was 3.13% Ni. In the data base for December 2002 to November 2003, the sampled mean grade of ore milled was 3.45% Ni. Despite the increase in the ore grade, the overall tailings losses were reduced from 0.543% Ni to 0.418% Ni. This reduction in tailings losses did not arise by chance. It was the result of three process changes in the concentrator. These were:

1. The rerouting of the recleaner tailings matchpoint from scavenger feed to primary column cleaner feed,
2. The introduction of gangue depressant to the rougher flotation unit operation,
3. The commissioning of a regrind mill to treat the cleaner tailings before scavenger flotation.

Further, using the model proposed in Figure 29, the shift in the distribution as a result of the above process changes shows that the weight of the higher tailing grade component has been reduced from 29% to 19%. The overall tailing losses, expressed as nickel loss from ore milled, have been reduced from 13% in 1998 to 10% in 2002/3. This is a much more significant improvement than would have been achieved if only the high-grade component of the tailings distribution had been targeted for process improvement (ca. 0.8% recovery equivalent). In other words, the most significant process improvement targets the two modes of this tailings distribution.

It is proposed that this form of probability plot as a means to track improvements in plant performance is a contribution to knowledge.

4.4. Findings from Characterisation Studies

The foregoing exercises with Raglan drill-core and operations data on Rougher Float Feed, Final Concentrate and Final Tailings have provided useful characterisation. The following may be concluded:

1. Lognormality is present to a measurable extent in drill-core. A three-component composite distribution is proposed, with each subdistribution being a two-parameter lognormal form. This is consistent with the primary geological characteristics of the Raglan End-Member set, wherein the three end members massive sulphides, net-textured sulphides, and disseminated sulphides, each have distinct grade domains. Further potential may exist in the advanced modelling of this composite distribution.
2. In Rougher Float Feed, a two-component composite distribution is suggested, with each subdistribution consistent with a two-parameter lognormal form. The level of lognormality is residual.

3. The physical processing of the ore at the Raglan concentrator separates the mineral phases into a Final Concentrate and a Final Tailing. The high recovery of nickel and copper places most of the paymetal sulphides in the Final Concentrate. This results in a normal distribution of nickel assays in Final Concentrate.
4. The above physical processing produces a Final Tailing stream, which contains a small amount of the paymetals present in the Rougher Float feed. This stream demonstrates the characteristics of a bimodal lognormal distribution, reflecting many possible causes.
5. When the distribution parameters are estimated for the above three distributions, the sample mean is efficiently estimated by both the arithmetic mean and by the geometric mean. This is because of the large number of observations, and the implications of the Central Limit Theorem, and because of the residual (rather than the full) lognormality.
6. When the acceptance limits at the 95% level are estimated for these three distributions, consistent differences are noted between those limits calculated from the Normal Distribution (associated with the arithmetic mean), and those calculated from the Lognormal Distribution (associated with the geometric mean). In this regard, the latter provides skewed acceptance limits, with the lower limit closer to the mean than is the case from the Normal Distribution. The upper confidence limit is also higher in the Lognormal mode than in the Normal mode.
7. The sampling of feed, concentrate and tailings in the Raglan operations should take the above characteristics into account.
8. It is proposed that another contribution to knowledge in this study is the disappearance of full lognormality from drill-core into the concentrator, as the unit processes liberate, then separate, the minerals. In the concentrator, residual lognormality was found in the Rougher Float Feed and the Final Tailings; the Final Concentrate showed

characteristics of a normal distribution. This aligns with the physical interpretation of **Cochran's** theory of stratified sampling, **1946**, and should be tested elsewhere.

4.5. Semivariogram Exercise

Short-Term Semivariogram

The auto-correlation in time of ore grades, measured as Rougher Flotation Feed, was studied. The Raglan operations data of June 1998 were used. These data are shown in Appendix 7, page 218. These took the form of Process Information (PI) books. Measurements of nickel in Raglan Rougher Float Feed exist at the 15-minute level. The method of measurement is by X-Ray Fluorescence (XRF) in the Courier system. The data extract was from 12h00 on the 1st June to 05h00 on the 2nd June 1998. A total of 69 data points resulted from this extract. All data within this extract were used. [*The reader is reminded that the XRF system is less accurate than the formal wet chemical methods which are used at Raglan for metal accounting purposes.*] The conventional Matheron semivariogram, as described in the statistical review, was used. The raw data and calculations for this exercise are shown in the Appendix. A summary of the semivariance by time is shown in Table 34.

Table 34 - Semivariance with Time

Raglan Rougher Float Feed 1 and 2 June 1998 : Ni

Datum #	Semivariance	Datum #	Semivariance	Datum #	Semivariance
1	0.002	24	0.011	47	0.011
2	0.004	25	0.011	48	0.011
3	0.007	26	0.011	49	0.012
4	0.008	27	0.010	50	0.012
5	0.008	28	0.010	51	0.012
6	0.009	29	0.010	52	0.011
7	0.009	30	0.010	53	0.011
8	0.009	31	0.011	54	0.010
9	0.009	32	0.011	55	0.009
10	0.009	33	0.012	56	0.008
11	0.008	34	0.013	57	0.007
12	0.008	35	0.013	58	0.005
13	0.008	36	0.014	59	0.005
14	0.009	37	0.015	60	0.006
15	0.010	38	0.015	61	0.008
16	0.010	39	0.015	62	0.011
17	0.010	40	0.014	63	0.011
18	0.010	41	0.014	64	0.010
19	0.010	42	0.014	65	0.009
20	0.011	43	0.014	66	0.006
21	0.012	44	0.013		
22	0.012	45	0.012		
23	0.012	46	0.011		

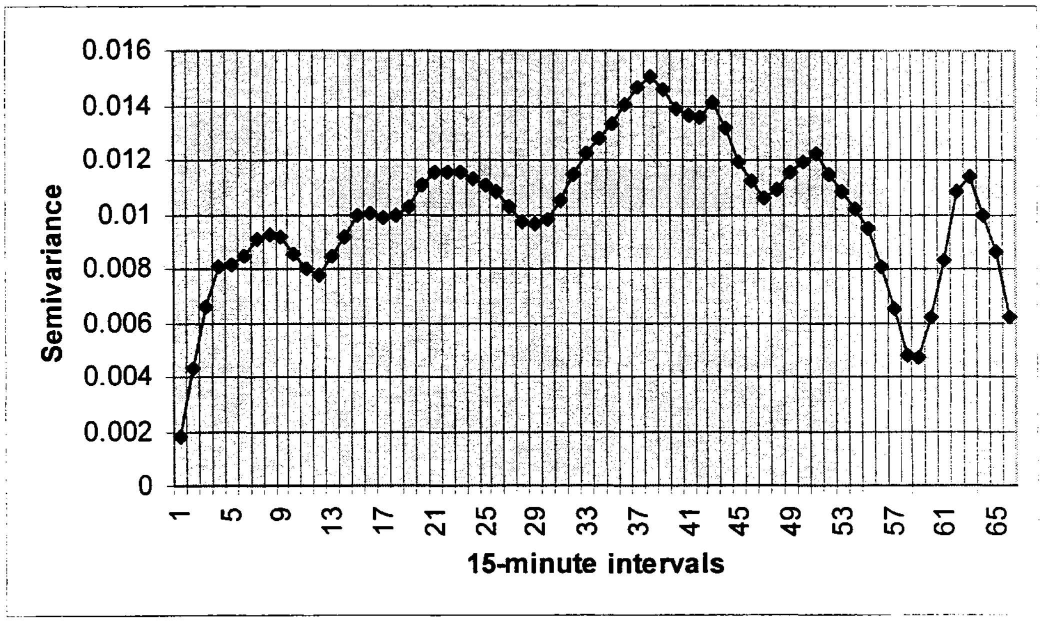


Figure 30 - Short-Term Semivariogram of Nickel in Rougher Float Feed

Inspection of the semivariogram in Figure 30 suggests a cyclic pattern. This has been found elsewhere in concentrator semivariogram exercises **[Hunt, 1990]**. In part for this example, some of the cycling could be attributed to the incomplete blending of the three Raglan End-Members in the crushing, ore storage and milling operations. A method was proposed whereby partial recycling of the mill feed to the ore storage bins in addition to the direct feeding of the stored ore to the primary mill **[Inoue and Imaizumi, 1990]**. For the present case however, it would seem that the area of influence lies between 1 and 5 sampling units, or between 15 and 75 minutes. Thereafter, a cycling pattern develops. An initial conclusion might be that the survey units, which are two hours long, first of all exceed this 75-minute period, and second, should be spaced at least 75 minutes apart. Due to the related logistics of consolidating sample material gathered from the two-hour survey unit, which involve some 3-4 hours anyway, practical considerations would space the closest set of two-hour survey units some 3-4 hours apart.

Long Term Semivariogram

A longer term semivariogram was constructed, using the Rougher Float Feed data from December 2002 to February 2003, Appendix 1, page 194. This was done in order to cross-check the above initial finding. Shifts during which the milling operations were off-line were considered as time in which mined and crushed ore was stockpiled or in process anyway, thus the last shift during which operations were active before the off-line time were regarded as the 'previous' shifts to the subsequent startup. The semivariogram for this exercise totalled 164 shifts and produced the pattern shown in Figure 31. Inspection of Figure 31 suggests that a trend exists for shift spacing values of between 5 and 10 shifts. Thereafter, a slight cycling pattern develops, in which the semivariance rises and falls about a sill. This is consistent with the cycling pattern of the short-term semivariogram. It is suggested that two-hour survey units be separated by at least 6 shifts or 3 days.

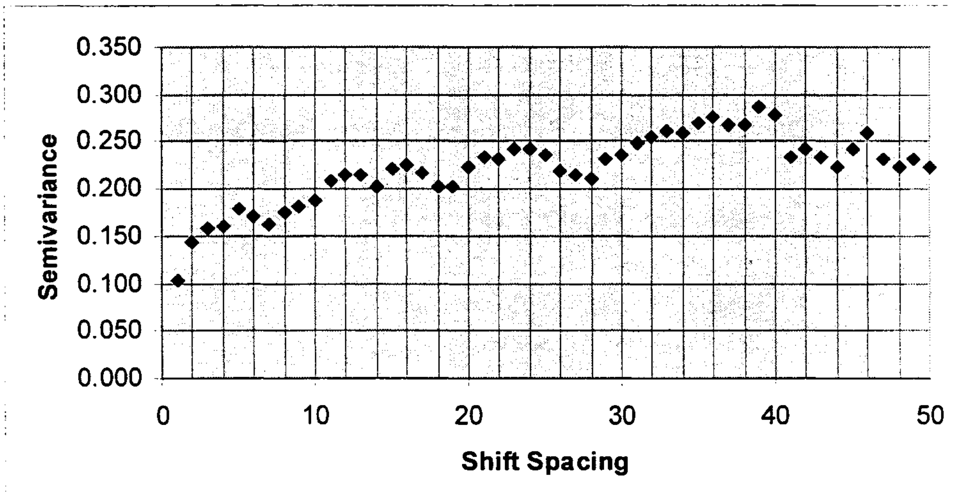


Figure 31 - Long-Term Semivariogram of Nickel in Rougher Float Feed

4.6. Raglan Survey June 1998

A review now follows, describing the Raglan 1998 survey. In this review, the reader is reminded that the ***unimproved standard Falconbridge heuristic prototype model*** was used, without any improvements such as may be developed in this study. The purpose of this review is to demonstrate the shortcomings of this particular model, and why and how it had to be improved. Without exception, this section of the review is performed on the basis of the Normal Distribution. At a later stage of this thesis, identified improvements that are the products of the study will be added, and the same Raglan 1998 survey data revisited, with demonstrations of these improvements.

Existing Model: 1997-2002

An internal reference distribution of the individual mill feed grades *i.e.* one mill feed grade measurement per two-hourly survey unit, was constructed. The individual results and those of the internal reference distribution are shown in Table 35.

Table 35 - Internal Reference Distribution : Raglan Mill Survey June 1998

Survey Number	Mean Sampled Mill Feed Grade, %Ni
1	3.05
2	2.82
3	2.57
4	2.63
5	3.38
6	3.28

The measurements of this distribution are: sample mean 2.96 % Ni, sample standard deviation 0.337% Ni, number of observations 6. Acceptance limits by the Normal Distribution are thus 3.62 and 2.30 % Ni. Inspection of the individual survey units against these criteria shows that all 6 survey units are accepted. An external reference distribution of the grade of ore milled was collected from the routine daily metal accounting assay data across the month surrounding the survey. These results are summarised in Table 36. In this instance the data were collected per 12-hour shift.

Table 36 - External Reference Distribution: Raglan Mill Survey June 1998

Shift No.	Production Circuit Feed Grade %Ni	Shift No.	Production Circuit Feed Grade %Ni	Shift No.	Production Circuit Feed Grade %Ni
1	3.47	19	2.90	37	2.94
2	3.54	20	2.96	38	2.75
3	3.45	21	2.81	39	3.03
4	3.27	22	2.77	40	3.36
5	3.40	23	2.93	41	3.23
6	2.92	24	3.04	42	3.09
7	2.74	25	3.00	43	3.27
8	2.94	26	2.82	44	3.41
9	2.74	27	2.85	45	3.48
10	2.82	28	2.84	46	3.35
11	3.22	29	2.84	47	3.42
12	3.59	30	2.69	48	4.04
13	3.47	31	2.65	49	3.75
14	3.41	32	2.70	50	3.97
15	3.32	33	2.85	51	3.32
16	3.17	34	2.79	52	3.36
17	2.86	35	2.81	53	3.74
18	2.95	36	2.93		

The measurements of this distribution are: sample mean 3.13 % Ni, sample standard deviation 0.344% Ni, observations 53. Acceptance limits are thus 3.82 and 2.44% Ni. Inspection of the internal reference distribution, *i.e.* the individual surveys, shows that all six surveys are accepted at the second stage of adjudication. Therefore, all six surveys are accepted into the composite. A difference in reference distribution means of (2.96 – 3.13) = -0.17% Ni, or (100*(-0.17))/3.13= -5.4%, is noted.

Comparison of Means

Reconciliation of the sample means of the internal and external reference distributions suggests the question as to whether these two means are significantly different (an issue that was not addressed in the 1997-2001 Falconbridge prototype). Initially the Variance Ratio Test, or Snedecor's F Test, leads us to

$$F = \frac{0.344^2}{0.337^2} = \frac{0.118}{0.114} = 1.04 \qquad [24]$$

Referring to the tables of the F distribution, the critical values of F for 5 and 52 degrees of freedom are 2.41 and 3.53 at the upper 5% and 1% points respectively **[Box *et al*, 1978, Table D, pp. 638-639]**. It is accordingly concluded that the variances of these two samples are sufficiently similar to allow comparison of the mean values. To perform this comparison, a pooled estimate of variance must first be made before entering the t-test, since the values of n are different for the two reference distributions.

The pooled estimate of variance may now be estimated from the two reference distributions, using the method of **Box *et. al.*, 1978, Chap. 3, pp. 74-77**. The details were reviewed in the Introduction and Scope, page 33.

The calculations from the Internal Reference Distribution are shown in Appendix 11.2. The sample mean is 2.955% Ni, with a sample standard deviation of

0.337% Ni (Table 10). The sum of squared differences from the sample mean is 0.567.

The equivalent calculations from the External Reference Distribution are shown in Appendix 11.3. The sample mean is 3.132% Ni, with a sample standard deviation of 0.337% Ni (Table 10). The sum of squared differences from the sample mean is 6.170.

The Pooled Estimate of Variance, or PEV, is now calculated as

$$PEV = \frac{\sum_{i=1}^{n_1}(x_i - m_1)^2 + \sum_{i=1}^{n_2}(y_i - m_2)^2}{(n_1 + n_2 - 2)} \quad [25]$$

where

m_1 = sample mean of the Internal Reference Distribution

m_2 = sample mean of the External Reference Distribution

x_i = the set of I data in the Internal Reference Distribution

y_i = the set of I data in the External Reference Distribution

n_1 = the number of observations in the Internal Reference Distribution

n_2 = the number of observations in the External Reference Distribution

With the data from Tables 13 and 14, this amounts to

$$PEV = \frac{(0.567 + 6.170)}{(6 + 53 - 2)} = 0.118 \quad [26]$$

The Pooled Standard Deviation, s_p, of the overall data is thus √ [0.118] or 0.344. The estimation of t is now approached using

$$t_0 = \frac{(m_1 - m_2)}{s_p(\frac{1}{n_1} + \frac{1}{n_2})^{0.5}} \quad [27]$$

From the above values,

$$t_0 = \frac{(2.955 - 3.132)}{0.344.(\frac{1}{6} + \frac{1}{53})^{0.5}} = -1.20 \quad [28]$$

Referring to the table of critical t values at the 5% and 10% levels, with 57 degrees of freedom, t_c is approximately 1.98 and 1.68, respectively. The observed difference in sample means is therefore not significant at the 5% level, but may be significant at a level below 10%. It has been the past practice of the Falconbridge heuristic statistical benchmark survey model to work at the 5% level, so in terms of that standard, there is no issue with the observed difference in sample means of the two reference distributions. Put differently, use of the t-distribution to estimate the significance level of this observed difference leads to a result of 19.3%, which for the purposes of this study is not significant. At the two-tail level, which is more restrictive than the one-tail level, and which is the appropriate test here (since there is no foreknowledge or hypothesis that one of the means should be higher than the other), one could not reach an 81% confidence, let alone a 95% confidence level, that the average nickel grade of the internal reference distribution was significantly different from that of the external reference distribution. Only an 80% confidence level was obtained from this test. However, this is the rejection confidence level *i.e.* that an 80% confidence has been reached that the Internal Reference Distribution has a sample mean grade that is significantly different from that of the External Reference Distribution.

Use of the Grubbs Test on the Internal Reference Distribution

As a cross-check, the internal reference distribution is inspected to consider any of the 6 observations as an outlier. Reference is now made to the method of

Grubbs, 1969. This is a more appropriate approach than the two sample standard deviations used in the existing Falconbridge model.

The Internal Reference Distribution is now examined by sorting the data from lowest to highest, then calculating the difference of each observation from the sample mean. This difference is then divided by the sample standard deviation. The results are shown in Table 37.

Table 37 - Grubbs Outlier Test : Internal Reference Distribution
Raglan Mill Survey June 1998

Survey Number	Sampled Grade of Rougher Float Feed, % Ni	Difference from Sample Mean	Grubbs t
3	2.57	-0.39	-1.16
4	2.63	-0.33	-0.98
2	2.82	-0.14	-0.42
1	3.05	+0.09	+0.27
6	3.28	+0.33	+0.98
5	3.38	+0.42	+1.25

This exercise was repeated for the Internal Reference Distribution, using the Pooled Estimate of Variance to obtain the Pooled Estimate of the Standard Deviation. This resulted in $s_p = 0.344$. The results of this test are shown in Table 38.

Table 38 - Grubbs Outlier Test Using the Pooled Estimate
of Standard Deviation Raglan Mill Survey June 1998

Survey Number	Sampled Grade of Rougher Float Feed, % Ni	Difference from Sample Mean	Grubbs t (Using PEV Std Dev)
3	2.57	-0.39	-1.13
4	2.63	-0.33	-0.96
2	2.82	-0.14	-0.41
1	3.05	+0.09	+0.26
6	3.28	+0.33	+0.96
5	3.38	+0.42	+1.22

The Internal Reference Distribution sample mean was 2.96% Ni, and standard deviation, 0.337% Ni. The critical value of Grubbs t for n=6 observations is 1.82 for the 5% level, so on the basis of either the standard deviation estimated from the Internal Reference Distribution, or the standard deviation estimated from the Pooled Estimate of Variance, no observation may at this stage be discarded. The cross-check of Grubbs agrees with the outcome of the Pooled Estimate of Variance.

4.7. Two New Survey Models

The reader is reminded that the main objective of this study is to develop and test two new survey models. These were functionally set out in section 1.3. Both of these new models will be based on combinations of full lognormality and residual lognormality, with the Normal distribution referred to when applicable. Central use will be made of the metal grade of ore as the testing parameter in both cases.

The *Statistical Benchmark Survey* will characterise typical metallurgical performance of the concentrator for typical ore treated. Its use will be to identify existing flowsheeting opportunities in the existing concentrator operation.

The *Campaign Survey* will characterise the metallurgical behaviour of either a problematic ore type or a future ore mixture. In this case, such ore has to be specially mined for a week - long concentrator operations campaign during which the operation is surveyed. The concentrator information can then be linked back to a specific mining area of the ore reserve. The benefit will be in the development of predictive information towards focussing design work on flowsheet changes that will be necessary in order to more effectively treat either of these ore cases.

For either the Statistical Benchmark Survey or the Campaign Survey, a better understanding of the form and use of the lognormal distribution will be required to some extent or another. Exercises in this regard now follow.

4.8. Transformation to the Logarithmic Dimension

The exercises on Raglan operations data concluded that lognormality is present to some extent in the time dimension, when measurements of mill feed grade are made. A new generation of 'conventional' mill survey is thus called for, in the case where a typical benchmark measurement of the mill performance is desired. In this second section of reviewing the data for the 1998 Raglan survey, the new lognormal models will be tried and tested. The reader is now reminded that, for the following section, parameter estimation will be performed from two lognormal platforms. These will be:

1. Krige, 1962, for larger data sets;
2. Sichel, 1966, for small data sets.

Adaptation of Krige's Method, 1962

This method is reviewed on p. 64. The May-July 1998 Raglan Rougher Float Feed nickel data were used. These data are shown in Appendix 4, page 212. The shift level nickel grades totalled 168 observations after exclusion of mill down time. Each shift data point as % Nickel was transformed to the Naperian logarithm base by the transform

$$z = Ln(x) \qquad [33]$$

where x= the raw nickel grade in % Ni, and z= the transformed value. The logarithmic mean was the calculated as

$$\bar{z} = \frac{1}{n}.\sum_{i=1}^{n} z_i \qquad [34]$$

where $\bar{z}$ is the logarithmic mean from n sample data. The logvariance was then calculated as Var_z according to

$$Var_z = \frac{1}{n-1}\sum_{i=1}^{n}(z_i - \bar{z})^2 \quad [35]$$

The transform standard deviation was then calculated as s_z, where

$$s_z = (Var_z)^{0.5} \quad [36]$$

The upper and lower transform limits at the 95% level (recommended by Krige) are then

Upper Limit UCL_z:

$$UCL_z = \bar{z} + 2 * s_z \quad [37]$$

Lower Limit LCL_z:

$$LCL_z = \bar{z} - 2 * s_z \quad [38]$$

The inverse transform functions then returned their arithmetic equivalents according to their Naperian antilogarithms:

Inverse Transform Mean z_0

$$z_0 = e^{\bar{z}} \quad [39]$$

Inverse Transform Upper Confidence Limit UCL_{z_0}

$$UCL_{z_0} = e^{UCL_z} \quad [40]$$

Inverse Transform Lower Confidence Limit LCL_z

$$LCL_{z_0} = e^{LCL_z} \quad [41]$$

Rougher Float Feed

These calculations resulted in the mean and associated confidence limits for the three-month data block May-July 1998, as shown in Table 39.

Table 39 - Krige Estimate of Raglan Rougher Float Feed

Estimated Mean and Associated Confidence Limits at the 95% Level
May-July 1998 Using Naperian Log Transform Method of Krige, 1962

In Naperian Log Transform		After Inverse Transform	
Logarithmic Mean	1.179	Mean % Ni	3.251
Logvariance	0.010209		
Log Standard Deviation	0.10104		
Upper Confidence Limit	1.381	Upper Confidence Limit	3.98
Lower Confidence Limit	0.977	Lower Confidence Limit	2.66

The arithmetic mean and associated limits at the 95% level for this data block are mean: 3.27% Ni; Upper Confidence Limit: 3.93% Ni; and Lower Confidence Limit: 2.61% Ni. Again the arithmetic mean is closely in agreement with the uncorrected geometric mean, however the associated confidence limits are very different.

This set of calculations was repeated for the month of June 1998. These calculations resulted in an arithmetic mean of 3.13% Ni, and upper and lower confidence limits of 3.82 and 2.44% Ni respectively. Use of the Naperian Transform by Krige produced a geomean of 3.11 % Ni with upper and lower confidence limits of 3.86 and 2.51% Ni respectively.

These two External Reference Distributions differ only by the sampling period, and are summarised in Table 40.

Table 40 - Summary of External Reference Distributions

Sampling Period	Parameter	Arithmetic Basis	Krige (Naperian) Basis
1-Month ERD June 1998 N=53	Mean	3.13	3.11
	Upper Control Limit 97.5%	3.82	3.86
	Lower Control Limit 2.5%	2.44	2.51
3-Month ERD May-July 1998 N=168	Mean	3.27	3.25
	Upper Control Limit 97.5%	3.93	3.98
	Lower Control Limit 2.5%	2.61	2.66

Table 41 lists the six measurements of Raglan Rougher Float Feed from the 1998 survey. Inspection of these opposite the confidence limits from Table 40 shows that the arithmetically-estimated parameters of the External Reference Distribution accept all six surveys. However, only one of the two geometrically-derived estimates identifies any outliers. These are surveys 3 and 4. This is done by the External Reference Distribution of May-July 1998, *i.e.* derived from 3 months of operations data. This distribution has upper and lower limits of 3.98 and 2.66% Ni. It would seem, therefore, that the geometrically-derived External Reference Distribution is the appropriate form, however it is sensitive to the size of data set used. The consistency of this outcome will be cross-checked by the outcome(s) of the Internal Reference Distribution by Sichel.

Table 41 - Extract from the Internal Reference Distribution
Raglan Survey 1998

.Survey No.	Grade of Rougher Float Feed, % Ni
1	3.05
2	2.82
3	**2.57**
4	**2.63**
5	3.38
6	3.28

As reported earlier, the arithmetic mean of this Internal Reference Distribution is 2.96% Ni. Exclusion of surveys 3 and 4 shortens the Internal Reference Distribution to 4 observations with a new sample mean of 3.13% Ni. Comparison of this new adjusted sample mean with that of the conventional Falconbridge External Reference Distribution (with a mean of 3.13% Ni : Table 21) shows that close agreement of the shortened Internal Reference Distribution and the conventional Falconbridge External Reference Distribution has now been achieved. This outcome is intuitive. The process of outlier elimination should move the shortened sample mean towards the robust mean of the External Reference Distribution. It is suggested that this successful outcome of outlier identification is a direct result of using a lognormal form of distribution for parameter estimation. Previous attempts using the Normal Distribution format failed to identify these outliers. Note that the Krige geometric mean fails to correct for logvariance, where the Sichel t estimator does. The reader is reminded that the correction of the geometric mean in the Sichel t-estimator consists of a multiplying term based on the number of observations and the logvariance. The exercise that follows now uses the Sichel t-estimator.

Use of the Sichel t-Estimator on the External Reference Distribution

The External Reference Distribution of the Raglan June 1998 survey (Table 36) was assessed using the Sichel t-estimator in TRIPOD. The results are summarised in Table 42.

Table 42 - Sichel t-Estimator Results - Raglan Survey 1998
External Reference Distribution

Parameter	Result
Uncorrected Geomean	**3.11**
Correction Factor	**1.0064**
Corrected Geomean	3.13
Logvariance	0.0115
Upper Confidence Limit Coefficient	1.0288
Upper Confidence Limit (97.5%)	3.22
Lower Confidence Limit Coefficient	0.971
Lower Confidence Limit (2.5%)	3.04

The value of Sichel's t for this External Reference Distribution is 3.13% Ni. Reference to Table 36, which reports the outcome of the arithmetic (normal) mean, shows an arithmetic mean of 3.13% Ni. At this low level of logvariance, which is found in residual lognormality, the sample means from either Sichel's model or the Normal Distribution are closely in agreement. In this case, exactly the same data block was used for either the Normal or the Sichel estimators. It is the associated confidence limits of the population mean rather than those of individual measurements.

Use of the Sichel t-Estimator on the Internal Reference Distribution

The trials of Normal Distribution Statistics on the Internal Reference Distribution, *i.e.* using two arithmetically estimated standard deviations as acceptance limits, could find no outlier in the data. Preliminary exercises on the nature of the distribution of Nickel values in Rougher Float Feed showed that residual effects of lognormality were present. In this case, the Krige lognormal method is inapplicable because of the small data set. Rather, it is proposed that the Sichel t-Estimator, which was written specifically for robust estimation of the mean and associated confidence limits in lognormal distributions, be used to describe the Internal Reference Distribution and perform outlier testing. It is acknowledged that the Rougher Float Feed nickel grade distribution is residually lognormal and at least bimodal. *[In his publication of **1966**, **Sichel** described the sampling of a* *unimodal fully lognormal distribution by his t-estimator. The following sampling exercise will be cross-checked with the outcomes of the larger External Reference Distribution by Krige, to determine whether consistent conclusions are drawn.]*

In applying Sichel's t-estimator to the Internal Reference Distribution, we are knowingly applying to individual data points the confidence limits of the population mean, resulting in a much higher probability of rejection than the '5%' implied in the 95% confidence limits. Thus, a rejection rate of 2 out of 6 surveys is not unexpected. This is consistent with the avowed objective of producing a high-confidence composite sample.

This can be illustrated with an analogy of the justice system, whose primary objective is to ensure that those convicted (*i.e.*rejected) are indeed guilty of the crime with a high degree of confidence (say 95%). The burden of proof is set very high to ensure that less than 5% of those convicted were in fact innocent. To achieve this objective, the conviction rate of criminals is less than 95%. In other words, to make sure that less than 5% of those convicted are innocent, we must accept that less than 95% of criminals will be convicted. Most statistical tests use a similar approach, and the probability of failing to reject a measurement that should be rejected is defined as the beta error, or 'false negative'. It is generally higher than the probability of accepting an unacceptable measurement (a false positive), alpha.

In the survey model, it is far more important to reject non-representative tests than to include all acceptable tests. With a sample size of 6 tests, more than 5% of the data can be rejected, and rejecting 1-2 surveys with tight controls still leaves 4-5 surveys for the composite samples. In other words, the probability of alpha is increased to reduce that of beta.

Within the domain of n = 6, there is no other statistical method that can robustly estimate its parameters. We can afford to lose more than 5% of the data. The idea is that of the 6 surveys, we have 1-2 expendable surveys or observations.

In this way, the luxury of having more surveys than one actually needs allows the use of tight control limits without depriving the accepted data set of the minimum of 4 which are needed. The raw data from the Internal Reference Distribution of Raglan Rougher Float Feed in the June 1998 survey were:

Table 43 - Internal Reference Distribution Raglan Survey 1998

.Survey No.	Grade of Rougher Float Feed, % Ni
1	3.05
2	2.82
3	2.57
4	2.63
5	3.38
6	3.28

The Sichel t-Estimator, in TRIPOD, produced the following estimate for associated confidence limits at the 95% level:

Table 44 - Sichel t-Estimator Results - Raglan Survey 1998
Internal Reference Distribution

Parameter	Result
Uncorrected Geomean	**2.939**
Correction Factor	**1.0054**
Corrected Geomean	2.955
Upper Confidence Limit Coefficient	1.1844
Upper Confidence Limit (97.5%)	3.50
Lower Confidence Limit Coefficient	0.9205
Lower Confidence Limit (2.5%)	2.72

The conversion by **Clark, 1987**, of the Sichel t-estimator to a computer programme that will calculate the associated confidence limits of the Sichel t at any desired level in a few moments of work, makes the above estimation simpler and more functional. This function is resident in TRIPOD **[Clark, 2004]**. Inspection of the Internal Reference Distribution data with these limits rejects any data outside $x > 3.50$ and $x < 2.72$% Ni, resulting in rejection of 2.57 and 2.63 % Ni, leaving a shortened data set of 3.05, 2.82, 3.38 and 3.28% Ni, now averaging 3.13 % Ni. This compares to the original June External Reference Distribution mean of 3.13% Ni, with a difference of (3.13-3.13) = 0.0% Ni or a relative error of

(100*0.0)/3.13 = 0 %. The Krige model has yielded an entirely consistent result after outlier identification and rejection.

So far this exercise has used all of the six survey measurements in the parameter estimation. Strictly speaking, for outlier rejection, the examination or inspection of any single one of these survey grades should exclude it in the parameter estimation, otherwise statistical independence might be lost. In order to address this matter, a separate exercise was performed with the six survey grades reported in Table 42. In this exercise, six steps were performed. In each of these six steps, one of the original six observations was chosen and was excluded from the Sichel t calculation. The upper and lower confidence limits were also calculated at the 95% level. The purpose of this exercise was to determine if such exclusion of single observations altered the original findings in terms of outlier rejection.

The six original measurements from Table 42 were: 3.05; 2.82; 2.57;2.63; 3.38 and 3.28% Ni. In all cases, TRIPOD was used for the parameter estimation at the 95% level.

In the first step, the last value, *i.e.* 3.28% Ni, is excluded from the data set. Thus, we are trying to determine if the value 3.28% Ni is an outlier by excluding that observation from the data set and estimating Sichel's parameters on the following data: 3.05; 2.82; 2.57;2.63; 3.38. The results were: t: 2.89%Ni; UCL: 3.54% Ni; and LCL: 2.63% Ni. The data point of 3.28% Ni falls within the upper and lower limits of 3.54 and 2.63%Ni, thus 3.28 cannot be rejected from the data set.

In the second step, the second last value, *i.e.* 3.38% Ni, is excluded from the data set. In a similar manner to the above calculations, we obtain : t: 2.87%Ni; UCL: 3.46% Ni; and LCL: 2.64% Ni. The data point of 3.38% Ni falls within the upper and lower limits of 3.46 and 2.64% Ni, thus 3.38 cannot be rejected from the data set.

In the third step, the third last value, *i.e* 2.63% Ni, is excluded from the data set. In a similar manner to the above calculations, we obtain: t: 3.02%Ni; UCL: 3.70% Ni; and LCL: 2.75% Ni. The data point of 2.63% Ni does not fall within the upper and lower limits of 3.70 and 2.75%Ni, thus 2.63 is rejected from the data set.

In the fourth step, the fourth last value, *i.e* 2.57% Ni, is excluded from the data set. In a similar manner to the above calculations, we obtain: t: 3.03%Ni; UCL: 3.67% Ni; and LCL: 2.78% Ni. The data point of 2.57% Ni does not fall within the upper and lower limits of 3.67 and 2.78%Ni, thus 2.57 is rejected from the data set.

In the fifth step, the fifth last value, *i.e.* 2.82% Ni, is excluded from the data set. In a similar manner to the above calculations, we obtain: t: 2.98%Ni; UCL: 3.75% Ni; and LCL: 2.69% Ni. The data point of 2.82% Ni falls within the upper and lower limits of 3.75 and 2.69%Ni, thus 2.82 cannot be rejected from the data set.

In the final step, the first value, i.e. 3.05 % Ni, is excluded from the data set. In a similar manner to the above calculations, we obtain: t: 2.94%Ni; UCL: 3.69% Ni; and LCL: 2.64% Ni. The data point of 3.05% Ni falls within the upper and lower limits of 3.69 and 2.64%Ni, thus 2.94 cannot be rejected from the data set.

As a result of this exercise, the values 2.57 and 2.63% Ni were rejected from the original data set. This finding is consistent with the original calculation using all six observations. It is also worth noting that in those steps which were testing data points 3.05; 2.82; 3.38 and 3.28% Ni., and in which none of these values were rejected, the outlier points 2.57 and 2.63% Ni were consistently identified as outliers by that short data set.

It is concluded that, for the Internal Reference Distribution, the population mean is efficiently estimated by either arithmetic or corrected geometric approaches. When using a lognormal format however, the estimation of the final sample mean in the Internal Reference Distribution is significantly affected by an outlier rejection model based on the Sichel t-estimator at the 95% confidence level. The estimation of the associated confidence limits is very different in the lognormal format, and causes a different outlier rejection result to that of the Normal Distribution. Use of the former shortens the Internal Reference Distribution and adjusts the mean to closer agreement with the mean of the External Reference Distribution.

Use of the Transformed t-Distribution

A cross-check on acceptance limits will now be made on the Sichel t-estimator method for the Internal Reference Distribution. The basis of this check will be to estimate the confidence interval of the population mean in the Naperian Log Transform. This amounts to a t-distribution in the Naperian Transform. The reader is referred to Table 42, page 130, for the raw data that amount to the 6 two-hour survey units of the Raglan 1998 survey.

The standard error of a mean is calculated as s_ε, where

$$s_\varepsilon = \frac{s}{\sqrt{n}} \qquad [42]$$

where

s = sample standard deviation

n = the number of observations.

In a data set of 6 observations, we lose one degree of freedom in estimating the population mean. In referring to the t-distribution for the 0.025 tail at 4 degrees of freedom, the critical value of t is 2.776. The 95% confidence limits of the population mean in this case are thus the population mean plus or minus $(2.776)/(5^{0.5})$ = 1.24 times the standard deviation.

If these calculations are made in the Naperian Log Transform, and the anti-Naperian log taken, a set of skewed confidence limits about the mean result.

In the case of the Raglan 1998 survey data (Table 42), this amounts to an uncorrected geomean of 2.94% Ni with upper and lower confidence limits of that mean of 3.39 and 2.55% Ni respectively. Comparison with the equivalent Sichel t estimator data show t = 2.955% Ni with upper and lower confidence limits of 3.50 and 2.72% Ni respectively. Where the low level of logvariance contributes to similar mean values, the associated confidence limits only appear similar for the lower confidence limit (compare Sichel : 2.72% Ni opposite Naperian Standard Error Method : 2.55% Ni). The upper confidence limits are quite different (compare Sichel: 3.50% Ni with Naperian Standard Error : 3.39% Ni). These differences are summarised in Table 45.

Table 45 - Comparison of Sichel t Estimator and Naperian Geomean
On Raglan Internal Reference Distribution June 1998 : Basis – All Data

Parameter	Sichel	Naperian Geomean
Mean Grade % Ni	2.96	2.94
Upper Confidence Limit of Mean at 97.5%	3.50	3.39
Lower Confidence Limit of Mean at 2.5%	2.72	2.55

Examination of the Raglan 1998 survey data (Table 42) shows that the Transformed t-Distribution approach accepts all six observations. So far this exercise has used all of the six survey measurements in the parameter estimation. Strictly speaking, for outlier rejection, the examination or inspection of any single one of these survey grades should exclude such a measurement in the parameter estimation, otherwise statistical independence might be lost. As was the case before in the Sichel t-estimator, in order to address this matter, a separate exercise was performed with the six survey grades reported in Table 42. In this exercise, the same six steps were performed as before with the Sichel t-estimator.

The six original measurements from Table 42 were: 3.05; 2.82; 2.57;2.63; 3.38 and 3.28% Ni. In all cases, the Transformed Standard Error was used for the parameter estimation at the 95% level.

In the first step, the last value, *i.e.* 3.28% Ni, is excluded from the data set. Thus, we are trying to determine if the value 3.28% Ni is an outlier by excluding that observation from the data set and estimating the transformed parameters on the following data: 3.05; 2.82; 2.57;2.63; 3.38. The results were: geomean: 2.88%Ni; UCL: 3.31% Ni; and LCL: 2.50% Ni. The data point of 3.28% Ni falls inside the upper and lower limits of 3.31 and 2.50% Ni, and is accepted in the data set.

In the second step, the second last value, *i.e.* 3.38% Ni, is excluded from the data set. In like manner to the above discussion, the results were: geomean: 2.86% Ni; UCL: 3.24% Ni; and LCL: 2.52% Ni. The data point of 3.38% Ni falls outside the upper and lower limits, and is rejected from the data set.

In the third step, the third last value, *i.e* 2.63% Ni, is excluded from the data set. In like manner to the above discussion, the results were: geomean: 3.00% Ni; UCL: 3.45% Ni; and LCL: 2.61% Ni. The data point of 2.63% Ni falls within the upper and lower limits, thus is accepted.

In the fourth step, the fourth last value, *i.e* 2.57% Ni, is excluded from the data set. In like manner to the above discussion, the results were: geomean: 3.02% Ni; UCL: 3.44% Ni; and LCL: 2.65% Ni. The data point of 2.57% Ni falls outside the upper and lower limits, and is rejected.

In the fifth step, the fifth last value, *i.e.* 2.82% Ni, is excluded from the data set. In like manner to the above discussion, the results were: geomean: 2.96% Ni; UCL: 3.46% Ni; and LCL: 2.54% Ni. The data point of 2.82% Ni falls inside the upper and lower limits, and is accepted.

In the final step, the first value, *i.e.* 3.05% Ni, is excluded from the data set. In like manner to the above discussion, the results were: geomean: 2.92% Ni; UCL: 3.41% Ni; and LCL: 2.50% Ni. The data point of 3.05% Ni falls inside the upper and lower limits, and is accepted.

Overall, this approach has rejected two data points from a set of six. The shortened data set now stands at: 3.28; 2.63; 2.82 and 3.05, with a geomean of

2.94 (arithmetic mean 2.95% Ni). Comparison of this new mean with the original derived from all six data points shows that no significant change has occurred in the estimate of the mean using this outlier method.

Use of the Krige Data Acceptance Limits

As a final cross-check on the Internal Reference Distribution, the method of Krige 1962 was tried on the small data set of the Internal Reference Distribution. This will demonstrate that the Krige method of 1962 is inappropriate for a small data set. The reliable estimation of standard deviation in this method relies on large numbers of data points. Since the confidence limits to be produced in this case are proportional to the estimated standard deviation, these limits are likely to be wide.

The Internal Reference Distribution data of Table 40 were used for this purpose to estimate the mean and data confidence limits at the 95% level. This resulted in mean : 2.94% Ni, with upper and lower limits of 3.69 and 2.34% Ni. Whereas the estimate of the mean is similar to other estimates, probably because of the low level of logvariance, the data acceptance limits are very wide in comparison to the other estimates from other models. This method is therefore inapplicable to the Internal Reference Distribution. This is a confidence interval on measured data, which is too wide. The confidence interval on the population mean would be the better approach.

Final Concentrate and Tailings

The foregoing analysis has indicated that the Final Concentrate and Final Tailings, being the two final processing products from the ore milled, have distributions of paymetal values that reflect the engineering process and the residual lognormality of the parent distribution in Rougher Float Feed, respectively. It is intuitive, therefore, to consider these two streams in the quality control process so as to improve the proof of representativeness. If the Final Concentrate and Final Tailings reference distributions indicate that the surveyed Final Concentrate and Tailings are comparable, improved proof is provided. An exercise was conducted, using three months of shift frequency data surrounding the 1998 Raglan survey, to characterise this feature, using the lognormal format of Krige, 1962.

Final Concentrate

The distribution of nickel grades in Final Concentrate by grade is summarised in Table 46. This is a Normal Distribution and is treated as such. The raw data are shown in Appendix 5.

Table 46 - Distribution of Nickel Grades in Raglan Final Concentrate
May-July 1998

Grade % Ni	Frequency %	Grade % Ni	Frequency %
12	0.00	16	36.90
13	1.19	17	23.81
14	4.76	18	10.12
15	20.83	19	2.38

An analysis of the sample mean and associated symmetrical confidence limits at the 95% level follows:

Table 47 - Raglan Final Concentrate by Normal Distribution
May-July 1998 Estimated Mean and Associated Confidence Limits at the 95% Level

In Normal Distribution Format	
Sample Mean	15.74
Standard Deviation	1.15
Upper Confidence Limit (97.5%)	18.03
Lower Confidence Limit (2.5%)	13.45

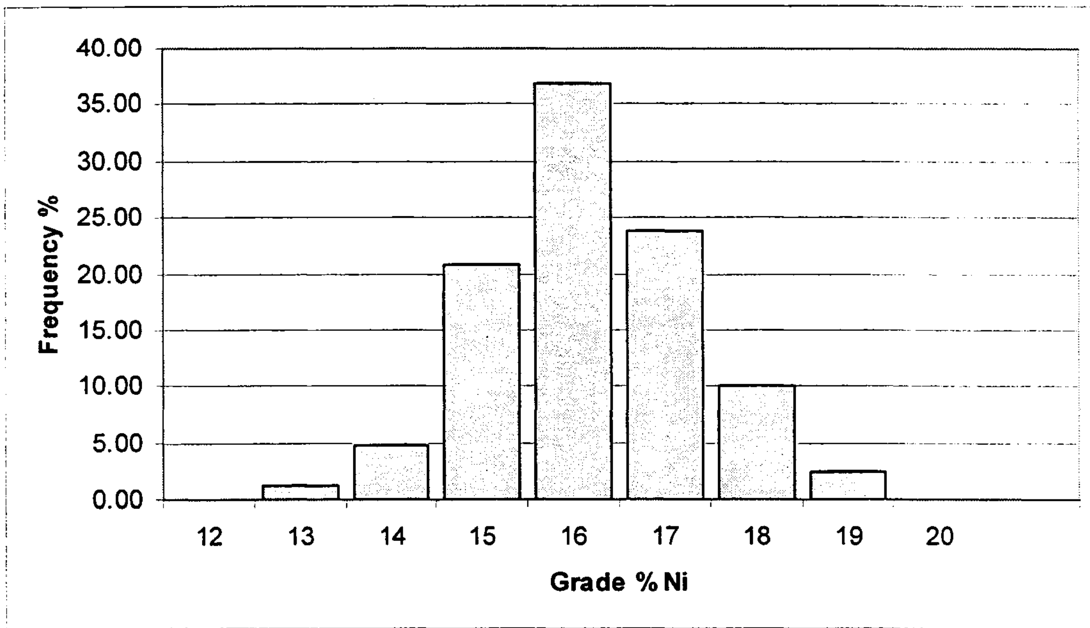


Figure 32 - Distribution of Nickel Values : Raglan Final Concentrate

The actual survey final concentrate values are summarised in Table 48.

Table 48 - Actual Final Concentrate Values from Individual Surveys

Raglan, June 1998

.Survey No.	Grade of Final Concentrate, % Ni
1	16.2
2	16.3
3	**15.5**
4	**15.8**
5	15.0
6	15.2

Inspection of the External Reference Distribution limits shows that none of the individual surveys in Table 35 can be rejected. However, since the Rougher Float Feed grades of survey units were disqualified by the new (lognormal) reference distribution, so also must the equivalent final concentrates, tailings and all other survey samples in those rejected two surveys be disqualified. This shortens the Final Concentrate Internal Reference Distribution to 4 observations averaging 15.68% Ni (compared to 15.67% Ni for all data points) with little or no significant effect on the mean grade of concentrate.

Final Tailings

The distribution of nickel grades in Final Tailings by grade is summarised in Table 49. The raw data are shown in Appendix 6, page 216.

Table 49 - Distribution of Nickel Grades in Raglan Final Tailings
May-July 1998

Grade % Ni	Frequency %	Grade % Ni	Frequency %
0.2	0.0	0.95	0.6
0.25	0.0	1	1.2
0.3	0.6	1.05	1.2
0.35	7.1	1.1	0.6
0.4	11.3	1.15	0.6
0.45	17.3	1.2	0.0
0.5	14.9	1.25	0.6
0.55	15.5	1.3	0.0
0.6	7.1	1.35	0.0
0.65	6.0	1.4	0.0
0.7	7.7	1.45	0.0
0.75	1.8	1.5	0.0
0.8	2.4	1.55	0.6
0.85	0.6	1.6	0.0
0.9	2.4		

This distribution was transformed into the Naperian form using the method of **Krige, 1962**, with the following results:

Table 50 - Raglan Final Tailings by Naperian Transform of Krige, 1962
May-July 1998 Estimated Mean and Associated Confidence Limits at the 95% Level

In Naperian Log Transform		After Inverse Transform	
Logarithmic Mean	-0.66	Mean % Ni	0.52
Logvariance	0.087		
Log Standard Deviation	0.295		
Upper Confidence Limit	-0.068	Upper Confidence Limit	0.93
Lower Confidence Limit	1.249	Lower Confidence Limit	0.29

Table 51 - Actual Final Tailings Values from Individual Surveys
Raglan, June 1998

.Survey No.	Grade of Rougher Float Feed, % Ni
1	0.37
2	0.64
3	**0.42**
4	**0.43**
5	0.55
6	0.47

As with the Final Concentrate, surveys 3 and 4 are disqualified because of the Rougher Float Feed values. This shortens the data set to n=4 with a sample mean of 0.51 % Ni (compare with raw data set mean of 0.48% Ni). Comparison of the remaining 4 observations 0.37, 0.64,0.55 and 0.47% Ni, with the skewed confidence limits of 0.93 and 0.29% Ni, does not select any of these observations for removal from the data set. The mean values of 0.51 % Ni (shortened survey data set mean) and 0.52% Ni (Naperian Log Transform mean of External Reference Distribution) compare well.

4.9. Comparison of Results – Benchmark Surveys, Old and New

The effects of the Naperian transform approach with the reference distributions appear to have improved the agreement between mean values of the Internal and External Reference Distributions. A series of comparisons follows in Table 52:

Table 52 - Comparison of Reference Distribution Means
for Old and New Survey Methods

Survey Type	Rougher Float Feed % Ni		Final Concentrate % Ni		Final Tailings % Ni	
	Internal	External	Internal	External	Internal	External
Old	2.96	3.13	15.67	*	0.48	*
New	3.13	3.13	15.68	15.70	0.51	0.52

*The Old model did not use External Reference Distributions for Final Concentrate and Tailings

4.10. Raglan Survey November 2003 – Campaign Model

Objectives

The reader is reminded that this survey was configured so as to mine and process a particular mixture of ore types, or End Members, so as to obtain predictive properties of the future ore to be mined and processed from 2005 onward. The challenges include the rearrangement of the External Reference Distribution so as to use geological and mining grade data to benchmark the grade of ore to be milled in the campaign. This is the first documented attempt at campaigning a particular ore mixture through a mill for a C*ampaign Survey.*

Estimation of Ore Grade

Actual Ore Mined

The actual ore mined and milled is summarised in Table 53.

Table 53 - Actual Ore Mined

Future Ore at Raglan Operations 16-19 November 2004
[Data Source : Mine Geology]

Date	Detail	Disseminated	Net Textured	Massive	Total	Total Dry
16	Wet Tonnes	394	2193	226	2813	2729
	Grade % Ni	1.7	2.75	5.3	2.81	2.81
17	Wet Tonnes	394	2177	141	2712	2631
	Grade % Ni	1.7	2.71	5.3	2.70	2.70
18	Wet Tonnes	394	2174	254	2822	2737
	Grade % Ni	1.7	2.64	5.3	2.75	2.75
19	Wet Tonnes	466	2004	422	2892	2805
	Grade % Ni	1.7	2.46	5.3	2.75	2.75
	Totals					
	End Member Mass %	**14.7**	**76.1**	**9.3**	**100.0**	
	Wet Tonnes	**1648**	**8548**	**1043**	**11239**	**10902**
	Grade % Ni	**1.7**	**2.64**	**5.3**	**2.75**	**2.75**

Drill-Core Estimate

A detailed drill-core database exists for the relevant blocks of ore to be mined. The data consist of actual chemical analyses of samples which were taken from drill-core and analysed. A total of 70 observations was identified from this database. The details are shown in Appendix 8, and the salient features, in Table 54. The method of **Krige, 1962**, was used to estimate the geometric mean and associated confidence limits. The waste dilution, 6.8%, was estimated by surveying after the mining operations. Grades have been accordingly adjusted before parameter estimation. When we examine the drill core data, there are unequal metres of drill core that represent the 70 data bits, as shown in Appendix 8, page 219. If we take the approach by weighting the drill core bits by meterage, as was logged in the drill core log, a better estimate of grade is obtained:

Table 54 - Drill Core Data by Meterage

Hole	Length, m	% Ni	Diluted % Ni
L4412-2	4.36	2.37	2.22
L4412-4A	7.50	2.85	2.67
W975-08	7.30	3.36	3.15
W987-08	11.80	3.32	3.11
W3800-9	6.00	3.46	3.24
W1265	5.50	3.87	3.62
CH412-02	7.00	2.39	2.24
I1337-2	11.00	1.63	1.53
S3915-4	15.80	6.66	6.24
I1350-2	9.30	2.63	2.46
718-193	9.71	2.63	2.46
E650-03	17.30	2.58	2.42

Table 55 - Drill Core Data by Meterage : Krige Uncorrected Geomean

In Naperian Log Transform		**After Inverse Transform**	
Logarithmic Mean	1.023	Mean % Ni	2.78
Log Standard Deviation	0.341	Standard Deviation	1.41
Upper Confidence Limit	1.70	Upper Confidence Limit	5.50
Lower Confidence Limit	0.340	Lower Confidence Limit	1.41

This amounts to an uncorrected geometric mean grade of 2.78% Ni, with an arithmetic mean of 2.95% Ni. The arithmetic mean has limits of 5.30 and 0.59% Ni at the 95% level. The Krige (skewed) limits are different to those of the (symmetrical) arithmetic mean, and reflect the individual measurements. The inadequate arithmetic estimator in this case is the set of confidence limits, not the sample mean.

Table 56 - Sichel t-Estimator Results - Raglan Survey November 2003
Weighted Mean of Drill Core Grades

Parameter	**Result**
Uncorrected Geomean	**2.78**
Correction Factor	**1.054**
Corrected Geomean	2.93
Upper Confidence Limit Coefficient	1.33
Upper Confidence Limit (97.5%)	3.91
Lower Confidence Limit Coefficient	0.843
Lower Confidence Limit (2.5%)	2.47

DataMine Model

The actual ore mined was logged, and grade estimates from each block mined were obtained from the DataMine database, which produces Kriged estimates of grade for the official ore reserve of the mine. From these two information sources, another independent estimate of the ore actually mined, accounting for the 6.4% dilution as was the case in the Drill Core Estimate, is obtained. The results are shown in Table 56.

Table 57 - Extract from Datamine

Source	Rounds	Tonnes	Grade % Ni	Tonnes Ni
Q1345-1	32	196	5.30	10.39
LL1330	11-15	1408	2.48	34.92
W1265	8-9	1455	3.76	54.71
S1370	17-1 and 2	847	5.50	46.59
Q1350 2E	5-6	844	3.64	30.72
I1510	10-11	1189	2.08	24.73
Champagne		3100	2.26	70.06
Zone C	1-14	1907	2.46	46.91
Zone I		1648	1.76	29.00
Total		**12594**	**2.76**	**348.03**

Actual Ore Milled

Shift tonnage and flotation feed grade for the actual ore milled between the 16th and 20th November 2003 are given in Table 58.

Table 58 - Actual Ore Milled*

of Future Ore at Raglan Operations 16-19 November 2004
[Data Source : Mill Operations]

Date	Shift	Tonnes	Grade % Ni	Date	Shift	Tonnes	Grade % Ni
16	Night	1318	2.83	18	Night	1397	2.78
17	Day	1302	2.75	19	Day	1377	2.73
	Night	1386	3.00		Night	1410	2.87
18	Day	1389	2.48	20	Day	1374	2.92

*The weighted mean of the total ore milled is 2.79% Ni. Total ore milled as 10 953 tonnes.

Actual Survey Units

Table 59 shows the grades of flotation circuit feed for the six statistical benchmark survey units taken surveyed between the 16th and 19th November 2003.

Table 59 - Actual Ore Surveyed

of Future Ore at Raglan Operations 16-20 November 2004
[Data Source : Statistical Benchmark Survey]

Date	% Ni	Date	% Ni
17	2.70	18	2.65
18	2.54	19	3.02
	2.57	20	3.20
Mean	**2.78**		

The arithmetic mean of the six survey units is 2.78% Ni. The associated confidence limits from the Normal Distribution, using the sample standard deviation, are 3.31 and 2.24% Ni at the 95% level. Use of the Sichel t-estimator on the same data produces t= 2.78% Ni with upper and lower limits of 3.20 and 2.59% Ni respectively at the 95% level. Exclusion of potential outliers on this basis shortens the data set to 4 observations (2.70; 2.65; 3.02 and 3.20% Ni). The Sichel t-estimate of the shortened data set is then 2.89% Ni.

Comparison of Results

The various grade estimates are summarised and compared in Table 60.

Table 60 - Summary of Independent Grade Estimates from Survey

Basis of Estimate	**Ore Grade Estimated % Ni**	
	Arithmetic	**Geometric**
Ore Mined	2.75	
Drill Core Data, Krige 1962		2.78
Sichel t, Sichel 1966		2.93
DataMine (Kriged Block Model)		2.76
Actual Ore Milled	2.80	
Surveys (Arithmetic Mean)	2.78	
Surveys (Sichel t estimator)		2.89

The sample mean of these estimates is thus (2.75+2.78+2.93+2.76+2.80+2.78 +2.89)/5 = 2.81% Ni. The question arises as to the existence of any significant difference between these mean values. Using the sample mean of 2.81% as a basis, the largest difference is with the Sichel drill core estimate of 2.93%, a difference of 0.12% Ni (absolute) or 4.27% relative. The proposed selection of reference distributions for the grade of ore in the survey will be discussed later.

Final Concentrator Products

In this case, no existing operations records exist for this ore mix other than the 8 consecutive shifts of campaign milling, and the six survey units taken from within that time. A more limited comparison will be made on this basis.

Final Concentrate

The 8 consecutive shifts of milling operations produced final concentrate as shown in Table 61.

Table 61 - Actual Final Concentrate Produced
From Future Ore at Raglan Operations 16-20 November 2004
[Data Source : Production Operations]

Date	Shift	Tonnes	Grade % Ni
16	Night	171	17.9
17	Day	163	19.2
	Night	185	19.6
18	Day	163	18.3
	Night	180	18.5
19	Day	173	18.5
	Night	195	18.0
20	Day	192	18.4
Total		1 422	
Mean		**177.8**	**18.55**

The Final Concentrate sampled during the survey is shown in Table 62.

Table 62 - Actual Final Concentrate Surveyed

From Future Ore at Raglan Operations 16-20 November 2004
[Data Source : Mine to Mill Survey]

Date	Grade % Ni	Date	Grade % Ni
17	19.0	19	17.7
18	16.6	20	18.0
	17.8		
	17.5		
Mean	**17.76**		

Since it has been established that the Final Concentrate nickel values are normally distributed, conventional statistics may use the data in Table 47 to estimate the sample standard deviation, and construct confidence limits. These calculations amount to s = 0.58% Ni with UCL = 19.72 and LCL = 17.42% Ni. None of the individual surveyed concentrate grades fall outside these limits. A Pooled Estimate of Variance was constructed to develop a t-test for potential significance of the two sample means (Eqs. 3-5):

n_1 = 8 (Operations)

n_2 = 6 (Surveys)

s_1^2 = 0.331

s_2^2 = 0.603

However, the two surveys of the 18th November were disqualified by the Sichel t-estimator on the basis of feed grade being outside the 95% confidence limits. The corresponding concentrates must therefore also be disqualified. This reduces the concentrate survey set to (19.0; 17.5;17.7;and 18.0% Ni) averaging 18.05% Ni.

Final Tailings

Table 63 - Actual Final Tailings Produced

From Future Ore at Raglan Operations 16-20 November 2004
[Data Source : Production Operations]

Date	Shift	Tonnes	Grade % Ni
16	Night	1147	0.59
17	Day	1139	0.40
	Night	1201	0.44
18	Day	1226	**0.36**
	Night	1216	0.44
19	Day	1205	0.47
	Night	1215	0.44
20	Day	1182	0.40
Total		9 531	
Mean		**1191.4**	**0.44**

Use of Sichel's t-estimator for small data sets estimates t = 0.44% Ni with UCL = 0.52 and LCL = 0.40% Ni. The Final Tailings sampled during the survey is shown in Table 64.

Table 64 - Actual Final Tailings Surveyed

From Future Ore at Raglan Operations 16-20 November 2004
[Data Source : Mine to Mill Survey]

Date	Grade % Ni	Date	Grade % Ni
17	0.30	19	0.48
18	0.41	20	0.42
	0.40		
	0.44		
Mean	**0.41**		

The sample mean and associated confidence limits of the Final Tailings produced (Table 60 : Operations Results), estimated above, would question the survey of the 17th November at 0.30 % Ni. Rejection of this as well as the first two surveys of the 18th November (on the basis of their feed grades being below the acceptance limits) produces a survey composite short average of 0.45% Ni.

There is now close alignment of the accepted mean grades of nickel in the Final Tailings; the adjudicated External Reference Distribution was 0.44% Ni, and the adjudicated survey composite was 0.45% Ni.

Reworking of the External Reference Distribution of Final Tailings grades (Table 63) using Krige 1962, produced a geomean of 0.44% Ni with upper and lower confidence limits of 0.59 and 0.33% Ni. Clearly these limits are wider than those by Sichel 1966 and are of no use in this exercise.

Tests of Significance

Appropriate tests of significance should be developed and applied to the grade estimates produced by the Mine to Mill survey. The reader is referred to Table 59, page 148, for the summary of all grade measurements. The objective is to find a method of testing the sample mean of the accepted survey unit feed grades within limits that are determined from independent means.

<u>Ore Milled</u>

In Table 59, the sample mean values of ore grade as % Ni show a close agreement. These means have been separated into a first group drawn from arithmetical means, and a second group drawn from geometric means. These two subsets produce averages of 2.78 and 2.84% Ni respectively, a small difference of 0.06% Ni or 2.2% relative difference.

The Sichel t estimate of the accepted two-hour survey units was 2.89% Ni. This is the number which has to be compared to independent limits.

One independent estimate might be the small distribution of shift-level feed grades taken across the week of the survey. These measurements were reported in Table 57, page 147. Sichel's t-estimate for these data is t = 2.80, upper and lower limits: 97.5%: 2.98 % Ni, 2.5%: 2.70% Ni. The accepted survey average of 2.89% Ni (again by Sichel t) is 2.89% Ni, and falls within these limits. On this basis, the survey mean of 2.89% Ni is accepted at the 95% level of confidence.

Another independent estimate might be the Sichel t estimate of the weighted mean grades from drill-core. These results are shown in Table 55, page 146. The parameters from this independent estimate are : t: 2.93% ni; upper and lower limits: 97.5%: 3.91% Ni; 2.5%: 2.47% Ni. Clearly the accepted survey mean of 2.89% Ni falls within these limits. On this basis, the survey mean is accepted at the 95% level of confidence.

It is proposed, therefore, that the Sichel t-estimator be adopted as the reference distribution basis for the Campaign survey model.

Findings from Campaign Survey Model

The foregoing exercises with Raglan operations in the Campaign Survey have tested the idea of a different type of concentrator survey format. In such a case the specific objectives are different from those of the Statistical Benchmark Survey, in that the Campaign model describes the concentrator performance for a specific ore mixture rather than 'typical ore milled'. The following may be concluded:

1. Lognormality is present to a measurable extent in drill-core. A weighted mean of drill intersections by meterage and nickel grade leads into either the uncorrected geomean estimate of Krige ot the corrected geomean estimate of Sichel.
2. The above grade estimation is corrected again for waste dilution, this estimate being obtained by surveying after actual mining has occurred.
3. Good agreement is obtained between the actual ore milled, the actual ore surveyed, and the above geomean estimates. This is after outlier rejection in the six survey units by Sichel's model. The order of the differences between the various geomean estimates is +3.16 to – 2.82% relative to the overall mean.
4. The Sichel t-estimator is proposed to be the most suitable common platform to use with the reference distributions.

CHAPTER 5 - DISCUSSION

5.1. Use of Tests Based on the Normal Distribution

The first approach in this study was to assume a Normal Distribution and gain the associated simplicity. It has been shown that the only distribution that demonstrated compatibility with this approach was that of the Final Concentrate. The Final Concentrate result could be explained by the non-natural process whereby the concentrate is produced, *viz.* a process targeted at producing a tight range of nickel concentrate grades.

The Grubbs outlier model failed to detect any outliers in the trial data. A possible explanation is that the Grubbs model is based on the Normal Distribution with symmetrical associated limits. Since the Rougher Float Feed contains the effects of residual lognormality, the Grubbs model overestimates the associated limits due to the larger variance in the data, thus 'includes' all of the observations in the accepted data set. Another reason may be that the Grubbs model is based on false positive protocols of 5%, which is too restrictive.

5.2. Use of Applicable Lognormal Distribution

Characterisation of the distribution of paymetals in space, using drill-core data, confirmed that nickel grade was lognormally distributed. Extension of this exercise into the Rougher Float Feed showed that sufficient residual lognormality was present to warrant parameter estimation through the Naperian log transform. This showed that the associated confidence limits were very different to their untransformed counterparts using the Normal Distribution. In this regard the proposal by **Krige, 1962**, proved to be most relevant. The estimation of mean value was relatively unaffected in this process, probably because of the lower level of lognormality. However, the associated confidence limits proved to be very different to those of the (symmetrical) arithmetic mean. For drill-core and

Rougher Float Feed, the following two figures summarise the transition from full spatial lognormality to residual lognormality.

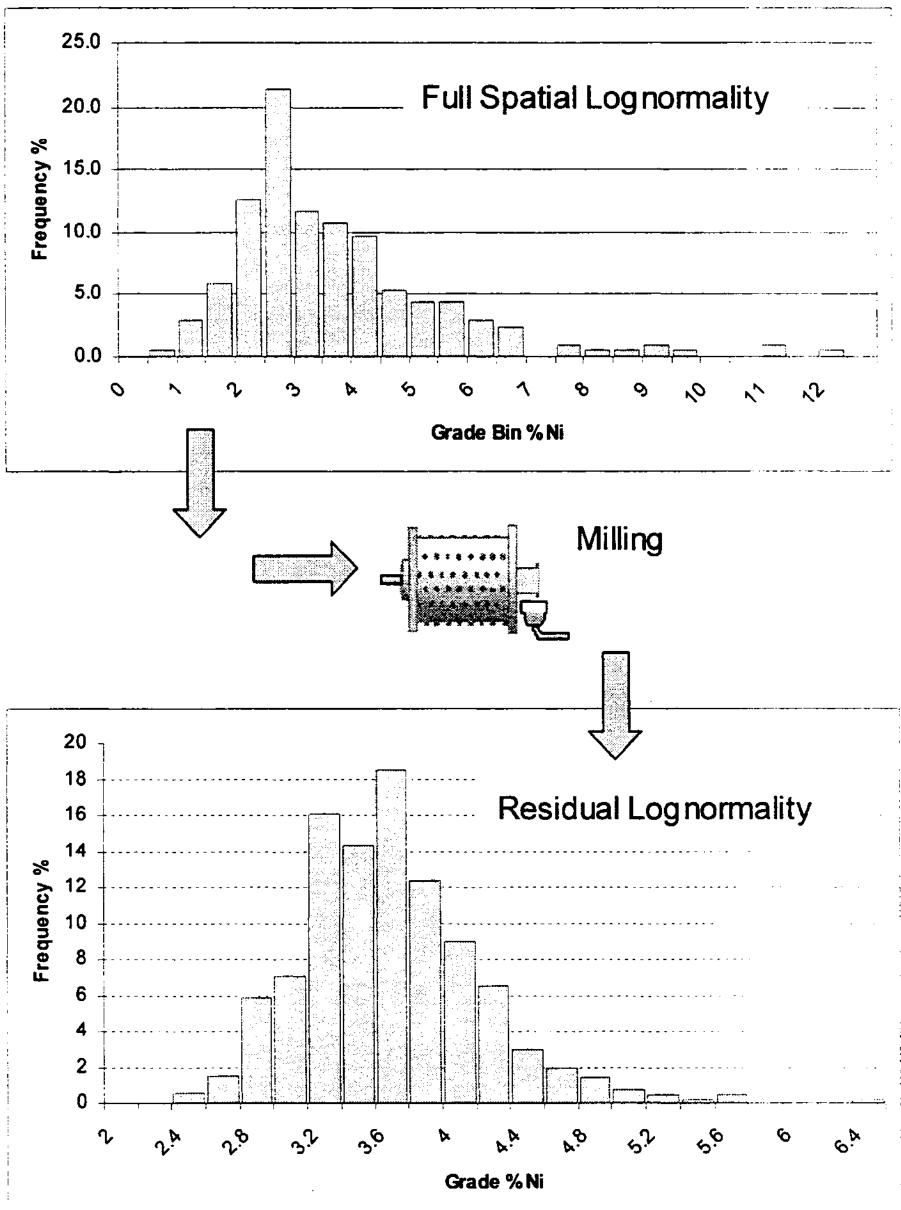


Figure 33 - Transition from Full to Residual Lognormality (Histogram)

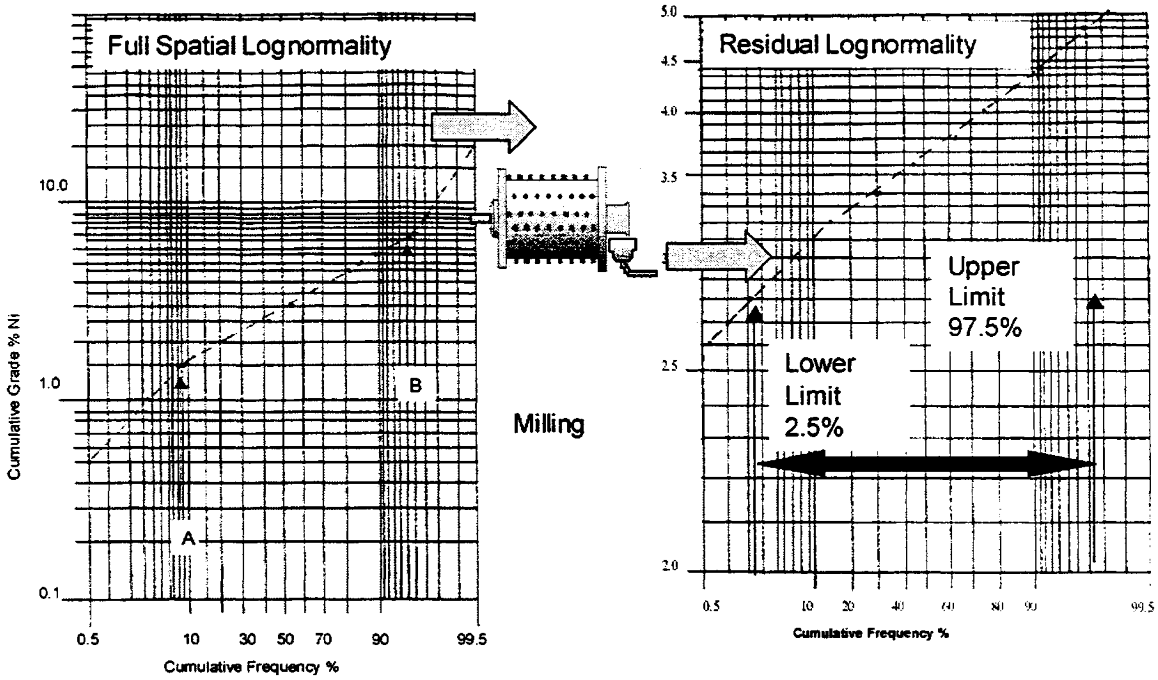


Figure 34 - Transition from Full to Residual Lognormality (Probability Plot)

Figures 33 and 34 show the reduction in the range of nickel grades from the spatial to the temporal dimension. In the drill-core data, the range of nickel grades observed was 1.0 to 11.5% Ni. In the Rougher Float Feed data, the range of nickel grades observed was 2.6 to 6.6% Ni. In Figure 34, the probability plots show the range of acceptable data from the reference distribution by overlaying marks at the 2.5 and 97.5% cumulative frequency points. The arrow inbetween indicates the acceptable range. This is a simple way of showing what is meant by the acceptance limits of the distribution. Another difference between full and residual lognormality is the scale of metal grades. In full lognormality, the scale is across orders of magnitude, i.e. 0.1, 1, 10, and 100% Ni. In residual lognormality, the range is closer and is confined within one order of magnitude.

The Final Tailings demonstrated some interesting residual lognormality. This aspect will be further discussed under compound distributions. It is counter-intuitive to expect that an orebody carrying full spatial lognormality could completely transform into a set of Normal Distributions simply as a result of physical milling.

The question arises as to whether this finding, *i.e.* of residual and compound lognormality in concentrator data, is generic. During this study, other sample data from other concentrators have been gathered. At the Strathcona Mill near Sudbury, for example, an equivalent copper/nickel ore is treated in a milling and flotation process. It is beyond the scope of this thesis to fully describe the flotation feed, copper concentrate, nickel concentrate, pyrrhotite tailings and scavenger tailings. That would otherwise amount to an exercise of similar length to the present one. Sufficient for the purposes of this discussion to state that the Strathcona Mill is a more mature operation, having started in the early 70's (Raglan started up in January 98), so the Strathcona efficiencies are better developed as a result of three decades of process improvement. A summary of the Strathcona feed grade distribution follows in Table 65. The data block size is 939 days.

Table 65 – Strathcona Rougher Flotation Feed

Grade Bin % Ni	Frequency %	Grade Bin % Ni	Frequency %
1	0	2.1	0.85
1.1	0.36	2.2	0.12
1.2	3.63	2.3	0.12
1.3	14.29	2.4	0
1.4	17.31	2.5	0
1.5	24.21	2.6	0.48
1.6	16.59	2.7	0.12
1.7	10.53	2.8	0.24
1.8	7.63	2.9	0
1.9	2.30	3.0	0
2.0	1.21		

These values show a residual lognormality, as shown in Figure 35. In the next section, a similar exercise for the tailings data will be presented.

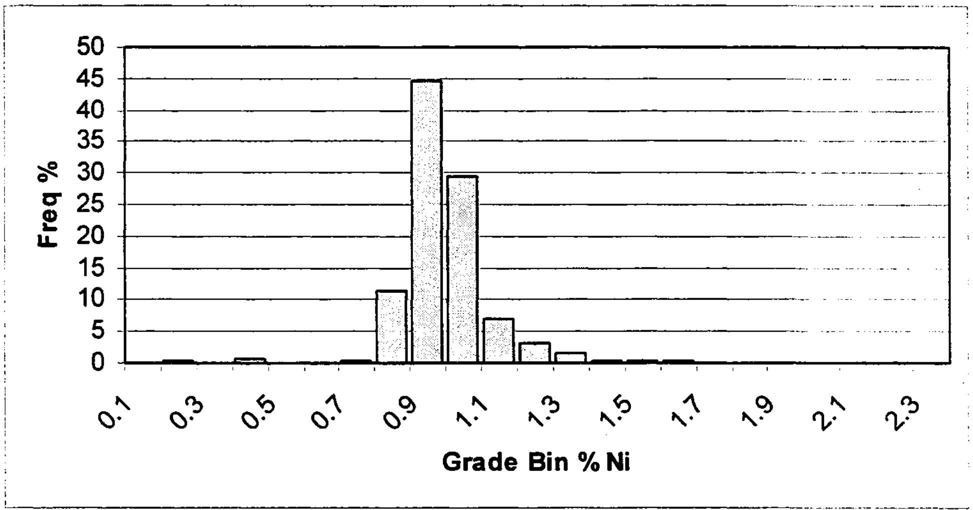


Figure 35 - Strathcona Rougher Flotation Feed - Histogram

5.3. Composite Distributions

Ore Grade

The geological review of the Raglan End-Member set proved to be a key foundation. This system is dominated by a range of three textures that are associated with equivalent ranges of nickel (and copper) grade. It is thus logical to expect a compound distribution in the spatial model. This was demonstrated, and TRIPOD was used to estimate the probable domains. In this regard the published work and software by Clark was of limited value. Where these interpretations from TRIPOD may have been cogent, it is probable that the domains of the subdistributions are loosely estimated and do not strictly correspond to the End-Member domains.

It is further important to note that the assessment of Rougher Float Feed nickel grade as a Reference Distribution brought out the value of recognising residual lognormality. Table 66 shows that the estimation of the mean is not significantly misled by using the arithmetic mean, however the associated confidence limits, in particular the lower limit, are different when estimated off a lognormal platform.

Table 66 - Rougher Float Feed Parameter Estimation
Raglan June 1998

Basis of Estimate	Sample Mean	Upper Limit (97.5%)	Lower Limit (2.5%)
Arithmetic	3.13	3.82	2.44
Lognormal	3.11	3.86	2.51

Final Tailings

The modelling of the lognormal bimodal distribution of nickel in Final Tailings drew some tentative conclusions. Two hypotheses develop as a result of this observation.

In the first hypothesis, called the *Particle Grade Model*, for which the further work is outside the scope of this thesis, the bimodality exists because these tailings contain a mixture of two different orders of magnitude of nickel concentration in the mineralisation. It is probable that further work to understand this will lead to a predictive quantitative model. It is argued that, because the mineral separation process is designed to recover most of the high-grade pentlandite into a concentrate by flotation, the tailings should contain mostly the low-grade silicate minerals (with much lower nickel concentration) and some unrecovered pentlandite. This is because flotation is an imperfect process. If this is the case, then a bimodal distribution will result. It is proposed that the lognormal probability plot in Figure 29 may be used as a monitoring model for the estimation of recoverable paymetal in Final Tailings. As the concentrator improves performance in year-on-year technical programmes, the amount of recoverable paymetal in Final Tailings should steadily reduce. The implications of Figure 29 are that the Raglan concentrator has done just that since commissioning in 1998. Reference to Qem*SCAN survey data (Table 30) did show that these tailings contained 88.12% silicates, 4.0% silicate-sulphide textures, and 7.88% sulphides, of which 0.84% was pure nickel sulphides. Reference to the Raglan Primary Composition File (Table 2) shows that the nickel sulphides range from 18.4 to 64.7% nickel, whilst the silicates range from 0.1 to 1.0% nickel. In the second hypothesis, called the *Liberation Model*, finer grinding would be needed to liberate fine-grained pentlandite from serpentine textures before such pentlandite could be recovered. In order to test this idea in another nickel/copper operation, an equivalent data extract was made for the scavenger tailings at the Strathcona mill in Sudbury. These results are shown in Table 67 and as a histogram in Figure 36.

Table 67 - Strathcona Scavenger Tailings

Grade Bin % Ni	Frequency %	Grade Bin % Ni	Frequency %
0.06	0	0.18	7.5
0.08	0.3	0.2	2.3
0.1	5.9	0.22	1
0.12	25.7	0.24	1
0.14	35.5	0.26	0
0.16	20.5	0.28	0
		0.3	0.3

The residual bimodal lognormality of these data is borne out by their lognormal probability plot in Figure 37.

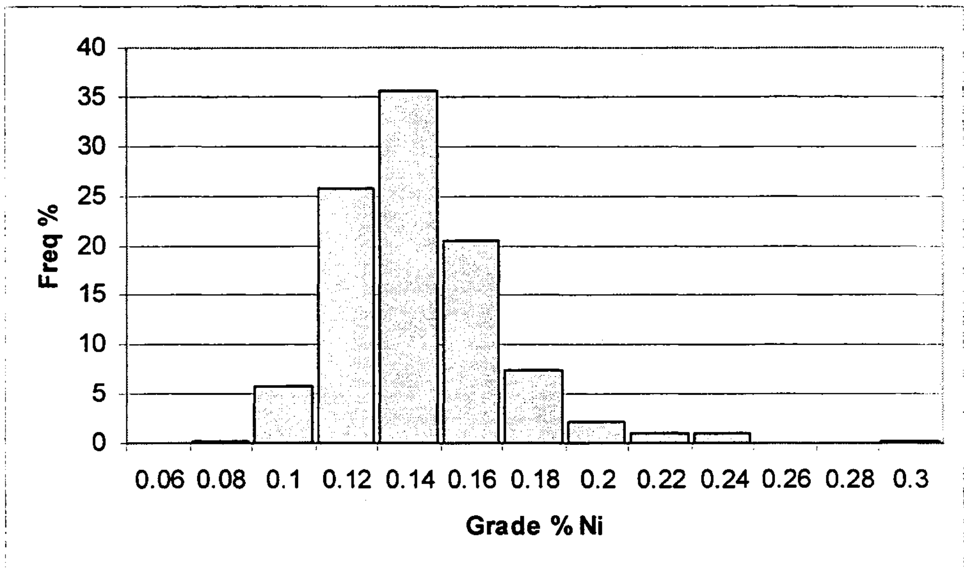


Figure 36 - Strathcona Scavenger Tailings – Histogram

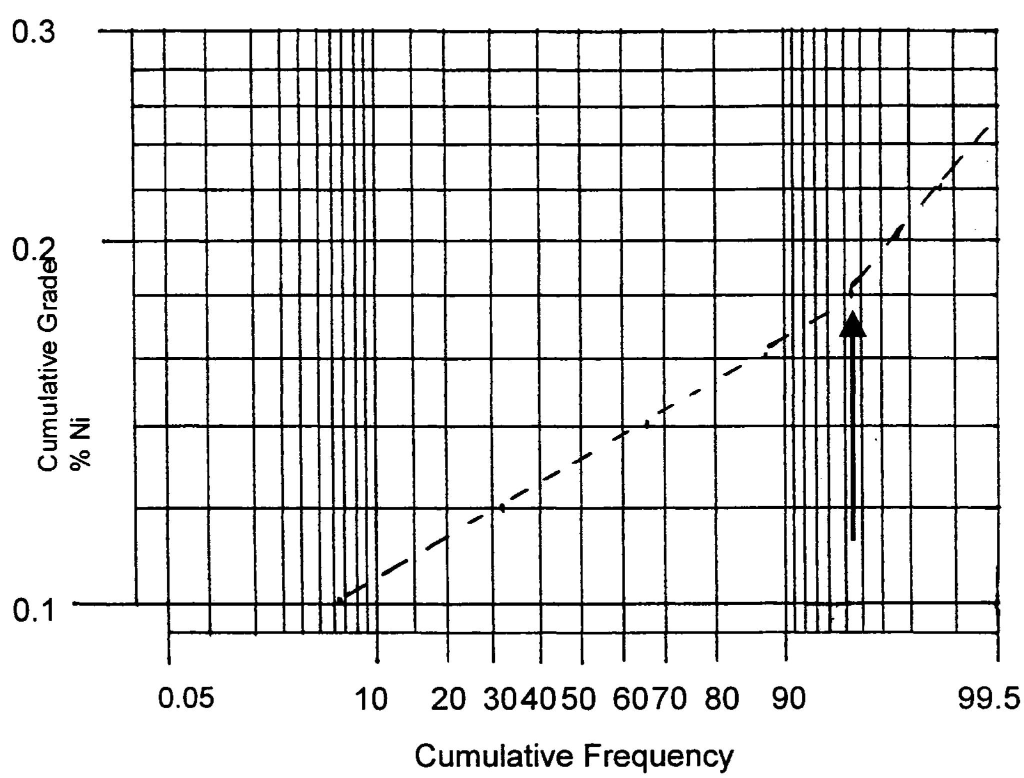


Figure 37 - Strathcona Scavenger Tailings – Probability Plot

Figure 37 shows a clear bimodal distribution with an inflection at 95.5% cumulative frequency. This point of inflection is different to that of the Raglan Final Tailings, in that it is further to the right of the distribution mean. This might be because the Strathcona operation is more mature than that of the Raglan **[Lotter and Tuzun, 1999]**. Further work to broaden this database is indicated. It is accordingly suggested that the lognormal bimodality reported for the Raglan tailings is not confined to that operation; rather, it is also to be found at the Strathcona operation near Sudbury.

5.4. Reference Distributions

An extract of the Rougher Float Feed nickel data for the three-month Krige External Reference Distribution was made and examined. If the purpose of the Reference Distributions is to obtain a composite survey (Internal Reference Distribution) mean that is comparable or similar to the ore that is 'typically milled' in the operation, then a series of simple cross-checks can be made. The reader is reminded that, in terms of residual lognormality, the estimation of the sample mean is relatively indifferent to arithmetic or geometric forms; it is the associated confidence limits that are sensitive to this choice. The actual survey dates for the 1998 survey were 18-29 June 1998. Reference to Appendix 4 shows that the actual continuous operations at Raglan recorded the Rougher Float Feed grade data as shown in Table 68.

Table 68 - Raglan Continuous Operations 18-29 June 1998
Actual Rougher Float Feed Data : Nickel

Date	Grade % Ni	Date	Grade % Ni
18	2.84	24	3.30
19	2.67	25	3.17
20	2.76	26	3.45
21	2.80	27	3.38
22	2.94	28	3.91
23	2.89	29	3.69

These data show an arithmetic mean of 3.15% Nickel. The extended data set for June operations averaged 3.13% Nickel. At first glance, it seems that the survey period did have an ore grade quite close to that of the whole month. The raw data of the six survey units taken during the period (Table 69) follows:

Table 69 - Internal Reference Distribution Raglan Survey 1998

.Survey No.	Grade of Rougher Float Feed, % Ni
1	3.05
2	2.82
3	2.57
4	2.63
5	3.38
6	3.28

These data average only 2.96% Nickel. It was shown that use of the Sichel t-estimator at the 95% confidence level (with upper and lower limits of 3.50 and 2.71% Ni respectively) rejected surveys 3 and 4 as being too low in grade and below the lower confidence limit. The Krige External Reference Distribution drew the same conclusion when constructed on the basis of a three-month operations history. The question arises as to what minimum size of External Reference Distribution is necessary in order to robustly estimate the associated confidence limits; it has already been shown that for Rougher Float Feed, large data sets provide comparable means from either the arithmetic mean or the uncorrected geomean. It is the skewed confidence limits that are more important in this context. An exercise was conducted on the May-July 1998 data to characterise the estimation of the upper and lower skewed confidence limits as the size of n was steadily increased. The results are summarised in Table 70.

Table 70 - Raglan External Reference Distribution

Estimated Geomean and Sample Confidence Limits for the May-July 1998 Using Krige, 1962

Size of n	Estimated Geomean	Estimated Upper Limit	Estimated Lower Limit
10	3.28	4.05	2.66
20	3.33	4.14	2.67
30	3.30	3.99	2.72
40	3.24	3.93	2.67
50	3.16	3.87	2.58
60	3.20	3.92	2.62
70	3.20	3.88	2.63
80	3.23	3.92	2.66

The overall result of all the data were reported in Table 41 as geomean : 3.25% Nickel, Upper Limit 3.98 and Lower Limit: 2.66% Nickel. From this analysis, both limits seem to stabilise from n=40 onwards, whilst the lower limit stabilises at an equivalent value of n. The same data base was used to audit the efficiency of parameter estimation using the Sichel t – estimator. In this case, because the estimator is designed for small data sets, the value of n here studied was 3 to 20. Table 71 summarises the outcomes of this exercise.

Table 71 - Parameter Estimation by Sichel t-Estimator

Raglan Rougher Float Feed May-July 1998 and Comparison with Arithmetic Estimates
(Arithmetic Limits are for the Sample Space, but the Sichel Limits are for the Population Mean)

Size of n	Arithmetic Mean	Estimated Corrected Geomean	Arithmetic Upper Limit	Estimated Sichel Upper Limit	Arithmetic Lower Limit	Estimated Sichel Lower Limit
3	2.92	3.24	3.53	3.49	2.31	3.11
4	3.06	3.06	3.67	4.28	2.45	2.67
5	3.15	3.15	3.76	4.01	2.54	2.82
6	3.22	3.20	3.83	3.86	2.60	2.92
7	3.25	3.24	3.86	3.77	2.64	2.99
8	3.26	3.25	3.87	3.68	2.65	3.03
9	3.27	3.26	3.88	3.63	2.66	3.07
10	3.30	3.29	3.91	3.61	2.69	3.11
11	3.30	3.29	3.91	3.58	2.69	3.13
12	3.32	3.31	3.93	3.57	2.70	3.15
13	3.35	3.35	3.96	3.61	2.74	3.19
14	3.32	3.31	3.93	3.57	2.71	3.16
15	3.32	3.32	3.93	3.55	2.71	3.17
16	3.31	3.30	3.92	3.52	2.70	3.17
17	3.28	3.27	3.89	3.49	2.67	3.13
18	3.32	3.31	3.93	3.55	2.71	3.17
19	3.34	3.33	3.95	3.56	2.73	3.19
20	3.34	3.34	3.95	3.55	2.73	3.20

Inspection of Table 70 shows that the arithmetic mean and the corrected geomean are in good agreement. However, the arithmetic lower limit is consistently lower than that of the Sichel model. This difference is in the order of (3.08 – 2.65) = 0.43% Ni or 13.1% relative. The equivalent comparison of the upper limit shows that the arithmetic estimate is higher, on the average (3.87 – 3.66) = 0.21% Ni or 6.4% relative (using the arithmetic mean of 3.26% Ni as the 100% basis for these calculations). Comparison of Tables 69 and 70 immediately suggests that the Sichel t-estimator produces tighter associated confidence limits than does the Krige model. Table 72 compares selected parameter estimates:

Table 72 - Parameter Estimation by Sichel t-Estimator and Krige
Raglan Rougher Float Feed May-July 1998

Size of n	Type	Estimated Geomean	Estimated Upper Limit	Estimated Lower Limit
10	Sichel	3.29	3.61	3.11
	Krige	3.28	4.05	2.66
20	Sichel	3.34	3.55	3.20
	Krige	3.33	4.14	2.67

The Sichel t-estimator was written specifically for small data sets that are lognormally distributed, with specialised models for the robust estimation of mean and associated confidence limits. These confidence limits in the Sichel model are for the t-distribution of the mean and not the sample space. The Krige model was also written for lognormal data sets, but for large n. In the Krige model, the confidence limits are those for the sample space, and not for the population mean. This is the fundamental explanation for the narrower confidence limits of Sichel. Note that by either model the estimation of the mean is very similar.

5.5. Outlier Rejection Models

Internal Reference Distribution

As abovementioned, the symmetrical approach of Grubbs failed to identify any outliers in the residual lognormal distribution of survey units in the Raglan 1998 survey. Use of the Sichel t-estimator at the 95% associated confidence limits proved to be more appropriate. These limits were upper: 3.50 and lower: 2.72% Ni. *In the Sichel t-estimator, these limits are of the population mean.* Two low-grade data points were found to lie outside the lower 95% confidence limit. The shortened sample mean after outlier rejection agreed closely with the robust mean of operations feed grades for that month. This exercise is shown in Figure 38.

For further confirmation, use of the robust Krige External Reference Distribution to estimate associated confidence limits to check these two outliers was successful. In this case, exactly the same outliers were identified and rejected. The agreement of the shortened Internal Reference Distribution with the operations feed grade mean of that month. Upper Limit at 97.5%: 3.98% Ni; Lower Limit at 2.5% : 2.66% Ni.

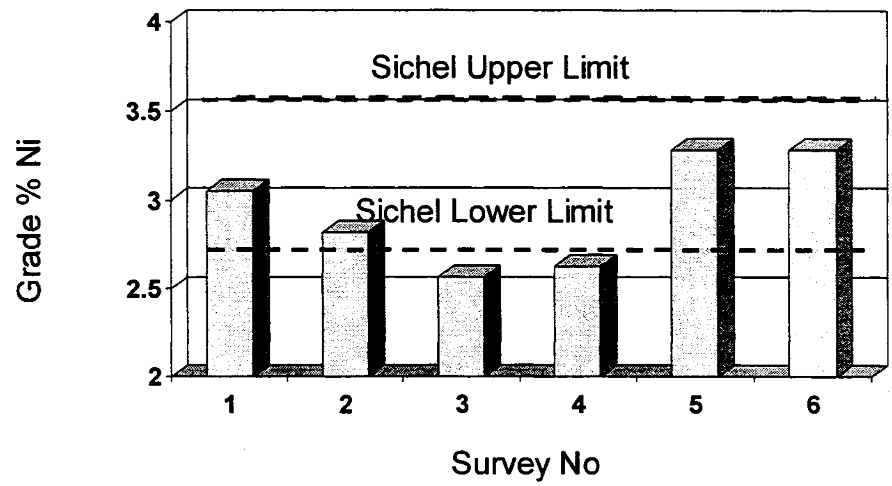


Figure 38 - Inspection of Individual Surveys by the Sichel t-Estimator

The remaining four surveys are now inspected by the External Reference Distribution using the Krige method of 1962, with skewed confidence limits at the 95% level. This exercise is shown in Figure 39. The upper and lower limits of the Krige reference distribution are 3.98 and 2.66% respectively. Although the Sichel distribution had already identified and rejected surveys 3 and 4 (at 2.57 and 2.63% Ni respectively), the Krige distribution would have done the same. It is, however, obvious that the Sichel estimator has tighter limits than that of Krige. By either means, the shortened survey mean is now (3.05+2.82+3.38+3.28)/4=3.13%. None of these data rejections would have been obvious in the Normal Distribution systems. The Normal Distribution thus overestimates the limits. Further, these limits are symmetrical.

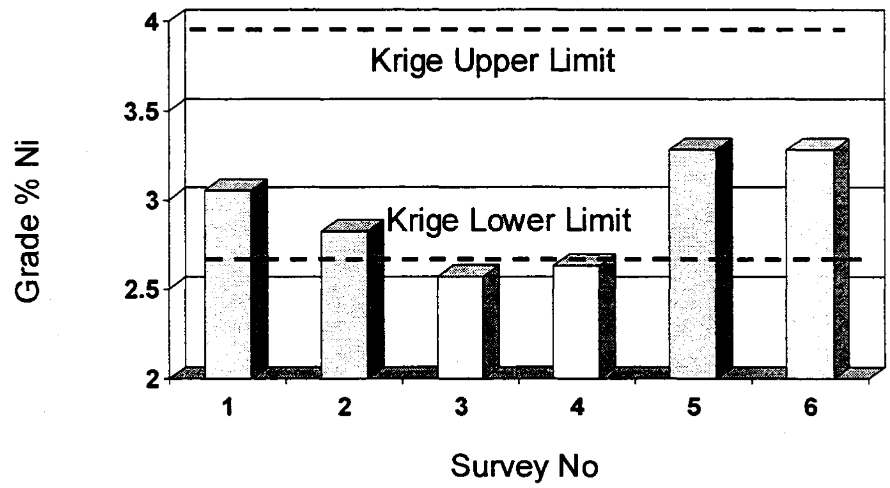


Figure 39 - Inspection of Surveys by Krige
External Reference Distribution (Surveys Rejected by Sichel are in hatched legend)

5.6. Auto-Correlation in Time

The use of the semivariogram in time to identify the minimum spacing of surveys, so as to break any effects of auto-correlation in time, has been demonstrated. Neither the 1998 and 2003 surveys for Raglan had taken this factor into account. Instead, the team had taken opportunistic surveys back-to-back under favourable steady state conditions, in the belief that they were being efficient and productive, 'getting the job done' in as short a time as possible. This has resulted in the new proposed quality control systems identifying these observations in the Internal Reference Distribution as outliers. When the percentage outlier rejection rates are calculated for each of these surveys, it is found that more than 5% of raw observations have been rejected:

Table 73 - Rejection Rate of Observations from Surveys

Survey	Internal Reference Distribution Total Observations	Total Observations Rejected	Rejection Rate %
Raglan June 1998	6	2	33.3
Raglan November 2003	6	2	33.3

It is accordingly recommended that future statistical benchmark surveying take the spacing of two-hour survey units into account by using the semivariogram.

5.7. Sensitivity to Ore Grade

The distinction between acceptance limits at the 95% confidence level, and confidence limits in the sample mean at the 95% confidence level, must be made. Most of this work has focused on the former. By writing a reference distribution system that produces a mean estimate of grade of ore typically milled at the operation, and by using the robust associated acceptance skew limits at the 95% level, one will be assured that the inspection of the Internal Reference Distribution will reject individual surveys that will not contribute meaningfully to a sample mean grade in the overall survey which is close to the mean of the External Reference Distribution. The term 'meaningfully', in this context, means 'relevant observation that adds value to the set of measurements'. In terms of the expected rate of rejection of individual surveys, the difference between the foundations of the External and Internal Reference Distributions should be noted. Because the Internal Reference Distribution is based on increments of 2 hours, and the External, on increments of 12 hours, there is a difference in the level of sampled variance. The Internal Reference Distribution will show a larger variance than the External. Also, use of the confidence limits of the population mean lead to a rate of rejection from the Internal Reference Distribution larger than 5%.

5.8. Campaign Model

Overestimation of the ore grade as drill-core, by the arithmetic mean, is some 5% (Table 52). It is therefore appropriate to use the lognormal form for this estimation. Further, the relevance of Krige's lognormal distribution as an outlier rejection system is again demonstrated by the rejection of three data points. Use of the Datamine model in a sense is redundant since this software was using the data already referred to in drill core. The mining estimate 'ore mined' falls into the same category. The actual ore milled during the period 16-19 November 2003 is in the domain of residual lognormality. Accordingly the Sichel t-estimator can be used to estimate the parameters. This resulted in a corrected geomean of 2.80% Ni with upper and lower limits of 2.98 and 2.70% Ni at the 97.5 and 2.5% tail areas respectively. Inspection of Table 55 (copied below as Table 74) shows that one shift, the day shift of the 18th November, reported a low ore grade at 2.48% Ni.

Table 74 - Actual Ore Milled
of Future Ore at Raglan Operations 16-19 November 2004 [Data Source : Mill Operations]

Date	Shift	Tonnes	Grade % Ni
16	Night	1318	2.83
17	Day	1302	2.75
	Night	1386	3.00
18	Day	1389	2.48
	Night	1397	2.78
19	Day	1377	2.73
	Night	1410	2.87
20	Day	1374	2.92
Total		10 953	
Mean		**1369.1**	**2.80**

Note that the Sichel t value of 2.78% Ni agrees well with the arithmetic mean value. Exclusion of the low grade observation changes the estimates to Sichel t:

2.84 % Ni, with limits at 2.97 and 2.78% Ni. The shortened arithmetic mean is now 2.84% Ni. Reference to the actual survey units during the campaign, copied below as Table 75, shows the possibility of outlier rejection using Sichel's t estimator for the internal reference distribution.

Table 75 - Actual Ore Surveyed

of Future Ore at Raglan Operations 16-20 November 2004
[Data Source : Statistical Benchmark Survey]

Date	% Ni	Date	% Ni
17	2.70	18	2.65
18	2.54	19	3.02
	2.57	20	3.20
Mean	**2.78**		

The Sichel t-estimator produced a mean of 2.78% Ni, with limits at 3.20 and 2.59% Ni. Inspection of Table 56 shows that two surveys, grading 2.54 and 2.57% Ni respectively, are rejected. This shortens the survey mean to 2.89% Ni.

Disregarding the Datamine estimate for reasons of redundancy, and the ore mined for reason of visual estimation, a comparison between three actually measured and adjudicated reference distributions is shown in Table 76.

Table 76 - Comparison of Adjudicated Reference Distributions

Campaign Model

Basis	**Grade % Ni**
Drill Core Data, Krige 1962 (Weighted Mean Basis)	2.78
Drill Core Data, Sichel 1966 (Weighted Mean Basis)	2.93
Actual Ore Milled	2.80
Accepted Surveys	2.89
Overall Mean	**2.85**

Using the approach of the *Statistical Benchmark Survey*, the aim is to produce a composite survey mean grade of ore that is close to, or not significantly different from, ore that is typically milled. This is because of the dominant influence of ore grade on metallurgical performance.

CHAPTER 6 - PROVISIONAL RECOMMENDATIONS

6.1. Statistical Benchmark Survey

The foregoing analysis of the Raglan 1998 survey will be used as a basis for a new structure of *Statistical Benchmark Surveying*. The salient features of the changes are:

1. Recognition of the residual lognormality in the distributions of feed and tailings in the concentrator,
2. Expansion of the External Reference Distribution to three months rather than one,
3. Use of Krige's Naperian logarithmic method to estimate the External Reference Distribution parameters,
4. Construction of a semivariogram on Rougher Flotation Feed to determine minimum spacing between survey units,
5. Maintenance of the Internal Reference Distribution at six two-hour survey units, and use of Sichel's t-Estimator for parameter estimation,
6. Increasing the number of cuts in each two-hour survey unit from four to twelve, thereby reducing the standard error,
7. Equivalent construction of External Reference Distributions for Final Concentrate and Tailings, respecting that the former is normally distributed,
8. Comparison of the Internal Reference Distributions of Final Concentrate and Tailings with their External counterparts.

A description of this new method now follows. This procedure is also shown in Figure 40.

Procedure : Statistical Benchmark Surveying

Survey Date and Planning

Planning, communication and teamwork form the foundation of a successful survey. This segment of the project has to be performed very carefully since it has a significant impact on the successful outcome of the survey. There are two key effects:

1. A clear plan is developed, which details the implementation of the survey,
2. Key involvement of the concentrator operations staff and associated disciplines, which achieves agreement and communication of what shall be done, when it shall be done, why, and by whom.

A two-week block of time is chosen well in advance of the survey. The survey is scheduled accordingly. The flowsheet format, reagent dosage, and treatment rates are then frozen. It is important at this stage that the mining and geology disciplines are consulted and made aware of the aims and intentions of the survey. They should be given the opportunity to provide input to the project. This is because there are secondary advantages to the type of information that will be produced from the survey. These secondary advantages are that the mining and geology disciplines can use this information for other purposes in the mining operation.

A scope of work detailing the specific objectives of the survey, the resources and associated schedules, is written, revised after consultation, and finalised. This form of survey is specifically to benchmark normal operations treating normal Run-of-Mine ore. The timing of the survey should be chosen not only with this in mind, but also so as to respect the implementation history of the last period of flowsheet changes that have been made towards better performance.

Sample List

In a detailed discussion between operations management and the survey team, a list of flowsheet samples is prepared. Beyond the obvious Rougher Float Feed, Final Concentrate and Final Tailings, intermediate streams such as Rougher Concentrate(s) and Cleaner Tailings are included. The criterion to be satisfied is that sufficient redundancy exists in the data in order to obtain a good mass and value balance.

External Reference Distribution

A continuous three-month block of concentrator operations data is chosen to surround the scheduled survey dates. Shifts, which are usually twelve-hour subsets of days, are used as the frequency level at which the External Reference Distribution data are collected. Shifts during which the concentrator operations were shut down are excluded from the data set. Also, shifts which had contaminated samples are excluded. This condition must however be verified by the operations log, for example stating that a leak in a concentrate pipe was discovered, and that concentrate slurry had entered the rougher flotation feed sampling assembly [which actually happened at Raglan in 1998]. The chief paymetal grade is used as the variable being described. In the case of Raglan, this is Nickel. On-line measurement systems should be avoided as the basis for this segment of data collection; what is preferred is the formal acid dissolution and atomic absorption or inductively coupled plasma technologies for measurement. This exercise should produce a data set of approximately 160-180 data points.

The parameters of the External Reference Distribution are then estimated using the method of **Krige, 1962**. The layout of these calculations is given in pp. 64-65, with an example in pp. 116-119. Note that the associated confidence limits are calculated in the Naperian log transform before the anti-Naperian logarithm is taken. This process derives numeric estimates of the mean, upper confidence limit, and lower confidence limit. These are recorded.

Semivariogram

A semivariogram is constructed for Rougher Flotation Feed using the method described on pp. 103-105. This determines the minimum spacing in time between the two-hour survey units.

Two-Hour Survey Units

During the two-week period of the *Statistical Benchmark Survey*, a total of six successful two-hour survey units are performed. In this context, 'successful' means that there was no operations interruption during any of the two-hour units, for example a pump tripping out (which would disturb the equilibrium of the mass and value balance). The two-hourly survey units should be arranged in a random stratified manner. Especially, the practice of back-to-back surveys is disallowed. A complete suite of flowsheet samples per the abovementioned Sample List is taken in every one of these two-hour units.

Internal Reference Distribution

A total of six two-hour survey units must be obtained prior to the construction of the Internal Reference Distribution. Priority treatment must be given to the processing of the Rougher Float Feed samples, because these will produce the information necessary to adjudicate which of the six survey units will pass the tests of the two Reference Distributions. Once the paymetal grade data are to hand from this process, (in this case Nickel), the Sichel t-Estimator is used to construct the Internal Reference Distribution. This method and example are shown in pp. 58-60 and pp. 119-120. Any data that fall outside the skewed confidence limits of the Sichel t-distribution are disqualified. The short data set then forms the first estimate of the Internal Reference Distribution.

Inspection of the Internal Reference Distribution by the External Reference Distribution

The External Reference Distribution skewed confidence limits, described above, are used to inspect each remaining datum of the Internal Reference Distribution. Those which lie within these limits are accepted into the final data set of the Internal Reference Distribution.

Final Concentrator Products

Once the above tests have been successfully performed on Rougher Flotation Feed, the reference distribution data for the Final Concentrate and Final Tailings are drawn. These are used to benchmark mean grades of concentrate and tailings for the operations month surrounding the actual survey. Internal Reference Distributions of Final Concentrate and Final Tailings are prepared from the *accepted* survey units. The sample means are compared to those from the External Reference Distributions. It is thus possible to state that the accepted survey composite represents typical grades of ore milled, final concentrate and final tailings; thus the nickel recovery will also be typical for those operating conditions.

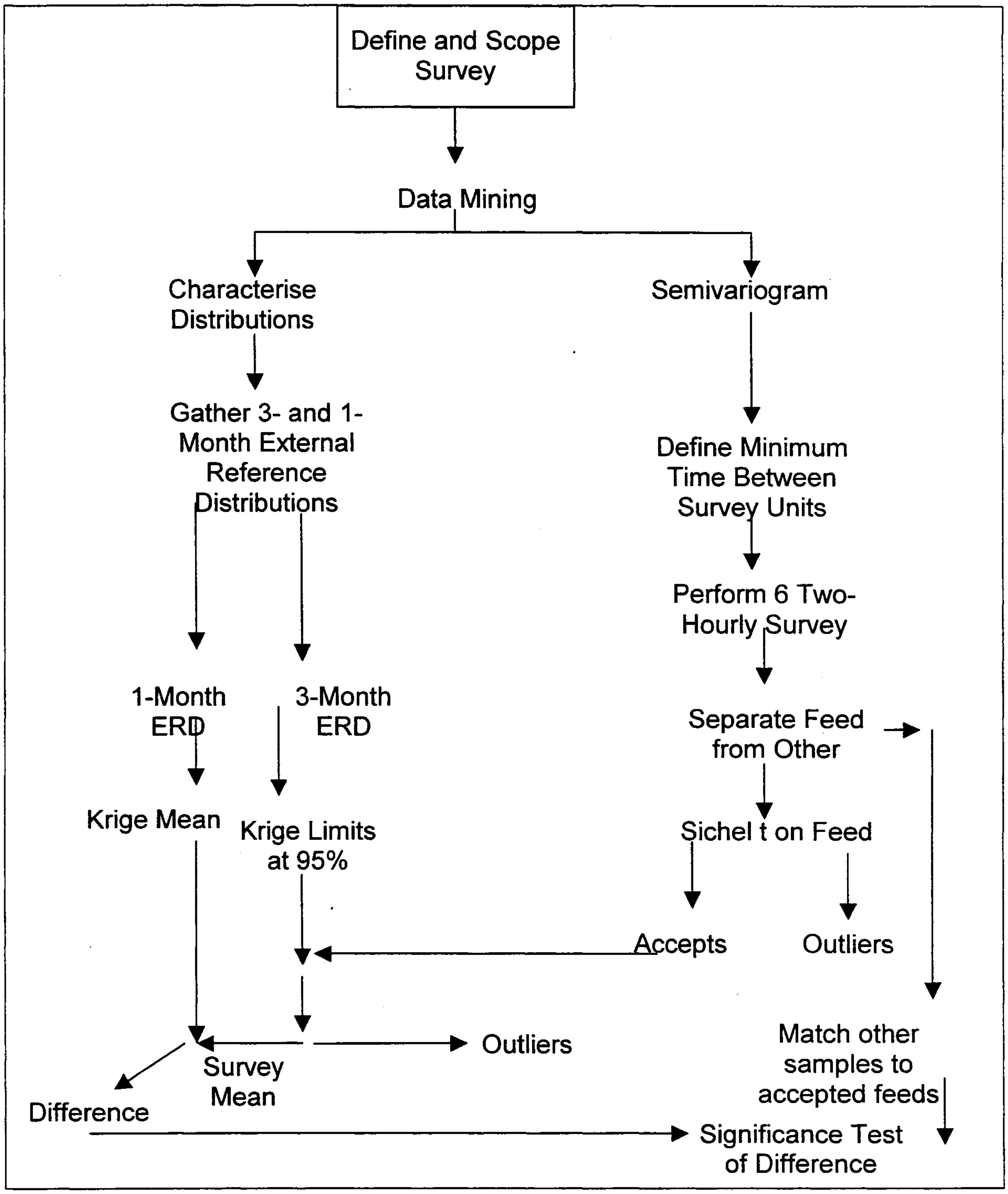


Figure 40 - New Flowsheet for Statistical Benchmark Survey

6.2. Campaign Model

The foregoing analysis of the Raglan November 2003 survey suggests that a good agreement can be obtained between drill-core grade estimates and ore milled. To achieve this, the true lognormality of the nickel distribution in space must be recognised. In addition, the treatment of the concentrator data with prior knowledge of the residual lognormality must be respected. A provisional layout of the Campaign Survey follows.

Survey Date and Planning

Planning, communication and teamwork form the foundation of a successful survey. The foregoing description of defining the survey in a scope of work applies equally to this survey option, the more so in terms of drawing on drill-core data and arranging the mining of chosen segments of the ore reserve to properly represent the desired ore mixture.

External Reference Distribution

The first External Reference Distribution is drawn from drill-core records. Krige's method of 1962, described in pp. 64-65 and pp. 116-119, is used to estimate the distribution parameters. After the ore is mined, the working faces are surveyed to obtain the degree of waste dilution. This figure is used to correct the undiluted grades. Sichel's t-estimator is used on the same data. It is important here to note that, prior to these calculations, the meterage per drill core section is used to weight each datum.

The second External Reference Distribution is drawn from the production records as ore milled. This form differs from that of the Statistical Benchmark Survey in that the number of observations is considerably lower.

Internal Reference Distribution

This distribution is constructed from the six two-hour survey units in exactly the same manner as for the *Statistical Benchmark Survey*.

CHAPTER 7 - CONTRIBUTION TO KNOWLEDGE

This study has attempted to open a dialogue between mineral processing, mineral science, geology, and applied statistics. There is far more yet to be done in this interesting hybrid between the aforementioned disciplines.

For the present, the following contributions to knowledge are suggested. The reader is reminded that these observations have been made on the Raglan system. In other applications, the number of modes in the compound distribution in ore, for example, may be different. It is therefore recommended that in such a case where another investigator applies these models to another ore, he or she would have to characterise that ore system from the beginning. The general method of reference distributions, however, should be generic:

7.1. There is residual lognormality in the Rougher Flotation Feed and Final Tailings of the concentrator which has been studied. The Rougher Float Feed displays a compound trimodal distribution, and the Final Tailings, a bimodal distribution. The Final Concentrate shows a normally-distributed set of nickel concentrate grades.

7.2. The milling and flotation processes treat ore which demonstrates full compound lognormality in space as drill-core. The mining, crushing and grinding processes reduce this to a residual compound lognormality in the time dimension, expressed as Rougher Float Feed. Thereafter, separation processes such as flotation physically stratify the compound distribution to simplified subdistributions.

7.3. The lognormal probability plot of Final Tailings nickel grades is a semi-quantitative estimator of recoverable nickel, and allows monitoring of concentrator operations progress as various changes are made to improve performance. This is economically done by data mining.

7.4. A standard *Statistical Benchmark Survey* model has been derived from first principles. Quantitative measurements for trueness of sampling have been derived and tested. This outcome resides in the close agreement of the final adjudicated sample means of the Internal and External Reference Distributions.

7.5. An optional *Mill Campaign* model has been derived, in which the full lognormality of metal value distribution in space had to be balanced with a limited milling campaign of a chosen ore mix. Quantitative measurements for trueness of sampling were successfully derived and tested. Again this outcome resides in the close agreement of the various reference distribution means.

CHAPTER 8 - CONCLUSIONS

8.1. Reference to Specific Objectives

Reference to the specific objectives of this thesis (page 22) is made. The study has, in terms of these specific objectives, shown the following:

One improved survey model, called Statistical Benchmark Surveying, has been produced for the typical characterisation of an operating concentrator. This was produced from a critical audit of the existing heuristic Falconbridge survey model. The improvements have been derived from appropriate characterisation of the End Members in the geology of the system, and in characterisation of the distributions of nickel in the Raglan orebody and in ore milled.

A new survey model has been developed for the mining and milling of a selected ore mixture. This is called the Campaign Survey, and successful interfaces with the full lognormal effects of nickel distribution with the residual counterparts in the concentrator have been proven.

The review of published literature has shown that only four publications on the subject of surveying concentrators could be found. None of these produced any evidence of a fundamental basis for their surveying. In all of these approaches, the concentrator was sampled only once. No statistical application was suggested in order to verify that the survey had been representative. The apparent absence of this level of knowledge in the public domain is therefore a real absence.

The case studies of the 1998 survey of Raglan, and of the campaign survey of Raglan in November 1998, have been reviewed in detail. Both new models have stood up to examinations by appropriate statistical methods.

8.2. Characterisation Of Distributions

Prior to any survey being contemplated for a particular operating concentrator, it is most advisable to perform characterisation exercises on the distributions of drill core, feed, concentrate and tailings. This informs the investigator of the type of parent and subordinate distributions that have to be sampled. Prior knowledge of the geological end-members and associated paymetal grade ranges enhances this interpretation.

8.3. Residual Lognormality

Within the concentrator feed and battery limit products, residual effects of lognormality reside in rougher flotation feed and final tailings; the final concentrate values are normally-distributed. Where the arithmetic and corrected geometric means agree well from these distributions, the associated confidence limits are very different and are more appropriately estimated using the lognormal platform.

8.4. Statistical Benchmark Survey

For the *Statistical Benchmark Surveying* of an operating concentrator to characterise typical operating performance, an improved surveying model has been developed and tested using the survey data of Raglan in 1998. In this model, the External Reference Distribution, using Krige's method of 1962, sets the very important standard of defining 'typical' ore milled. The Internal Reference Distribution, consisting of the six two-hour survey units, emulates the External Reference Distribution, and is modelled using the Sichel t-Estimator. Quality control on the Internal Reference Distribution by either Sichel's t estimator within the Internal Reference Distribution, or by Krige's uncorrected geometric mean and associated limits from the External Reference Distribution, is consistently performed.

Overall Assessment

In overall assessment of the new statistical benchmark survey, Table 77 (originally Table 52) summarises the results. The improvement in the agreement between independent measures of ore grade milled is significant. Supplementary examination of Final Concentrate and Final Tailings is consistent.

Table 77 - Comparison of Reference Distribution Means for Old and New Survey Methods

Survey Type	Rougher Float Feed % Ni		Final Concentrate % Ni		Final Tailings % Ni	
	Internal	External	Internal	External	Internal	External
Old	2.96	3.13	15.67	*	0.48	*
New	3.13	3.13	15.68	15.70	0.51	0.52

*Old model did not use External Reference Distributions for Final Concentrate and Tailings

It is concluded that the new statistical benchmark survey has improved the agreement between the two reference distributions in feed, and has added checks on the Final Tailings and Concentrate. For the latter, the agreement is good. Repetition of this work at other concentrators would be outside the scope of this thesis. Preliminary work on the Strathcona Concentrator in Sudbury, Ontario, has already indicated that residual compound lognormality is to be found in the data for the battery limit samples **[Lotter and Langlois, 2003]**.

8.5. Campaign Survey

The new *Campaign Survey* model is designed for the case where a specially-selected ore type or mixture of ore types is to be mined and milled on its own in a finite campaign of approximately one week duration. This is useful for either describing a problematic ore or some future ore mix that will be treated at a later stage of the mine plan. In such a case, the full lognormality of paymetal distribution applies for the External Reference Distribution if calculated from drill-core data (including waste dilution, which is surveyed after the mining campaign).

For the rest of the reference distributions, a similar approach to the above *Statistical Benchmark Survey* is applied. This results in an outlier rejection process that brings the shortened survey data set closer in agreement with the two External Reference Distributions.

Overall Assessment

Independent measures of ore grade and concentrator product grades for the new model are shown in Table 78.

Table 78 - Comparison of Reference Distribution Means
For Campaign Survey, November 2003

Element	Rougher Float Feed % Ni		Final Concentrate % Ni		Final Tailings % Ni	
	External	Internal	External	Internal	External	Internal
Ni	2.87*	2.89	18.55	18.05	0.44	0.45

*Mean of drill core and ore milled External Reference Distributions

It is concluded that the proposed Campaign Survey model has demonstrated successful matching of drill core, ore milled and survey data, across a dimension between space and time. Significance testing has been drafted and tried. Initial indications are encouraging.

CHAPTER 9 - FURTHER WORK

Further work has been identified from this study. Some of this further work has been identified directly from this study; other further work is obvious from the remaining broad scope of this field. It is suggested that the following be considered:

9.1. Decomposition of Compound Distributions

A deeper examination of the possible quantitative models from the compound distributions found in drill core, ore milled, and final tailings, might lead to further predictive or diagnostic properties from the orebody and the concentrator operations. In particular, the hypothesis that the log bimodal distribution in tailings exists because of two different orders of magnitude of nickel concentration (arising from different nickel mineral hosts) should be tested. The balance between high-grade particles and low-grade particles in the Final Tailings is clearly the basis of the monitoring model. As the flowsheet performance improves, less and less of the high grade particles will be present; and more of the low-grade particles will be present. The TRIPOD results were not particularly encouraging, and a different approach may be necessary.

9.2. Extension into Other Concentrator Applications

It will be worthwhile to extend the above models into other sulphide concentrator operations. In the case where the lead paymetal(s) are either Gold or Platinum Group Elements, because of their lower level of concentration, there may be necessary improvements to the Reference Distributions in order to address the higher level of measurement error from lead collection fire-assaying.

9.3. Processing of Final Survey Samples

The subsequent sampling and sizing processes that follow the assembly of the final composite sample material from the survey are as important in maintaining representativity, or trueness, as the prior activities of primary surveying, which have been described in this study. It is suggested that further work in studying and optimising this field of sample mass reduction to the polished section would be rewarding.

9.4. Overall Errors and the Nested Analysis of Variance

It would be informative to conduct a nested analysis of variance experiment in order to quantify the various contributing errors to the total error in the distribution of paymetal grades for this type of surveying system. In earlier work, the comparative errors of such a system in laboratory scale flotation tests were characterised **[Lotter, 1995a]**. In this work, the effects of sample preparation ahead of lead collection fire-assaying were significant and potentially reduced the bias of the mean grade estimate(s) in flotation test product(s). These sample preparation treatments were physical, *i.e.* comminution by one means or another. Use of specialist software such as Crystal Ball to model the variance would enhance the interpretation.

9.5. Compound Distributions

The earlier work abovementioned by **Lotter, 1995a**, also reported the existence of compound distributions in the Platinum Group Element (PGE) measurements for the Merensky Reef, South Africa. Although this was not the main focus of the study at the time, the potential modelling value of this approach was then suggested. The present work, based on nickel measurement in the Raglan system, confirms the presence of compound distributions in space and time. It is suggested that there are more base metal and PGE mines/concentrator operations that may also show this feature. Further work is indicated in this regard.

9.6 Error Modelling

A further exercise should be conducted on the modelling and characterisation of total errors in the sampling of the Internal Reference Distribution. This work will cross-check the approach which has been developed and proposed in this thesis.

CHAPTER 10 - REFERENCES

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APPENDIX

1. Raglan Rougher Float Feed December 2002-November 2003

December 2002

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	1	3.31	0.84	11	21	4.65	1.35	21	41	3.80	1.10
	2	2.65	0.73		22	4.92	1.44		42	3.97	1.15
2	3	3.01	0.79	12	23	5.10	1.39	22	43	3.94	1.13
	4	3.01	0.79		24	3.76	1.05		44	3.75	1.10
3	5	3.49	1.03	13	25	3.96	1.07	23	45	3.91	1.10
	6	3.46	0.92		26	4.61	1.20		46	3.30	0.93
4	7	3.89	1.11	14	27			24	47	3.35	0.99
	8	3.37	0.95		28	4.33	1.13		48	3.70	1.06
5	9	3.82	1.07	15	29	3.68	1.03	25	49	3.32	0.98
	10	3.54	0.94		30	3.65	1.03		50	3.19	0.87
6	11			16	31	4.02	1.05	26	51	3.51	0.95
	12				32	4.05	1.00		52	6.30	1.36
7	13			17	33	3.78	0.94	27	53	4.18	1.13
	14				34	3.96	0.95		54	4.36	1.24
8	15			18	35			28	55	4.44	1.27
	16				36				56	3.95	1.05
9	17			19	37	3.56	0.98	29	57	3.57	0.97
	18				38	4.03	1.08		58	3.63	1.05
10	19	3.91	1.13	20	39	3.68	0.94	30	59	3.90	1.13
	20	3.62	1.04		40	3.51	0.95		60	3.30	0.88
								31	61	3.53	0.96
									62	3.77	0.98

January 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	63	3.58	0.98	11	83	3.09	0.89	21	103	4.15	1.19
	64	3.79	1.11		84	3.36	1.01		104	4.36	1.18
2	65	3.32	1.02	12	85	3.15	0.91	22	105	4.21	0.92
	66	3.41	1.19		86	3.00	0.87		106	4.11	0.91
3	67	3.41	1.02	13	87	3.10	0.95	23	107	4.08	1.09
	68	3.09	0.94		88	2.96	0.93		108	4.29	1.33
4	69			14	89	2.74	0.88	24	109	3.78	1.12
	70	3.32	1.10		90	3.18	0.99		110	3.60	1.05
5	71	3.19	0.92	15	91	2.80	0.87	25	111	3.58	1.05
	72	3.04	0.96		92	3.46	0.95		112	3.92	1.17
6	73	3.24	1.03	16	93	3.56	1.02	26	113	5.50	1.53
	74	3.65	1.28		94	3.72	1.06		114	3.91	1.18
7	75	3.25	1.29	17	95	3.50	1.04	27	115	3.64	1.06
	76	2.70	0.87		96	3.66	1.07		116	3.92	1.16
8	77	3.17	0.85	18	97			28	117	3.23	0.92
	78	3.14	0.92		98	4.02	1.15		118	3.37	0.83
9	79	3.08	0.83	19	99	3.76	0.97	29	119	3.99	1.10
	80	3.11	0.73		100	3.30	0.86		120	3.87	0.89
10	81	3.48	0.88	20	101	3.49	0.94	30	121	3.75	0.76
	82	3.34	0.92		102	4.05	1.14		122		
								31	123	3.89	0.93
									124		

February 2002

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	125	3.58	0.93	11	145	3.43	1.05	21	165	3.51	0.93
	126	3.52	0.91		146	3.38	1.00		166	3.63	0.89
2	127	3.66	0.97	12	147	3.78	1.01	22	167		
	128	4.51	1.11		148	3.73	1.09		168	3.57	1.12
3	129	4.05	0.97	13	149	3.51	0.92	23	169	3.30	0.94
	130	3.47	0.91		150	3.50	1.09		155	3.10	1.00
4	131	3.08	0.77	14	151	3.58	1.05	24	171	3.50	0.96
	132	3.51	0.99		152	3.04	0.91		172	3.41	0.99
5	133	3.45	0.99	15	153	3.31	0.98	25	173	3.20	0.93
	134	3.91	1.05		154	3.16	0.94		174	3.27	0.85
6	135	3.70	1.04	16	155	3.10	1.00	26	175	3.38	0.94
	136	3.83	1.15		156	3.82	0.98		176	3.59	1.05
7	137	3.35	0.99	17	157	3.75	0.93	27	177	3.59	1.05
	138	3.26	0.94		158	4.04	1.02		178	3.42	0.95
8	139	3.04	0.82	18	159	3.72	0.89	28	179	3.72	1.07
	140	4.01	0.96		160	3.37	0.88		180	3.80	1.17
9	141	4.18	1.10	19	161	3.27	0.85				
	142	4.46	1.31		162	3.20	0.85				
10	143	3.42	0.96	20	163	3.22	0.93				
	144	3.56	1.07		164	3.16	0.95				

March 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	181	3.01	0.87	11	201	3.61	1.01	21	221	3.59	1.00
	182	3.41	0.97		202	3.69	1.14		222	4.16	1.11
2	183	3.20	0.94	12	203	3.68	1.09	22	223	4.12	1.11
	184	3.15	0.93		204	2.86	0.80		224	4.39	1.14
3	185	3.04	1.04	13	205	2.70	0.74	23	225	3.74	0.96
	186	3.64	1.18		206	3.57	1.13		226	3.57	0.98
4	187	3.70	1.06	14	207	3.59	1.16	24	227	3.95	1.07
	188	3.47	1.05		208	3.31	1.08		228	4.00	1.09
5	189	3.38	1.12	15	209	3.61	1.13	25	229	4.18	1.10
	190	3.65	1.06		210	3.75	1.22		230	3.95	1.05
6	191	3.19	0.92	16	211	3.81	1.37	26	231	4.08	1.30
	192	3.43	1.01		212	3.81	1.37		232	3.85	1.16
7	193	2.95	0.87	17	213	3.64	1.22	27	233	4.24	1.26
	194	3.58	0.86		214	3.73	1.20		234	3.14	1.02
8	195	3.42	0.88	18	215	3.90	1.23	28	235		
	196	3.97	1.08		216	3.45	1.05		236		
9	197	4.49	1.20	19	217	3.19	1.04	29	237		
	198	4.27	1.04		218	3.48	1.11		238		
10	199	4.09	1.09	20	219	3.86	1.10	30	239		
	200	4.14	1.13		220	3.77	1.01		240		
								31	241		
									242		

April 2002

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	243			11	263	3.34	0.87	21	283	5.45	1.26
	244				264	3.87	0.92		284	4.39	0.98
2	245			12	265	3.60	0.93	22	285	4.51	0.98
	246	2.85	0.91		266	3.60	0.91		286	4.52	1.01
3	247	3.27	1.04	13	267	3.16	0.76	23	287	4.17	1.11
	248				268	3.05	0.78		288	4.00	0.85
4	249			14	269	3.08	0.82	24	289	4.20	0.99
	250	4.90	1.34		270	3.43	0.84		290	4.82	1.10
5	251	4.62	1.27	15	271	3.86	0.97	25	291	4.51	1.06
	252	4.25	1.10		272	3.70	0.90		292	3.67	0.85
6	253	4.08	0.93	16	273	3.89	0.95	26	293	3.87	0.92
	254	4.65	0.99		274	2.52	0.66		294	3.57	0.84
7	255	3.55	0.94	17	275	3.55	0.82	27	295	3.30	0.83
	256	3.51	0.96		276	5.16	1.16		296	3.96	0.93
8	257	3.57	0.92	18	277	4.47	1.10	28	297	3.95	0.98
	258	3.58	0.89		278	3.45	0.81		298	3.43	0.84
9	259	3.79	0.89	19	279	4.51	1.04	29	299	3.70	0.91
	260	3.27	0.84		280	3.94	0.94		300	4.12	0.97
10	261	2.49	0.76	20	281	5.50	1.20	30	301	3.65	0.94
	262	3.01	0.73		282	4.74	1.10		302	3.50	0.84

May 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	303	3.92	0.97	11	323	3.45	0.91	21	343	3.02	0.81
	304	3.37	0.88		324	3.45	0.89		344	3.01	0.85
2	305	3.57	0.95	12	325	3.45	0.91	22	345	3.01	0.81
	306	3.52	0.91		326	3.45	0.89		346	3.04	0.85
3	307			13	327	3.15	0.82	23	347	3.11	0.87
	308	4.02	0.98		328	3.24	0.87		348	3.40	0.88
4	309	3.79	0.96	14	329	2.73	0.71	24	349	2.81	0.76
	310	3.65	0.94		330	4.06	1.04		350	3.09	0.80
5	311	3.61	0.97	15	331	3.66	0.93	25	351	3.41	0.94
	312	3.51	0.94		332	4.43	1.07		352	3.41	0.91
6	313	3.66	0.84	16	333	3.71	0.89	26	353	3.32	0.83
	314	2.78	0.77		334	3.22	0.84		354	3.22	0.86
7	315	2.86	0.78	17	335	3.58	0.94	27	355	2.97	0.80
	316	3.50	0.91		336	3.51	0.94		356	2.97	0.80
8	317	3.55	0.88	18	337	3.02	0.84	28	357	2.79	0.81
	318	3.54	0.89		338	2.86	0.81		358	2.93	0.86
9	319	2.81	0.77	19	339	2.99	0.81	29	359	3.12	0.88
	320	2.77	0.74		340	3.23	0.80		360	3.23	0.90
10	321	3.35	0.80	20	341	3.27	0.83	30	361	3.50	0.92
	322				342	3.27	0.83		362	3.13	0.84
								31	363	3.20	0.89
									364	3.39	0.93

June 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	365	3.19	0.87	11	385	3.19	0.91	21	405	3.56	1.04
	366	3.31	0.90		386	3.19	0.85		406	4.04	1.21
2	367			12	387	3.17	0.84	22	407	4.91	1.51
	368	3.21	0.88		388	3.39	0.88		408	5.30	1.44
3	369	3.80	0.96	13	389	3.90	1.08	23	409	5.17	1.52
	370	3.89	0.95		390	3.61	0.97		410	4.64	1.32
4	371	3.54	0.90	14	391			24	411	4.36	1.27
	372	3.74	0.96		392	4.34	1.20		412	3.61	1.08
5	373	4.06	0.99	15	393	3.95	1.11	25	413	3.24	0.95
	374	4.50	1.11		394	3.98	1.08		414	3.87	1.19
6	375	3.77	0.96	16	395	4.23	1.20	26	415	3.65	1.08
	376	4.21	1.07		396	4.93	1.35		416	3.68	1.16
7	377			17	397	4.02	1.05	27	417	2.81	0.82
	378	3.16	0.83		398	3.42	0.82		418	3.31	1.01
8	379	3.22	0.87	18	399	3.53	0.87	28	419	4.12	1.37
	380	4.21	1.07		400	3.60	1.07		420	4.12	1.37
9	381	3.79	1.05	19	401	3.44	1.01	29	421	3.73	1.14
	382	2.84	0.83		402	3.08	0.85		422	4.08	1.06
10	383	3.29	0.96	20	403	3.11	0.88	30	423	3.98	1.12
	384	3.19	0.93		404	3.50	1.10		424	4.10	1.22

July 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	425	3.76	1.13	11	445	2.46	0.75	21	465	2.92	0.84
	426	3.56	1.00		446	2.56	0.80		466	3.15	0.84
2	427	3.72	1.20	12	447			22	467	3.38	0.91
	428	3.64	1.11		448	2.93	0.90		468	3.55	0.92
3	429	3.45	0.95	13	449	3.47	0.99	23	469	2.77	0.79
	430	4.00	1.10		450	3.81	0.91		470		
4	431	3.55	0.90	14	451	3.11	0.78	24	471		
	432	3.86	1.06		452	2.91	0.73		472		
5	433	3.40	0.99	15	453	3.08	0.78	25	473	3.11	0.79
	434	3.65	1.03		454	3.21	0.81		474	3.61	0.94
6	435	3.76	1.06	16	455	3.43	0.90	26	475	3.61	0.88
	436	3.56	1.06		456	3.47	0.88		476	3.51	0.91
7	437	4.03	1.16	17	457	3.34	0.87	27	477	3.28	0.85
	438	4.22	1.22		458	3.40	0.91		478	3.39	0.87
8	439	3.94	1.12	18	459	3.39	0.96	28	479	3.15	0.82
	440	3.94	1.12		460	3.41	0.93		480	3.61	1.00
9	441	4.03	1.16	19	461	3.19	0.89	29	481	3.91	0.97
	442	4.13	1.20		462	2.67	0.77		482	3.21	0.88
10	443	3.70	0.98	20	463	3.24	0.95	30	483		
	444	3.10	0.85						484	3.39	0.88
								31	485	2.98	0.85
									486	2.94	0.81

August 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	487	3.15	0.84	11	507	2.79	0.79	21	527	2.95	0.91
	488	3.05	0.80		508	2.78	0.78		528	2.98	0.95
2	489	2.76	0.77	12	509	2.47	0.69	22	529	2.79	0.85
	490	4.04	1.00		510	2.70	0.74		530	2.72	0.80
3	491	3.93	1.06	13	511	2.73	0.73	23	531	2.94	0.86
	492	3.17	0.90		512	2.59	0.72		532	3.16	0.90
4	493	2.71	0.74	14	513	2.48	0.69	24	533	3.11	0.89
	494	2.84	0.76		514	2.38	0.64		534	3.41	1.08
5	495	2.47	0.68	15	515	2.33	0.61	25	535	3.22	0.95
	496	2.50	0.67		516				536	3.53	1.07
6	497	2.73	0.79	16	517			26	537	2.98	0.88
	498	3.94	1.02		518				538	3.06	0.91
7	499	4.38	1.23	17	519	2.66	0.73	27	539	3.09	0.88
	500	2.99	0.87		520	3.87	0.92		540	2.94	0.89
8	501	3.16	0.90	18	521	3.00	0.85	28	541	2.84	0.81
	502	2.66	0.72		522	2.67	0.79		542	3.61	0.97
9	503	2.63	0.72	19	523	2.39	0.70	29	543	3.38	0.95
	504	2.74	0.71		524	3.15	0.95		544	3.54	1.10
10	505	2.38	0.62	20	525	2.67	0.81	30	545	3.47	1.04
	506	2.80	0.76		526	2.94	0.87		546	3.53	1.09
								31	547	3.09	0.91
									548	3.47	1.04

September 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	549	3.25	0.99	11	569	3.34	0.98	21	589	4.07	1.60
	550	3.17	1.01		570	3.65	1.00		590	3.90	1.56
2	551	3.38	1.08	12	571	3.40	1.07	22	591	3.65	1.28
	552	3.54	1.19		572	3.43	1.21		592	3.82	1.24
3	553			13	573	3.02	1.03	23	593	3.54	1.18
	554	2.88	0.95		574	3.16	1.17		594	3.72	1.30
4	555	3.45	1.11	14	575	2.98	1.08	24	595	3.85	1.29
	556	3.51	1.15		576	3.33	1.19		596	4.28	1.32
5	557	3.12	0.99	15	577	3.52	1.26	25	597	4.24	1.30
	558	3.19	1.10		578	3.10	1.34		598	4.61	1.41
6	559	3.24	0.97	16	579	3.15	1.19	26	599	4.56	1.58
	560	3.19	1.03		580	3.46	1.22		600	4.73	1.65
7	561	3.49	1.08	17	581	3.62	1.35	27	601	4.02	1.47
	562	4.04	1.18		582	3.58	1.17		602	4.15	1.47
8	563	3.72	1.14	18	583	3.45	1.14	28	603	3.53	1.16
	564	3.32	1.01		584	3.29	1.21		604	3.55	1.20
9	565	3.62	1.08	19	585	3.46	0.98	29	605	3.87	1.30
	566	3.84	1.27		586	3.94	0.97		606	3.97	1.42
10	567	3.54	1.13	20	587	4.48	1.14	30	607	4.64	1.44
	568	3.43	1.03		588	4.19	1.46		608	3.71	1.32

October 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	609	3.36	1.15	11	629	2.67	0.78	21	649	3.30	0.91
	610	3.37	1.25		630	3.05	0.92		650	3.46	0.99
2	611	3.55	1.19	12	631	2.95	0.88	22	651	3.26	0.93
	612	3.09	0.95		632	3.24	1.04		652	3.12	0.86
3	613	3.10	1.01	13	633	2.70	0.75	23	653	3.18	0.85
	614				634	2.62	0.70		654		
4	615			14	635	2.67	0.67	24	655		
	616	2.87	0.84		636	3.11	0.81		656		
5	617			15	637	3.38	0.90	25	657		
	618	3.25	1.19		638	3.10	0.85		658		
6	619	3.49	1.25	16	639	3.01	0.85	26	659		
	620	3.33	1.12		640	3.02	0.81		660		
7	621			17	641	3.27	0.82	27	661		
	622	3.63	1.16		642	3.16	0.88		662		
8	623			18	643	3.31	1.01	28	663		
	624				644	3.01	0.83		664		
9	625	3.17	0.92	19	645	3.17	0.96	29	665	3.00	0.93
	626	3.35	0.96		646	3.15	0.86		666	4.04	1.02
10	627	3.02	0.91	20	647	3.25	0.92	30	667	3.16	0.84
	628	2.99	0.93		648	3.05	0.83		668	3.29	0.85
								31	669	3.67	1.00
									670	3.31	1.03

November 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	3.51	1.06	3.51	11	3.01	0.94	3.01	21	3.01	0.90	3.01
	3.81	1.16	3.81		3.01	0.94	3.01		2.83	0.88	2.83
2	4.37	1.31	4.37	12	3.79	1.12	3.79	22	2.90	0.86	2.90
	4.05	1.27	4.05		3.41	1.04	3.41		2.98	0.88	2.98
3	3.77	1.10	3.77	13	2.97	0.92	2.97	23	3.59	1.17	3.59
	3.21	1.04	3.21		2.99	0.94	2.99		2.74	0.75	2.74
4	3.28	0.99	3.28	14	3.13	1.00	3.13	24	2.77	0.75	2.77
					3.45	1.06	3.45		2.94	0.82	2.94
5				15	3.32	1.09	3.32	25	3.10	0.90	3.10
					3.58	1.07	3.58		3.38	1.06	3.38
6				16	3.44	1.22	3.44	26	3.06	0.92	3.06
					2.83	0.84	2.83		3.05	0.89	3.05
7	3.13	0.98	3.13	17	2.75	0.82	2.75	27	2.83	0.85	2.83
	3.13	0.98	3.13		3.00	0.80	3.00		3.00	0.91	3.00
8	3.26	0.96	3.26	18	2.48	0.78	2.48	28	2.78	0.85	2.78
	3.35	1.05	3.35		2.78	0.78	2.78		2.73	0.80	2.73
9	3.25	0.98	3.25	19	2.73	0.84	2.73	29	2.76	0.81	2.76
	3.23	0.95	3.23		2.87	0.82	2.87		3.49	0.99	3.49
10	2.92	0.86	2.92	20	2.92	0.98	2.92	30	3.46	0.95	3.46
	2.90	0.88	2.90		3.24	0.89	3.24		3.72	1.02	3.72

2. Raglan Final Concentrate December 2002-November 2003

December 2002

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	1	20.7	5.05	11	21	19.8	5.80	21	41	17.8	5.22
	2	20.1	5.02		22	19.0	5.76		42	18.7	5.55
2	3	20.5	5.22	12	23	18.6	5.40	22	43	17.4	4.89
	4	20.5	5.22		24	18.9	5.08		44	18.5	5.39
3	5	21.1	5.92	13	25	19.1	5.34	23	45	20.3	5.67
	6	19.4	5.10		26	21.0	5.28		46	21.6	6.20
4	7	18.8	5.27	14	27			24	47	21.4	6.12
	8	19.1	4.94		28	21.5	5.61		48	21.0	5.88
5	9	20.1	5.81	15	29	20.0	5.61	25	49	19.6	5.71
	10	21.1	5.21		30	18.4	5.14		50	21.9	6.69
6	11			16	31	18.3	4.75	26	51	22.1	6.35
	12				32	20.6	4.93		52	23.1	5.36
7	13			17	33	19.3	4.67	27	53	17.1	4.65
	14				34	20.2	4.79		54	19.5	5.75
8	15			18	35			28	55	21.5	6.13
	16				36				56	22.3	5.83
9	17			19	37	18.5	5.51	29	57	21.2	5.68
	18				38	16.8	4.35		58	20.4	5.57
10	19	18.3	5.43	20	39	18.6	4.72	30	59	20.5	5.83
	20	18.2	5.14		40	18.7	4.72		60	21.5	5.25
								31	61	19.5	5.20
									62	21.4	5.21

January 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	63	21.8	5.75	11	83	18.7	5.01	21	103	18.7	5.55
	64	21.6	6.40		84	19.6	5.77		104	18.8	5.28
2	65	21.0	6.24	12	85	19.6	5.25	22	105	21.5	4.84
	66	19.0	6.53		86	19.5	5.40		106	21.2	4.94
3	67	20.8	6.05	13	87	19.4	5.53	23	107	19.1	5.09
	68	20.1	5.82		88	19.5	5.91		108	18.5	5.90
4	69			14	89	19.5	5.70	24	109	19.3	5.55
	70	18.6	6.09		90	20.2	6.11		110	18.4	5.16
5	71	19.4	5.46	15	91	19.8	5.83	25	111	19.1	5.59
	72	19.7	5.64		92	20.9	5.60		112	17.5	5.32
6	73	19.2	5.94	16	93	20.6	5.65	26	113	16.7	5.23
	74	18.7	6.50		94	19.5	5.36		114	18.0	5.25
7	75	18.0	6.97	17	95	20.1	5.84	27	115	19.5	5.51
	76	18.1	5.84		96	19.2	5.45		116	20.4	5.86
8	77	20.9	5.43	18	97			28	117	19.5	5.20
	78	20.0	5.68		98	19.2	5.52		118	19.0	4.53
9	79	20.8	5.37	19	99	18.8	4.81	29	119	18.1	4.96
	80	20.3	4.73		100	17.9	4.34		120	19.0	4.46
10	81	20.9	5.08	20	101	18.6	4.92	30	121	16.7	3.30
	82	20.5	5.50		102	20.2	5.47		122		
								31	123		
									106	21.2	4.94

February 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	125	18.1	4.20	11	145	17.3	5.04	21	165	20.0	5.26
	126	20.0	5.30		146	19.1	5.55		166	19.2	4.47
2	127	20.3	5.29	12	147	19.8	5.21	22	167		
	128	20.3	5.29		148	19.5	5.69		168	17.8	5.67
3	129	21.7	5.06	13	149	18.8	4.75	23	169	18.2	5.14
	130	20.6	5.08		150	18.7	5.81		170	18.3	4.45
4	131	19.6	4.54	14	151	19.5	5.58	24	171	17.8	4.79
	132	18.2	5.11		152	19.4	5.55		172	19.3	5.44
5	133	19.4	5.41	15	153	19.9	5.74	25	173	18.5	6.16
	134	20.0	5.46		154	19.6	5.73		174	19.9	4.84
6	135	20.0	5.53	16	155	19.3	5.96	26	175	20.7	5.38
	136	19.3	5.75		156	20.8	5.17		176	20.8	6.18
7	137	19.2	5.52	17	157	19.8	4.54	27	177	20.1	4.99
	138	18.4	5.31		158	20.0	4.84		178	20.6	5.45
8	139	18.5	4.97	18	159	19.5	4.42	28	179	19.3	5.46
	140	21.2	5.16		160	20.7	5.17		180	20.6	6.58
9	141	18.2	4.59	19	161	18.7	4.54				
	142	18.1	5.68		162	19.0	4.80				
10	143	18.5	4.97	20	163	19.1	5.27				
	144	19.0	6.02		164	18.3	5.32				

March 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	181	20.2	5.54	11	201	20.1	5.30	21	221	20.5	5.65
	182	19.9	5.52		202	19.3	5.89		222	21.4	5.67
2	183	19.7	5.53	12	203	20.0	5.82	22	223	19.9	5.26
	184	19.8	5.51		204	20.4	5.36		224	20.3	5.07
3	185	18.1	6.12	13	205	18.3	4.66	23	225	20.7	5.75
	186	19.2	6.02		206	20.5	6.22		226	19.3	5.09
4	187	19.0	5.43	14	207	20.4	6.54	24	227	19.7	5.24
	188	18.4	6.01		208	18.9	5.76		228	20.9	5.79
5	189	19.0	5.98	15	209	19.6	6.29	25	229	20.9	5.15
	190	19.7	5.49		210	19.4	6.04		230	22.2	5.56
6	191	20.0	5.53	16	211	18.9	7.43	26	231	20.5	6.08
	192	21.3	6.39		212	18.9	7.43		232	19.1	5.77
7	193	19.7	5.45	17	213	16.7	5.48	27	233	17.7	5.26
	194	19.7	4.48		214	19.1	6.06		234	18.2	5.30
8	195	19.9	4.86	18	215	19.2	5.90	28	235		
	196	21.1	5.54		216	18.4	5.51		236		
9	197	19.2	5.21	19	217	18.6	5.84	29	237		
	198	21.7	5.11		218	18.7	5.79		238		
10	199	20.2	5.08	20	219	18.9	5.39	30	239		
	200	21.1	5.52		220	20.7	5.38		240		
								31	241		
									242		

April 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	243			11	263	18.3	4.74	21	283	21.0	5.09
	244				264	19.1	4.48		284	21.3	4.81
2	245	20.1	7.00	12	265	21.2	5.37	22	285	22.2	5.17
	246	20.1	7.00		266	18.4	4.32		286	21.9	5.14
3	247	17.9	5.79	13	267	19.2	4.46	23	287	20.8	5.38
	248				268	19.3	4.54		288	22.4	4.75
4	249			14	269	18.9	5.06	24	289	22.1	4.98
	250	19.9	5.60		270	19.1	4.99		290	20.7	4.72
5	251	18.4	5.39	15	271	18.7	4.66	25	291	21.9	5.25
	252	18.0	4.61		272	20.2	5.18		292	20.2	4.51
6	253	18.7	4.19	16	273	21.8	5.39	26	293	20.9	4.79
	254	18.4	3.94		274	19.5	4.61		294	19.2	4.24
7	255	18.2	4.89	17	275	22.8	5.49	27	295	20.9	4.91
	256	18.5	4.93		276	22.6	4.95		296	20.7	4.67
8	257	19.2	4.77	18	277	21.2	5.35	28	297	20.4	4.77
	258	18.9	4.45		278	21.9	5.01		298	20.5	4.68
9	259	19.5	4.47	19	279	23.0	5.52	29	299	19.6	4.70
	260	18.6	4.54		280	21.8	5.20		300	19.6	4.58
10	261	17.6	5.07	20	281	22.6	5.20	30	301	19.3	4.72
	262	17.3	3.92		282	22.5	5.19		302	19.4	4.44

May 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	303	19.5	4.60	11	323	17.9	4.66	21	343	20.1	4.91
	304	19.3	4.77		324	21.4	5.12		344	20.2	5.32
2	305	19.4	4.93	12	325	17.9	4.66	22	345	20.7	5.18
	306	20.2	5.13		326	21.4	5.12		346	20.4	5.24
3	307			13	327	19.1	4.92	23	347	19.4	5.07
	308	20.6	4.80		328	19.1	4.91		348	20.8	5.13
4	309	19.8	4.79	14	329	20.0	4.82	24	349	19.3	4.96
	310	19.5	4.87		330	20.4	5.26		350	20.7	4.95
5	311	20.1	5.18	15	331	20.3	5.04	25	351	19.9	5.31
	312	19.5	4.88		332	20.5	4.85		352	18.9	4.75
6	313	20.5	4.66	16	333	21.0	4.88	26	353	20.3	4.77
	314	19.3	4.71		334	19.5	4.85		354	20.0	4.97
7	315	19.3	4.94	17	335	19.2	4.92	27	355	19.3	4.69
	316	21.8	5.24		336	20.9	5.34		356	20.3	5.07
8	317	18.2	4.04	18	337	21.6	5.59	28	357	19.6	5.24
	318	19.0	4.43		338	19.3	5.01		358	19.8	5.47
9	319	18.3	4.64	19	339	19.7	4.93	29	359	20.8	5.46
	320	17.9	4.35		340	21.6	4.94		360	20.0	5.22
10	321	18.4	4.24	20	341	21.6	5.21	30	361	19.8	4.99
	322				342	21.6	5.21		362	18.9	4.67
								31	363	20.8	5.46
									364	21.1	5.71

June 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	365	21.4	5.63	11	385	20.0	5.30	21	405	19.7	5.62
	366	20.8	5.36		386	20.7	5.17		406	20.3	6.01
2	367			12	387	20.0	5.06	22	407	19.9	6.15
	368	20.1	5.16		388	20.3	4.95		408	21.1	5.97
3	369	20.9	5.17	13	389	19.1	5.06	23	409	19.9	6.17
	370	20.7	4.85		390	20.8	5.38		410	20.8	6.04
4	371	19.8	4.65	14	391	0.0	0.00	24	411	19.6	5.55
	372	20.5	5.01		392	20.7	5.73		412	19.2	5.62
5	373	19.7	4.61	15	393	20.0	5.30	25	413	20.0	5.43
	374	20.9	5.10		394	21.0	5.43		414	20.4	6.04
6	375	20.1	4.99	16	395	21.1	5.92	26	415	18.6	5.38
	376	20.4	5.16		396	19.5	5.17		416	19.1	5.75
7	377	0.0	0.00	17	397	20.2	5.12	27	417	19.1	5.38
	378	19.9	5.73		398	21.0	4.99		418	17.3	5.17
8	379	18.9	5.12	18	399	19.1	4.69	28	419	20.2	6.93
	380	19.8	4.95		400	18.0	5.16		420	20.2	6.93
9	381	19.2	5.14	19	401	18.8	5.23	29	421	19.2	5.77
	382	19.5	5.37		402	18.4	4.77		422	20.5	5.40
10	383	21.1	5.96	20	403	17.5	4.49	30	423	20.7	5.68
	384	19.5	5.26		404	18.0	5.50		424	20.6	6.03

July 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	425	19.7	5.80	11	445	16.9	4.71	21	465	19.8	5.41
	426	21.0	5.74		446	16.8	4.94		466	19.9	5.21
2	427	18.4	5.98	12	447	0.0	0.00	22	467	19.9	5.42
	428	20.1	5.84		448	18.2	5.11		468	18.9	4.48
3	429	19.6	5.38	13	449	19.3	5.06	23	469	19.9	5.28
	430	20.4	5.49		450	20.3	4.59		470		
4	431	21.9	5.46	14	451	19.8	4.64	24	471		
	432	20.5	5.76		452	20.3	4.82		472	20.7	5.07
5	433	20.1	5.50	15	453	20.3	4.63	25	473	20.0	4.74
	434	19.7	5.46		454	21.5	5.00		474	20.9	5.29
6	435	20.2	5.36	16	455	20.8	5.05	26	475	20.1	4.84
	436	20.1	5.64		456	19.4	4.55		476	20.0	5.05
7	437	20.4	5.84	17	457	20.3	4.85	27	477	19.1	4.81
	438	20.6	5.80		458	19.8	4.91		478	20.3	4.99
8	439	19.0	5.02	18	459	19.5	5.18	28	479	20.2	4.86
	440	19.6	5.66		460	20.3	5.27		480	20.6	5.56
9	441	17.7	5.42	19	461	20.9	5.48	29	481	19.7	4.62
	442	18.6	5.18		462	19.8	5.37		482	18.5	4.71
10	443	18.4	4.56	20	463	20.5	5.96	30	483		
	444	18.1	4.42		464	19.8	5.07		484	18.8	4.55
								31	485	19.1	5.11
									486	19.6	4.94

August 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	487	19.6	4.94	11	507	18.8	4.85	21	527	5.98	0.361
	488	19.5	4.81		508	18.1	4.81		528	6.02	0.353
2	489	19.7	5.33	12	509	18.6	5.10	22	529	19.7	5.83
	490	20.1	4.97		510	20.4	5.44		530	19.2	5.37
3	491	18.4	4.87	13	511	21.2	5.40	23	531	20.3	5.65
	492	17.9	4.59		512	21.2	5.61		532	20.1	5.18
4	493	19.0	4.79	14	513	20.8	5.60	24	533	20.0	5.34
	494	18.1	4.60		514	21.6	5.64		534	19.3	5.78
5	495	18.2	4.65	15	515	21.6	5.37	25	535	20.1	5.75
	496	18.0	4.53		516				536	20.1	5.52
6	497	19.3	5.13	16	517			26	537	20.5	5.82
	498	19.6	5.23		518				538	20.3	5.78
7	499	19.5	5.42	17	519	18.8	4.92	27	539	19.7	5.33
	500	18.2	4.93		520	18.6	5.08		540	19.8	5.81
8	501	19.0	5.02	18	521	19.0	5.15	28	541	20.3	5.55
	502	18.3	4.49		522	18.9	5.28		542	19.3	5.12
9	503	20.1	5.13	19	523	18.5	5.11	29	543	19.6	5.31
	504	20.1	4.78		524	18.2	5.38		544	19.7	5.90
10	505	21.3	5.19	20	525	19.3	5.66	30	545	21.2	6.22
	506	19.8	5.02		526	18.3	5.27		546	19.1	5.78
								31	547	19.2	5.58
									548	19.1	5.41

September 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	549	20.3	5.91	11	569	18.5	5.58	21	589	16.9	6.62
	550	19.7	6.02		570	18.6	5.39		590	18.2	7.60
2	551	20.8	6.58	12	571	19.5	5.91	22	591	20.0	6.89
	552	19.9	6.54		572	18.1	6.44		592	20.0	6.33
3	553	0.0	0.00	13	573	18.7	6.66	23	593	17.1	5.63
	554	20.4	6.61		574	17.9	6.56		594	17.5	6.28
4	555	19.4	6.14	14	575	18.2	6.55	24	595	17.8	6.02
	556	19.2	6.23		576	18.4	6.39		596	19.8	6.29
5	557	19.7	6.10	15	577	18.9	6.31	25	597	18.1	5.66
	558	19.8	6.68		578	18.4	7.71		598	18.2	5.58
6	559	21.1	6.38	16	579	17.7	6.66	26	599	18.6	6.66
	560	19.6	6.16		580	18.7	6.38		600	19.3	6.57
7	561	20.5	6.28	17	581	18.3	6.78	27	601	19.2	7.13
	562	20.2	5.75		582	19.1	6.24		602	19.3	6.91
8	563	18.9	5.76	18	583	18.6	6.33	28	603	18.9	6.08
	564	19.8	6.15		584	18.1	6.33		604	18.8	6.79
9	565	20.1	5.78	19	585	19.3	5.37	29	605	19.8	6.67
	566	17.6	5.75		586	18.7	4.45		606	19.5	7.20
10	567	18.4	5.99	20	587	19.6	4.81	30	607	20.3	6.40
	568	18.3	5.29		588	17.6	6.29		608	19.2	7.00

October 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	609	18.5	6.42	11	629	20.00	5.66	21	649	18.11	4.59
	610	18.4	6.98		630	20.2	6.22		650	19.0	4.94
2	611	18.5	6.36	12	631	18.45	5.39	22	651	17.62	4.80
	612	18.6	5.71		632	20.3	5.62		652	19.0	5.09
3	613	18.0	5.67	13	633	19.50	5.01	23	653	20.09	5.22
	614	19.5	6.04		634	21.1	5.37		654	20.9	5.47
4	615	17.0	5.40	14	635	22.00	5.28	24	655		
	616	19.4	5.33		636	21.0	5.30		656		
5	617	20.1	5.66	15	637	19.60	5.10	25	657		
	618	17.6	6.41		638	17.6	4.53		658		
6	619	19.1	6.89	16	639	19.40	5.33	26	659		
	620	18.6	6.63		640	20.3	5.06		660		
7	621	19.1	5.92	17	641	21.85	5.25	27	661		
	622	19.2	6.14		642	20.2	5.07		662		
8	623	0.00	0.00	18	643	17.95	4.72	28	663		
	624	0.0	0.00		644	18.2	5.67		664		
9	625	19.45	5.40	19	645	18.66	4.87	29	665	21.50	6.67
	626	19.4	5.18		646	20.0	5.25		666	21.0	5.18
10	627	18.55	5.43	20	647	18.86	4.85	30	667	21.28	5.61
	609	18.5	6.42		648	18.5	4.88		668	22.7	5.81
								31	669	18.95	5.02
									670	17.5	5.15

November 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	671	19.2	5.53	11	686	18.8	5.37	21	706	17.4	5.05
	672	18.5	5.35		687	18.8	5.37		707	17.2	4.72
2	673	16.9	4.77	12	688	19.9	5.59	22	708	17.8	4.89
	674	19.5	5.91		689	19.5	5.31		709	17.8	4.93
3	675	15.9	4.31	13	690	20.2	5.42	23	710	16.9	5.18
	676	17.9	5.49		691	20.2	5.43		711	18.5	4.66
4	677	19.2	5.47	14	692	19.6	5.51	24	712	19.0	4.86
	673				693	19.4	5.35		713	18.1	4.71
5	674			15	694	17.6	5.65	25	714	17.8	4.88
	675				695	17.6	5.16		715	16.7	5.03
6	676			16	696	16.8	6.01	26	716	17.55	5.07
	677				697	17.9	4.96		717	18.1	4.94
7	678	19.9	6.05	17	698	19.2	5.43	27	718	17.61	5.06
	679	19.9	6.05		699	19.6	4.93		720	18.31	5.33
8	680	19.6	5.48	18	700	18.3	5.55	28	721	19.0	5.25
	681	20.5	6.09		701	18.5	4.99		722	18.19	5.12
9	682	19.3	5.59	19	702	18.5	5.48	29	723	18.3	4.97
	683	18.8	5.26		703	18.0	4.91		724	17.93	4.77
10	684	18.8	5.17	20	704	18.4	6.12	30	725	18.3	4.97
	685	19.3	5.50		705	16.5	4.37		726	16.8	6.01

3. Raglan Final Tailings December 2002 - November 2003

December 2002

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	1	0.63	0.23	11	21	0.49	0.16	21	41	0.50	0.20
	2	0.51	0.21		22	0.50	0.16		42	0.72	0.25
2	3	0.29	0.14	12	23	0.66	0.28	22	43	0.75	0.33
	4	0.29	0.14		24	0.44	0.17		44	0.52	0.22
3	5	0.47	0.22	13	25	0.40	0.14	23	45	0.55	0.21
	6	0.49	0.19		26	0.51	0.21		46	0.42	0.19
4	7	0.37	0.17	14	27			24	47	0.56	0.25
	8	0.48	0.23		28	0.37	0.11		48	0.41	0.17
5	9	0.47	0.20	15	29	0.30	0.11	25	49	0.39	0.18
	10	0.44	0.17		30	0.29	0.13		50	0.85	0.20
6	11			16	31	0.36	0.16	26	51	0.47	0.14
	12				32	0.45	0.16		52	0.77	0.16
7	13			17	33	0.50	0.22	27	53	0.49	0.14
	14				34	0.53	0.22		54	0.55	0.16
8	15			18	35			28	55	0.72	0.21
	16				36				56	0.42	0.13
9	17			19	37	0.31	0.12	29	57	0.34	0.11
	18				38	0.43	0.16		58	0.34	0.13
10	19	0.37	0.14	20	39	0.37	0.14	30	59	0.38	0.16
	20	0.42	0.17		40	0.48	0.18		60	0.36	0.14
								31	61	0.34	0.14
									62	0.62	0.26

January 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	63	0.54	0.25	11	83	0.35	0.15	21	103	0.59	0.19
	64	0.49	0.24		84	0.42	0.20		104	0.62	0.20
2	65	0.57	0.30	12	85	0.36	0.16	22	105	0.60	0.16
	66	0.43	0.21		86	0.38	0.20		106	0.58	0.14
3	67	0.48	0.21	13	87	0.49	0.27	23	107	0.61	0.15
	68	0.31	0.13		88	0.52	0.28		108	0.83	0.22
4	69			14	89	0.54	0.30	24	109	0.67	0.21
	70	0.29	0.13		90	0.56	0.31		110	0.43	0.17
5	71	0.29	0.12	15	91	0.44	0.26	25	111	0.51	0.19
	72	0.28	0.11		92	0.64	0.34		112	0.65	0.17
6	73	0.31	0.14	16	93	0.53	0.29	26	113	0.66	0.17
	74	0.37	0.15		94	0.77	0.39		114	0.52	0.18
7	75	0.34	0.13	17	95	0.43	0.16	27	115	0.46	0.18
	76	0.33	0.13		96	0.79	0.35		116	0.59	0.21
8	77	0.36	0.13	18	97			28	117	0.50	0.19
	78	0.32	0.12		98	0.40	0.16		118	0.37	0.16
9	79	0.33	0.13	19	99	0.39	0.14	29	119	0.42	0.16
	80	0.32	0.11		100	0.39	0.17		120	0.46	0.15
10	81	0.40	0.13	20	101	0.42	0.17	30	121	0.93	0.21
	82	0.37	0.14		102	0.60	0.19		122		
								31	123		
									124	0.43	0.13

February 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	125	0.39	0.15	11	145	0.51	0.21	21	165	0.31	0.15
	126	0.32	0.15		146	0.50	0.20		166	0.39	0.17
2	127	0.42	0.17	12	147	0.47	0.16	22	167	0.00	0.00
	128	0.51	0.16		148	0.42	0.17		168	0.30	0.15
3	129	0.63	0.18	13	149	0.47	0.17	23	169	0.35	0.16
	130	0.55	0.18		150	0.47	0.17		170	0.35	0.15
4	131	0.42	0.14	14	151	0.40	0.16	24	171	0.37	0.16
	132	0.37	0.15		152	0.41	0.16		172	0.47	0.18
5	133	0.42	0.17	15	153	0.35	0.15	25	173	0.48	0.18
	134	0.68	0.21		154	0.39	0.16		174	0.45	0.17
6	135	0.48	0.17	16	155	0.34	0.15	26	175	0.37	0.17
	136	0.53	0.18		156	0.42	0.18		176	0.49	0.21
7	137	0.37	0.14	17	157	0.56	0.22	27	177	0.43	0.14
	138	0.37	0.16		158	0.40	0.16		178	0.37	0.16
8	139	0.34	0.14	18	159	0.39	0.15	28	179	0.41	0.16
	140	0.49	0.17		160	0.40	0.15		180	0.55	0.19
9	141	0.67	0.22	19	161	0.41	0.17				
	142	0.76	0.22		162	0.35	0.15				
10	143	0.60	0.17	20	163	0.30	0.13				
	144	0.54	0.19		164	0.31	0.15				

March 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	181	0.25	0.12	11	201	0.36	0.14	21	221	0.36	0.15
	182	0.32	0.13		202	0.31	0.13		222	0.44	0.16
2	183	0.29	0.12	12	203	0.37	0.14	22	223	0.40	0.16
	184	0.29	0.13		204	0.36	0.15		224	0.45	0.17
3	185	0.31	0.14	13	205	0.31	0.13	23	225	0.39	0.16
	186	0.40	0.16		206	0.39	0.15		226	0.27	0.12
4	187	0.36	0.14	14	207	0.38	0.15	24	227	0.34	0.15
	188	0.48	0.19		208	0.35	0.13		228	0.31	0.16
5	189	0.35	0.14	15	209	0.35	0.14	25	229	0.42	0.16
	190	0.31	0.13		210	0.37	0.15		230	0.45	0.17
6	191	0.35	0.13	16	211	0.29	0.13	26	231	0.44	0.20
	192	0.45	0.16		212	0.29	0.13		232	0.45	0.20
7	193	0.36	0.14	17	213	0.29	0.14	27	233	0.63	0.29
	194	0.33	0.12		214	0.38	0.17		234	0.51	0.15
8	195	0.34	0.14	18	215	0.38	0.15	28	235		
	196	0.37	0.15		216	0.39	0.15		236		
9	197	0.42	0.18	19	217	0.33	0.14	29	237		
	198	0.46	0.14		218	0.33	0.13		238		
10	199	0.40	0.16	20	219	0.36	0.15	30	239		
	200	0.51	0.18		220	0.43	0.16		240		
								31	241		
									242		

April 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	243		243	11	263	0.34	0.13	21	283	0.44	0.13
	244		244		264	0.33	0.12		284	0.39	0.10
2	245	0.72	245	12	265	0.32	0.13	22	285	0.34	0.10
	246	0.72	246		266	0.40	0.15		286	0.34	0.09
3	247	0.37	247	13	267	0.37	0.15	23	287	0.41	0.13
	248		248		268	0.32	0.16		288	0.31	0.08
4	249		249	14	269	0.34	0.12	24	289	0.34	0.09
	250	0.35	250		270	0.35	0.11		290	0.40	0.11
5	251	0.40	251	15	271	0.43	0.13	25	291	0.41	0.12
	252	0.30	252		272	0.49	0.13		292	0.37	0.12
6	253	0.38	253	16	273	0.48	0.11	26	293	0.37	0.14
	254	0.40	254		274	0.27	0.11		294	0.37	0.14
7	255	0.45	255	17	275	0.38	0.11	27	295	0.34	0.15
	256	0.45	256		276	0.50	0.10		296	0.35	0.13
8	257	0.43	257	18	277	0.44	0.12	28	297	0.37	0.15
	258	0.34	258		278	0.38	0.10		298	0.38	0.16
9	259	0.38	259	19	279	0.48	0.13	29	299	0.36	0.16
	260	0.35	260		280	0.41	0.10		300	0.48	0.16
10	261	0.35	261	20	281	0.52	0.12	30	301	0.42	0.17
	262	0.31	262		282	0.57	0.12		302	0.32	0.12

May 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	303	0.34	0.14	11	323	0.36	0.17	21	343	0.36	0.16
	304	0.33	0.14		324	0.53	0.20		344	0.37	0.17
2	305	0.37	0.14	12	325	0.30	0.12	22	345	0.41	0.17
	306	0.38	0.14		326	0.29	0.13		346	0.35	0.17
3	307	0.00	0.00	13	327	0.31	0.13	23	347	0.45	0.21
	308	0.35	0.13		328	0.30	0.11		348	0.34	0.15
4	309	0.37	0.15	14	329	0.33	0.12	24	349	0.39	0.16
	310	0.36	0.13		330	0.40	0.14		350	0.33	0.16
5	311	0.35	0.12	15	331	0.39	0.13	25	351	0.41	0.19
	312	0.35	0.13		332	0.35	0.13		352	0.41	0.19
6	313	0.33	0.14	16	333	0.36	0.15	26	353	0.37	0.16
	314	0.30	0.14		334	0.37	0.16		354	0.35	0.16
7	315	0.36	0.16	17	335	0.38	0.19	27	355	0.34	0.17
	316	0.37	0.17		336	0.35	0.18		356	0.33	0.17
8	317	0.35	0.16	18	337	0.40	0.17	28	357	0.34	0.17
	318	0.29	0.16		338	0.32	0.15		358	0.34	0.17
9	319	0.39	0.17	19	339	0.36	0.17	29	359	0.40	0.19
	320				340	0.36	0.17		360	0.40	0.19
10	321	0.36	0.17	20	341	0.36	0.17	30	361	0.47	0.20
	322	0.53	0.20		342	0.37	0.16		362	0.38	0.16
								31	363	0.41	0.17
									364	0.46	0.16

June 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	365	0.41	0.16	11	385	0.36	0.18	21	405	0.35	0.14
	366	0.41	0.16		386	0.33	0.14		406	0.44	0.15
2	367			12	387	0.34	0.15	22	407	0.56	0.18
	368	0.33	0.15		388	0.37	0.14		408	0.54	0.15
3	369	0.39	0.17	13	389	0.34	0.15	23	409	0.51	0.17
	370	0.45	0.15		390	0.35	0.15		410	0.60	0.18
4	371	0.41	0.15	14	391			24	411	0.52	0.18
	372	0.39	0.16		392	0.37	0.16		412	0.44	0.16
5	373	0.51	0.18	15	393	0.35	0.17	25	413	0.49	0.18
	374	0.60	0.20		394	0.37	0.16		414	0.44	0.19
6	375	0.41	0.14	16	395	0.40	0.15	26	415	0.38	0.16
	376	0.50	0.16		396	0.47	0.14		416	0.37	0.17
7	377	0.00	0.00	17	397	0.39	0.14	27	417	0.37	0.15
	378	0.32	0.12		398	0.40	0.14		418	0.39	0.16
8	379	0.33	0.13	18	399	0.40	0.14	28	419	0.40	0.18
	380	0.41	0.13		400	0.45	0.18		420	0.40	0.18
9	381	0.41	0.17	19	401	0.40	0.17	29	421	0.38	0.17
	382	0.34	0.15		402	0.40	0.17		422	0.51	0.16
10	383	0.38	0.17	20	403	0.37	0.16	30	423	0.50	0.18
	384	0.38	0.17		404	0.35	0.18		424	0.41	0.14

July 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	425	0.42	0.17	11	445	0.36	0.15	21	465	0.42	0.14
	426	0.36	0.14		446				466	0.44	0.15
2	427	0.36	0.15	12	447			22	467	0.43	0.15
	428	0.35	0.14		448	0.39	0.19		468	0.44	0.15
3	429	0.32	0.13	13	449	0.50	0.22	23	469	0.38	0.16
	430	0.40	0.16		450	0.51	0.18		470		
4	431	0.42	0.16	14	451	0.40	0.15	24	471		
	432	0.36	0.13		452	0.34	0.13		472	0.30	0.13
5	433	0.40	0.16	15	453	0.34	0.13	25	473	0.37	0.15
	434	0.39	0.15		454	0.38	0.14		474	0.46	0.16
6	435	0.46	0.20	16	455	0.36	0.15	26	475	0.49	0.16
	436	0.35	0.14		456	0.52	0.21		476	0.40	0.13
7	437	0.44	0.17	17	457	0.52	0.20	27	477	0.41	0.15
	438	0.73	0.27		458	0.50	0.20		478	0.44	0.19
8	439	0.80	0.38	18	459	0.44	0.17	28	479	0.38	0.16
	440	0.47	0.21		460	0.43	0.16		480	0.43	0.20
9	441	0.41	0.19	19	461	0.39	0.15	29	481	0.47	0.18
	442	0.40	0.17		462	0.37	0.14		482	0.39	0.15
10	443	0.38	0.15	20	463	0.40	0.17	30	483		
	444	0.36	0.15		464	0.41	0.13		484	0.37	0.17
								31	485	0.39	0.17
									486	0.39	0.17

August 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	487	0.38	0.16	11	507	0.35	0.16	21	527	0.23	0.010
	488	0.39	0.17		508	0.31	0.14		528	0.18	0.008
2	489	0.37	0.14	12	509	0.34	0.13	22	529	0.31	0.12
	490	0.47	0.14		510	0.41	0.14		530	0.29	0.12
3	491	0.41	0.14	13	511	0.44	0.16	23	531	0.38	0.18
	492	0.45	0.18		512	0.39	0.14		532	0.36	0.18
4	493	0.37	0.14	14	513	0.41	0.13	24	533	0.51	0.24
	494	0.36	0.16		514	0.38	0.12		534	0.40	0.19
5	495	0.38	0.14	15	515	0.38	0.13	25	535	0.44	0.18
	496	0.37	0.14		516				536	0.39	0.14
6	497	0.42	0.16	16	517			26	537	0.37	0.14
	498	0.49	0.17		518				538	0.33	0.13
7	499	0.66	0.19	17	519	0.31	0.13	27	539	0.36	0.14
	500	0.40	0.16		520	0.29	0.13		540	0.35	0.14
8	501	0.40	0.16	18	521	0.37	0.15	28	541	0.30	0.13
	502	0.36	0.16		522	0.34	0.14		542	0.40	0.16
9	503	0.36	0.16	19	523	0.32	0.12	29	543	0.38	0.16
	504	0.35	0.14		524	0.40	0.17		544	0.36	0.15
10	505	0.33	0.12	20	525	0.43	0.19	30	545	0.36	0.15
	506	0.34	0.13		526	0.48	0.24		546	0.42	0.15
								31	547	0.38	0.15
									548	0.43	0.18

September 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	549	0.44	0.20	11	569	0.43	0.19	21	589	0.44	0.20
	550	0.35	0.17		570	0.52	0.19		590	0.47	0.21
2	551	0.35	0.16	12	571	0.40	0.18	22	591	0.51	0.21
	552	0.45	0.18		572	0.42	0.18		592	0.55	0.20
3	553			13	573	0.36	0.17	23	593	0.48	0.18
	554	0.38	0.16		574	0.41	0.19		594	0.43	0.16
4	555	0.40	0.16	14	575	0.41	0.21	24	595	0.45	0.16
	556	0.43	0.17		576	0.45	0.24		596	0.50	0.16
5	557	0.35	0.15	15	577	0.38	0.18	25	597	0.54	0.16
	558	0.49	0.22		578	0.47	0.23		598	0.54	0.16
6	559	0.42	0.16	16	579	0.40	0.19	26	599	0.69	0.23
	560	0.37	0.14		580	0.45	0.19		600	0.63	0.19
7	561	0.38	0.15	17	581	0.40	0.20	27	601	0.50	0.19
	562	0.44	0.16		582	0.51	0.17		602	0.53	0.19
8	563	0.43	0.18	18	583	0.44	0.16	28	603	0.48	0.17
	564	0.50	0.19		584	0.48	0.21		604	0.39	0.13
9	565	0.45	0.17	19	585	0.44	0.17	29	605	0.51	0.18
	566	0.55	0.20		586	0.59	0.19		606	0.46	0.17
10	567	0.48	0.20	20	587	0.57	0.20	30	607	0.80	0.23
	568	0.46	0.19		588	0.49	0.20		608	0.44	0.14

October 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	609	0.45	0.16	11	629	0.34	0.15	21	649	0.35	0.16
	610	0.40	0.15		630	0.36	0.14		650	0.37	0.16
2	611	0.50	0.19	12	631	0.36	0.16	22	651	0.39	0.17
	612	0.40	0.12		632	0.36	0.17		652	0.42	0.18
3	613	0.47	0.15	13	633	0.37	0.16	23	653	0.37	0.15
	614	0.63	0.17		634	0.33	0.13		654	0.46	0.14
4	615	0.71	0.20	14	635	0.33	0.12	24	655		
	616	0.41	0.16		636	0.41	0.15		656		
5	617	0.39	0.15	15	637	0.41	0.15	25	657		
	618	0.40	0.17		638	0.39	0.16		658		
6	619	0.52	0.19	16	639	0.31	0.14	26	659		
	620	0.44	0.19		640	0.38	0.17		660		
7	621	0.45	0.19	17	641	0.40	0.16	27	661		
	622	0.41	0.18		642	0.41	0.19		662		
8	623			18	643	0.41	0.18	28	663		
	624				644	0.47	0.21		664		
9	625	0.37	0.15	19	645	0.37	0.15	29	665	0.55	0.17
	626	0.45	0.18		646	0.35	0.14		666	0.47	0.14
10	627	0.33	0.15	20	647	0.38	0.16	30	667	0.49	0.14
	628	0.39	0.16		648	0.36	0.16		668	0.49	0.14
								31	669	0.51	0.17
									670	0.50	0.21

November 2003

Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%	Date	No.	Ni%	Cu%
1	671	0.49	0.19	11	686	0.53	0.24	21	706	0.4	0.15
	672	0.55	0.23		687	0.53	0.24		707	0.54	0.27
2	673	0.81	0.33	12	688	0.55	0.22	22	708	0.4	0.19
	674	0.64	0.24		689	0.6	0.27		709	0.36	0.16
3	675	0.79	0.31	13	690	0.92	0.39	23	710	0.46	0.23
	676	0.51	0.22		691	0.9	0.39		711	0.44	0.18
4	677	0.51	0.2	14	692	0.51	0.28	24	712	0.31	0.13
	673				693	0.68	0.32		713	0.34	0.15
5	674			15	694	0.59	0.22	25	714	0.34	0.15
	675				695	0.59	0.2		716	0.38	0.15
6	676			16	696	0.6	0.21	26	717	0.53	0.24
	677				697	0.59	0.23		718	0.37	0.17
7	678	0.42	0.16	17	698	0.4	0.16	27	719	0.34	0.14
	679	0.42	0.16		699	0.44	0.16		720	0.35	0.16
8	680	0.46	0.19	18	700	0.36	0.15	28	721	0.32	0.14
	681	0.51	0.22		701	0.44	0.16		722	0.3	0.14
9	682	0.48	0.19	19	702	0.47	0.18	29	723	0.33	0.13
	683	0.49	0.19		703	0.44	0.16		724	0.4	0.15
10	684	0.5	0.2	20	704	0.4	0.15	30	725	0.43	0.15
	685	0.42	0.19		705	0.56	0.19		726	0.51	0.15

4. Raglan Rougher Float Feed May-July 1998

May 1998

Date	No.	Ni%	Date	No.	Ni%	Date	No.	Ni%
1	1		11	21	3.35	21	41	3.81
	2			22	3.34		42	3.48
2	3		12	23	3.56	22	43	3.38
	4			24	3.52		44	3.58
3	5	3.5	13	25	3.11	23	45	3.67
	6	3.25		26	3.51		46	3.28
4	7	2.64	14	27	3.37	24	47	3.44
	8	2.49		28	3.36		48	3.05
5	9	2.77	15	29	4.11	25	49	3.07
	10	3.01		30	3.54		50	3.35
6	11	3.41	16	31	2.91	26	51	3.19
	12	3.53		32	2.87		52	2.91
7	13	3.45	17	33	3.17	27	53	2.75
	14	3.52		34	3.65		54	3.00
8	15	3.73	18	35	3.14	28	55	3.14
	16	3.4		36	2.98		56	3.40
9	17		19	37	2.83	29	57	3.10
	18	3.45		38	2.76		58	3.08
10	19	3.42	20	39	3.72	30	59	3.08
	20	3.26		40	4.39		60	3.55
						31	61	3.24
							62	3.40

June 1998

Date	No.	Ni%	Date	No.	Ni%	Date	No.	Ni%
1	63	3.47	11	83	2.86	21	103	2.79
	64	3.54		84	2.95		104	2.81
2	65		12	85		22	105	2.93
	66			86	2.90		106	2.94
3	67		13	87	2.96	23	107	2.75
	68			88	2.81		108	3.03
4	69	3.45	14	89	2.77	24	109	3.36
	70	3.27		90	2.93		110	3.23
5	71	3.40	15	91	3.04	25	111	3.09
	72	2.92		92			112	3.27
6	73	2.74	16	93		26	113	3.41
	74	2.94		94	3.00		114	3.48
7	75	2.74	17	95	2.82	27	115	3.35
	76	2.82		96	2.85		116	3.42
8	77	3.22	18	97	2.84	28	117	4.04
	78	3.59		98	2.84		118	3.75
9	79	3.47	19	99	2.69	29	119	3.97
	80	3.41		100	2.65		120	3.32
10	81	3.32	20	101	2.70	30	121	3.36
	82	3.17		102	2.85		122	3.74

July 1998

Date	No.	Ni%	Date	No.	Ni%	Date	No.	Ni%
1	123	3.35	11	143	2.96	21	163	3.86
	124	3.16		144	3.24		164	3.86
2	125	3.47	12	145	2.75	22	165	3.91
	126	3.21		146	3.13		166	3.18
3	127		13	147	2.78	23	167	3.33
	128			148	3.53		168	3.46
4	129	3.47	14	149	3.29	24	169	3.59
	130	3.24		150	3.36		170	3.9
5	131	3.08	15	151	3.38	25	171	3.6
	132	3.31		152	3.50		172	3.66
6	133	3.22	16	153	3.36	26	173	3.5
	134	3.79		154	3.33		174	3.12
7	135	3.43	17	155	3.26	27	175	3.42
	136	3.36		156	3.16		176	3.59
8	137	3.23	18	157	3.4	28	177	3.52
	138	3.60		158	3.43		178	3.47
9	139	3.21	19	159	3.41	29	179	3.28
	140	3.18		160	3.32		180	3.2
10	141	3.02	20	161		30	181	3.32
	142	2.76		162			182	3.54
						31	183	3.38
							184	3.68

5. Raglan Final Concentrate May-July 1998

May 1998

Date	No.	Ni%	Date	No.	Ni%	Date	No.	Ni%
1	1	0	11	21	15.2	21	41	14.3
	2	0		22	14.9		42	14.5
2	3	0	12	23	14.3	22	43	16.2
	4	0		24	15.3		44	16.7
3	5	17.9	13	25	14.2	23	45	15.7
	6	17.9		26	15.3		46	15.2
4	7	17.9	14	27	14.6	24	47	15.7
	8	17.3		28	16.4		48	16.4
5	9	17	15	29	17	25	49	16.6
	10	17.5		30	16.2		50	16.3
6	11	17.2	16	31	14.9	26	51	16.4
	12	18.1		32	15.7		52	15.2
7	13	16.6	17	33	15.7	27	53	16.3
	14	16.3		34	17.6		54	15.8
8	15	16.8	18	35	16.3	28	55	16.7
	16	17.7		36	18		56	15.8
9	17	0	19	37	15.3	29	57	17
	18	15.4		38	17		58	15.7
10	19	16.4	20	39	16.4	30	59	15.1
	20	17.2		40	16.4		60	14.8
						31	61	13
							62	16.1

June 1998

Date	No.	Ni%	Date	No.	Ni%	Date	No.	Ni%
1	63	14	11	83	14.8	21	103	17.9
	64	15.2		84	14.6		104	15.5
2	65	0	12	85	0	22	105	16.5
	66	0		86	15.5		106	15.6
3	67	0	13	87	16	23	107	15.7
	68	0		88	15.4		108	15.8
4	69	16.4	14	89	15.7	24	109	15.4
	70	16.5		90	15.5		110	15.2
5	71	16.1	15	91	16	25	111	14.8
	72	13.8		92	0		112	16.4
6	73	12.3	16	93	0	26	113	15.4
	74	14.1		94	17.5		114	15
7	75	14.6	17	95	15.5	27	115	14.8
	76	14.9		96	16.2		116	14.1
8	77	15.3	18	97	15	28	117	13.9
	78	15		98	15.6		118	13.9
9	79	14	19	99	14.9	29	119	15.4
	80	13.9		100	14.7		120	15.2
10	81	15.1	20	101	15.5	30	121	14.4
	82	14.5		102	16.1		122	13.3

July 1998

Date	No.	Ni%	Date	No.	Ni%	Date	No.	Ni%
1	123	14	11	143	15.8	21	163	19
	124	14.8		144	16.4		164	19
2	125	15.6	12	145	17.4	22	165	14.9
	126	14.6		146	17.8		166	15.2
3	127	0	13	147	16.4	23	167	17.7
	128	0		148	14.8		168	18.5
4	129	16.8	14	149	14.4	24	169	17.7
	130	15.4		150	15.3		170	15.9
5	131	15.5	15	151	15.1	25	171	16.4
	132	14.2		152	15.6		172	15.8
6	133	14.5	16	153	16.5	26	173	15
	134	16.2		154	15.1		174	14.7
7	135	15.6	17	155	16.4	27	175	15.6
	136	15.7		156	17.5		176	14.4
8	137	15.5	18	157	16.6	28	177	15.8
	138	15.5		158	15.7		178	15.9
9	139	15.5	19	159	15.7	29	179	15.8
	140	14.8		160	16.4		180	15.8
10	141	16.4	20	161	0	30	181	14.7
	142	15.5		162	0		182	15.1
						31	183	15.8
							184	16.6

6. Raglan Final Tailings May-July 1998

May 1998

Date	No.	Ni%	Date	No.	Ni%	Date	No.	Ni%
1	1	0.00	11	21	0.52	21	41	0.49
	2	0.00		22	0.67		42	0.66
2	3	0.00	12	23	0.73	22	43	0.59
	4	0.00		24	0.73		44	0.66
3	5	0.50	13	25	0.87	23	45	0.84
	6	0.35		26	0.55		46	0.63
4	7	0.32	14	27	0.45	24	47	0.40
	8	0.32		28	0.55		48	0.41
5	9	0.37	15	29	0.61	25	49	0.46
	10	0.53		30	0.55		50	0.54
6	11	0.43	16	31	0.63	26	51	0.63
	12	0.46		32	0.32		52	0.51
7	13	0.66	17	33	0.47	27	53	0.42
	14	0.55		34	0.70		54	0.56
8	15	0.51	18	35	0.49	28	55	0.67
	16	0.54		36	0.53		56	0.38
9	17	0.00	19	37	0.46	29	57	0.69
	18	0.77		38	0.56		58	1.23
10	19	0.46	20	39	0.74	30	59	1.06
	20	0.67		40	0.55		60	1.00
						31	61	0.66
							62	1.14

June 1998

Date	No.	Ni%	Date	No.	Ni%	Date	No.	Ni%
1	63	0.62	11	83	0.44	21	103	0.44
	64	0.56		84	0.50		104	0.40
2	65	0	12	85	0.00	22	105	0.46
	66	0		86	0.31		106	0.44
3	67	0.00	13	87	0.31	23	107	0.41
	68	0.00		88	0.36		108	0.38
4	69	0.54	14	89	0.43	24	109	0.54
	70	0.57		90	0.51		110	0.52
5	71	0.88	15	91	0.69	25	111	0.48
	72	1.05		92			112	0.54
6	73	1.53	16	93		26	113	0.50
	74	0.76		94	0.34		114	0.52
7	75	0.33	17	95	0.37	27	115	0.52
	76	0.34		96	0.36		116	0.47
8	77	0.39	18	97	0.39	28	117	0.61
	78	0.43		98	0.45		118	0.69
9	79	0.62	19	99	0.44	29	119	0.54
	80	0.58		100	0.44		120	0.86
10	81	0.63	20	101	0.44	30	121	1.01
	82	0.49		102	0.44		122	0.94

July 1998

Date	No.	Ni%	Date	No.	Ni%	Date	No.	Ni%
1	123	0.57	11	143	0.42	21	163	0.48
	124	0.29		144	0.55		164	0.48
2	125	0.42	12	145	0.38	22	165	0.43
	126	0.48		146	0.52		166	0.38
3	127	0.00	13	147	0.38	23	167	0.36
	128	0.00		148	0.56		168	0.41
4	129	0.35	14	149	0.78	24	169	0.49
	130	0.34		150	0.96		170	0.59
5	131	0.45	15	151	0.63	25	171	0.56
	132	0.43		152	0.44		172	0.68
6	133	0.53	16	153	0.43	26	173	0.56
	134	0.51		154	0.42		174	0.69
7	135	0.41	17	155	0.40	27	175	0.89
	136	0.48		156	0.39		176	0.77
8	137	0.48	18	157	0.39	28	177	0.56
	138	0.50		158	0.40		178	0.47
9	139	0.43	19	159	0.35	29	179	0.50
	140	0.44		160	0.37		180	0.53
10	141	0.44	20	161	0.00	30	181	0.46
	142	0.42		162	0.00		182	0.52
						31	183	0.62
							184	0.47

7. Semivariogram Exercise

Data Source : Raglan Rougher Float Feed Nickel Assays by XRF – on-line per Courier system

Datum #	Date/Time	Ni%	Datum #	Date/Time	Ni%
1	6/1/98 12:00	3.46	36	6/1/98 20:45	3.68
2	6/1/98 12:15	3.44	37	6/1/98 21:00	3.69
3	6/1/98 12:30	3.42	38	6/1/98 21:15	3.69
4	6/1/98 12:45	3.58	39	6/1/98 21:30	3.69
5	6/1/98 13:00	3.76	40	6/1/98 21:45	3.69
6	6/1/98 13:15	3.74	41	6/1/98 22:00	3.69
7	6/1/98 13:30	3.72	42	6/1/98 22:15	3.70
8	6/1/98 13:45	3.70	43	6/1/98 22:30	3.70
9	6/1/98 14:00	3.69	44	6/1/98 22:45	3.83
10	6/1/98 14:15	3.67	45	6/1/98 23:00	3.83
11	6/1/98 14:30	3.65	46	6/1/98 23:15	3.84
12	6/1/98 14:45	3.63	47	6/1/98 23:30	3.84
13	6/1/98 15:00	3.62	48	6/1/98 23:45	3.85
14	6/1/98 15:15	3.60	49	6/2/98 0:00	3.85
15	6/1/98 15:30	3.58	50	6/2/98 0:15	3.86
16	6/1/98 15:45	3.71	51	6/2/98 0:30	3.86
17	6/1/98 16:00	3.71	52	6/2/98 0:45	3.86
18	6/1/98 16:15	3.72	53	6/2/98 1:00	3.87
19	6/1/98 16:30	3.73	54	6/2/98 1:15	3.87
20	6/1/98 16:45	3.63	55	6/2/98 1:30	3.75
21	6/1/98 17:00	3.65	56	6/2/98 1:45	3.76
22	6/1/98 17:15	3.67	57	6/2/98 2:00	3.77
23	6/1/98 17:30	3.70	58	6/2/98 2:15	3.77
24	6/1/98 17:45	3.67	59	6/2/98 2:30	3.67
25	6/1/98 18:00	3.54	60	6/2/98 2:45	3.64
26	6/1/98 18:15	3.54	61	6/2/98 3:00	3.62
27	6/1/98 18:30	3.54	62	6/2/98 3:15	3.61
28	6/1/98 18:45	3.55	63	6/2/98 3:30	3.81
29	6/1/98 19:00	3.55	64	6/2/98 3:45	3.95
30	6/1/98 19:15	3.55	65	6/2/98 4:00	3.97
31	6/1/98 19:30	3.56	66	6/2/98 4:15	3.98
32	6/1/98 19:45	3.59	67	6/2/98 4:30	3.82
33	6/1/98 20:00	3.62	68	6/2/98 4:45	3.84
34	6/1/98 20:15	3.65	69	6/2/98 5:00	3.86
35	6/1/98 20:30	3.68			

8. Raglan Campaign Survey November 2003 – Drill Core Data

Datum #	Ni%	Datum #	Ni%
1	0.67	36	2.70
2	0.93	37	2.72
3	1.14	38	2.73
4	1.17	39	2.78
5	1.47	40	2.84
6	1.66	41	2.88
7	1.71	42	2.95
8	1.80	43	3.11
9	1.82	44	3.14
10	1.90	45	3.15
11	1.95	46	3.19
12	2.04	47	3.29
13	2.06	48	3.30
14	2.12	49	3.34
15	2.12	50	3.42
16	2.12	51	3.44
17	2.15	52	3.44
18	2.19	53	3.52
19	2.19	54	3.56
20	2.22	55	3.78
21	2.23	56	3.79
22	2.28	57	3.96
23	2.31	58	4.45
24	2.33	59	4.46
25	2.33	60	4.46
26	2.42	61	4.96
27	2.42	62	5.02
28	2.44	63	5.24
29	2.50	64	5.90
30	2.51	65	5.99
31	2.53	66	6.36
32	2.58	67	6.46
33	2.64	68	6.93
34	2.65	69	6.93
35	2.69	70	10.30