MINERAL RESOURCE EVALUATION OF A PLATINUM TAILINGS RESOURCE: A CASE STUDY.

by

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DECLARATION

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Mashudu I. Muthavhine

27/07/2017
Date

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ABSTRACT

The project investigated the application of geostatistical techniques in evaluating a mechanically deposited platinum tailings resource. The project was undertaken on one of the Anglo American Platinum tailings dams, the identity of which cannot be revealed, due to the agreement in place or permission given.

Remnant unrecovered minerals of economic potential still exist in tailings dams. These unrecovered minerals have influenced several mining companies to turn their attention to the economic potential that still exists in tailings, making them a key strategic component of their resources and reserves.

Geostatistics has been developed and thoroughly tested or improved to address challenges experienced in estimating *in situ* geological ore bodies. The main aim of this Research Project is to test whether these fundamental principles and theories of geostatistics are relevant and appropriate in evaluating man-made ore bodies, such as a Platinum tailings dam, without any significant changes needed on the underlying principles or estimation algorithms. The findings on the Case Study tailings resource can be applied in the evaluation of other tailings dams, as well as any other man-made structures such as low grade rock dumps, muck piles, with related characteristics.

A standard approach (methodology) was followed to evaluate the Case Study tailings resource. Drilling and sampling was conducted through sonic drilling. It is a dry drilling technique that is suitable for sampling unconsolidated particles such as tailings. Thereafter, samples were sent to the laboratory to establish grade (concentration) of Platinum Group Metals (Platinum, Palladium and Rhodium), Gold and Base Metals (Copper and Nickel). Density was also measured, and comprehensively analysed as part of variables of interest in this research.

Statistical analyses were performed on all variables of interest contained in the dam: which are Platinum (Pt), Palladium (Pd), Gold (Au), 3E (two PGMs plus Gold), Copper (Cu), Nickel (Ni) and Density. The underlying statistical distributions of all metals and density were found to be non-symmetrical and slightly positive skewed. The skewness of the distributions was established to be marginal. Differences between raw data (untransformed) averages and the log-normal estimates were analysed and found to be insignificant. As such Ordinary Kriging of untransformed data was concluded to be the appropriate geostatistical technique for Case Study tailings resource.

Analysis of mineralisation continuity (variography), a pre-requisite for geostatistical techniques such as Ordinary Kriging applied on the case study tailings resource, was also performed. Reasonable and sufficient mineralisation continuity was established to exist in the Case Study tailings resource. Although characterised by high nugget effect, these spatial correlations were established to be continuous with ranges of influence well beyond 450 m in all variables. Anisotropic variograms were modelled for all variables and are comprised of nested structures with two to three spherical models.

Resource estimation was conducted through Ordinary Kriging in Datamine. All the seven variables were successfully interpolated into each cell of the 5m x 5m x 5m block model.

Rigorous validation of the resource model was performed to establish the quality and reliability of the estimation carried out. Estimated resource model was analysed against the original borehole data, through comparison of grade profiles, statistical analysis, QQ Plots and histograms.

The grade profile was recognised to be similar between boreholes (5 m composites) and the adjacent cells that have been estimated. Furthermore, statistical analyses revealed minimal differences between means of the estimated model and the original borehole data: the highest difference being 1.7% realised on 3E, followed by 1.1% on Density and Gold (Au). The rest of the variables (Pt, Pd, Cu, and Ni) have differences that are below 1%.

QQ plots and histogram were plotted from resource model with 5m x 5m x 5m cells and 5 m composited boreholes. Although these data sets are of different (slightly incompatible) supports, the intended purpose of comparing distributions was achieved. QQ plots and histograms revealed approximately identical shaped distributions of the two data sets, with some minor deviations noticeable in graphs of only two variables (Au and Density) that are underlain by two populations.

The validation process carried out gave a compelling assurance on the quality and reliability of the resource model produced. The Case Study tailings resource therefore is successfully estimated by Ordinary Kriging.

The results achieved on the Case Study tailings dam has successfully proved that geostatistical principles and theories can confidently be applied, in their current form or understanding, to any man-made tailings resource.

To **GOD** be the glory through **Christ Jesus** (Eph 3:20-21)

I dedicate this Research Report to my family:

My wife (Kanu), my sons (Zwivhuya, Maanda, Tshedza) and my baby daughter (Murendwa).

ACKNOWLEDGEMENTS

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Anglo American Platinum, for the permission to conduct project in one of their tailings dams, and their resources as well as their support that made this project successful.

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CHAPTER ONE: INTRODUCTION

1.1. Location

This research project was conducted on a Tailings Dam owned by Rustenburg Platinum Mines (RPM), a wholly owned subsidiary of Anglo American Platinum (Figure 1). The Case Study tailings dam has reached its operating life and no pumping of tailings is currently happening, or expected to ever happen to the dam in the future. The dam was mainly filled with Merensky Reef, and occasional UG2 Reef.

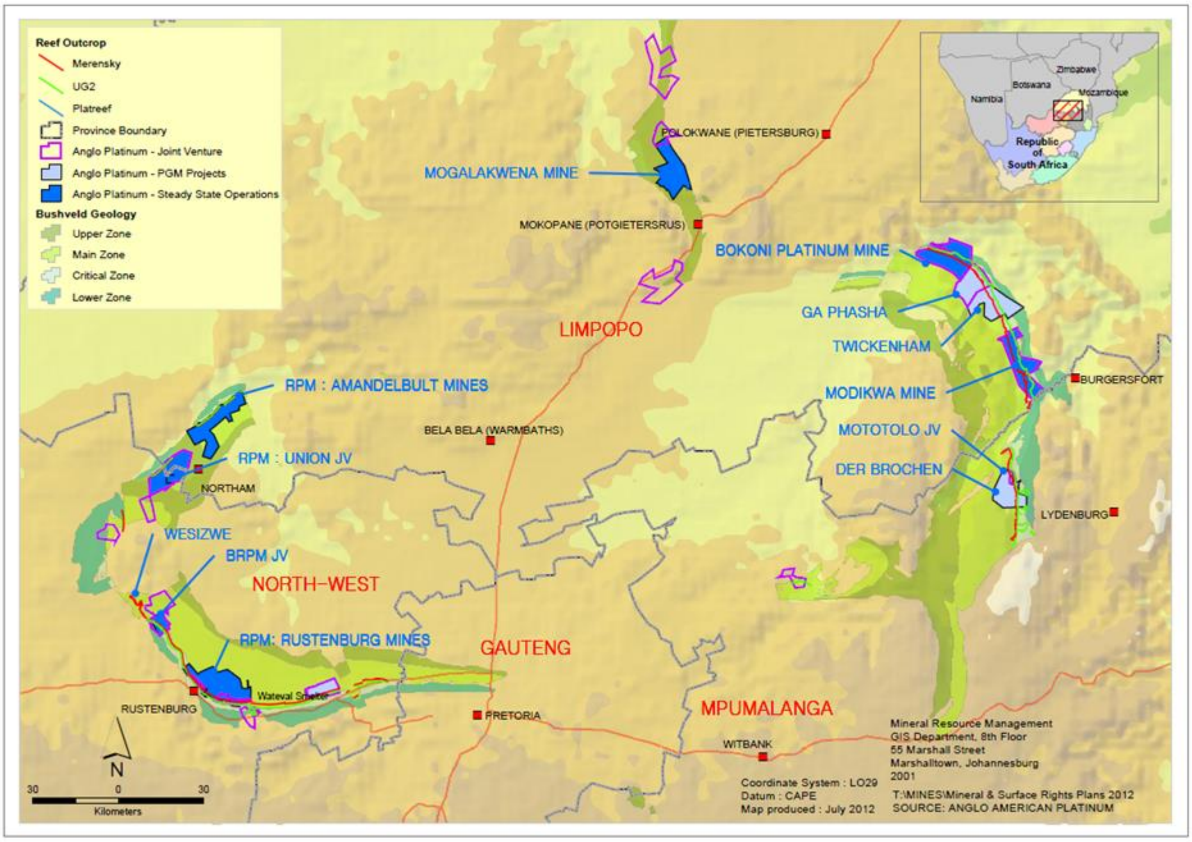

Figure 1: Location map of Anglo American Platinum Mines throughout the Bushveld Complex (Anglo American Platinum, (2012)).

1.2. Background

This research is focused on the geostatistical evaluation of a tailings dam grade estimation due to the monetary value or economic potential that can be realised if the dam is reprocessed. Tailings are comprised of discarded and unconsolidated mineral particles that were crushed and grinded from ore material during processing to extract valuable minerals. Total extraction (100% recovery rate) is highly unlikely with the current level of technology;

therefore, not all valuable minerals are currently separated from the gangue minerals and concentrated at the plant. The remaining value or economic potential is inversely proportional to the recovery rate of the concentrator plant. Thus, the lower the recovery rate (%), the higher the economic potential of the tailings derived or sourced from that plant.

The unrecovered value in tailings has influenced strategies of many companies to have tailings resources as part of their portfolio. A considerable portion of reserves of such companies is derived from retreating tailings dams. DRDGOLD, Anglo American Platinum, De Beers and Petra Diamonds are some of the companies that are retreating tailings, especially from olden days when plant processing was less efficient.

DRDGOLD is one of the world leaders in the recovery of gold through retreatment of tailings, currently retreating historic surface tailings of gold mines in the Witwatersrand basin, Johannesburg. It literally has up to three main plants dedicated specifically to retreating tailings. Tailings resources currently contribute massively to its total resource base. Table 1 clearly indicates that tailings contribute over 59% of their measured gold (Moz) resources, 62% of their indicated and 13% of their inferred gold (Moz) resources. That works out to be over 23% of their total or combined gold resources (Moz) (DRDGOLD, (2016)).

Table 1: DRDGOLD Resources published in 2015 Integrated Report (DRDGOLD, (2016)).

Surface (tailings)	**Measured**	**Indicated**	**Inferred**	**Total**
Tonnes (Mt)	159.735	569.104	822.121	1 550.960
Gold content (Au tons)	48.068	156.069	164.424	368.561
Gold (Moz)	1.545	5.018	5.2886	11.8516
Total resources (surface + Underground)				
Tonnes (Mt)	164.387	581.05	982.742	1728.179
Gold content (Au tons)	81.411	251.932	1244.597	1 577.940
Gold (Moz)	2.617	8.100	40.014	50.731
% (Gold Moz) of tailings over total resource owned				
Percentage	59.04	61.95	13.22	23.36

In the diamond industry, Ekapa Minerals, an investor consortium comprising Ekapa Mining and Petra Diamonds, had recently bought Kimberly Mines, including the tailings mineral resources (TMR) from De Beers (De Beers, 2016)). The Kimberly tailings operation will have a resource of 2.8 million carats at a grade of 11 carats per 100 tonnes (cpht) for the first three years, and then dropping to 6 cpht. There are a further 97 million tonnes of these deposits containing an estimated 4.4 million carats. In total tailings resources contain an estimated 7.2 million carats (BDlive, (2016)).

Furthermore, in the Platinum industry, Anglo American Platinum is one of the companies that are actively involved in retreating tailings resources for additional PGE ounces. The company has been producing significant platinum ounces from a tailings retreatment plant for the past five years, results of which are declared in their 2015 annual results. Table 2 is the production of 4E (Pt, Pd, Rh & Au) metals from tailings retreatment, as sourced from 2015 Annual results (AMPLATS, (2015)).

Table 2: Anglo American Platinum 4E production from tailings (AMPLATS, (2015)).

Year	**2015**	**2014**	**2013**	**2012**	**2011**
4E (000 oz)	44.8	50.4	59.7	48.3	41.5

It is not only South African companies or operations that are evaluating tailings resources for potential retreatment and additional production; mining houses abroad are also assessing their tailings deposits. SRK Consulting carried out resource estimation on Elsa tailings for ALEXCO Resource Corporation in Canada, in June 2010. Elsa Tailings resources were classified as Indicated (Table 3). SRK hold the opinion that estimation was based on detailed and reliable exploration data, with boreholes spaced closely enough for geological and grade continuity to be reasonably assumed (SRK Consulting, (2010)).

Table 3: Mineral Resource Statement* for the Elsa Tailings Project, (SRK Consulting, (2010)).

Category	**Quantity**	**Grade**				**Contained Metal**	
	[Tonnes]	**Ag [gpt]**	**Au [gpt]**	**Pb [%]**	**Zn [%]**	**Ag [oz]**	**Au [oz]**
Indicated	2 490 000	119.0	0.12	0.99	0.70	9 526 000	9 600

** Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures have been rounded to reflect the relative accuracy of the estimates. Includes all blocks in the block model and effectively reported at a 50 gpt silver cut-off grade assuming metal prices of US$17 per troy ounce silver and US$1,000 per troy ounce gold, silver recovery of 85% and gold recovery of 35%. Lead and zinc values are not considered.*

Although Mineral Resources in Elsa tailings were not converted to reserves at the time, due to economic non-viability, it does show that tailings are playing or could play a crucial role in supplementing main stream ore bodies for companies all over the world. From the background provided above, it is obvious that resource estimation of tailings dams is becoming more relevant. Therefore, this emerging field of tailings resource estimation requires reliable data and robust estimation methods and must be carried out by a Competent Person (CP) in the field of resource estimation.

The resultant estimation reports, apart from the eventual short term and long term mine planning, will be the base on which business decisions on whether to incorporate tailings reserves into company portfolio or not, are anchored. In addition, the resource report of listed companies that have tailings resources making a material contribution to their overall resources and reserves, must fulfil statutory requirements enforced by the stock exchanges and any other regulatory bodies. Investors (Shareholders) and their Advisors are likely to do an in-depth review of any estimation report of interest when conducting due diligence, further prompting robust and high quality evaluation report on those tailings resources.

Grade estimation of tailings dams, although becoming very common, is not yet as wide spread as *in situ* ore body estimation. Details of such resource evaluations are not readily available or accessible for research and reference purposes. Therefore, any research that focuses on tailings resource estimation would be of practical value. An opportunity therefore exists for this research report to highlight challenges associated with mineral estimation of tailings resource.

The aim and focus of this research is to investigate the application of fundamental principles of geostatistics in evaluating a mechanically deposited ore body, a tailings mineral resource.

1.3. Literature Review

Estimation of tailings dams, similar as with any other *in situ* ore body estimation, is performed through the application of geostatistical tools (techniques), of which the kriging approach is widely used by many practitioners in resource evaluation.

Geostatistics has evolved from the “Theory of Regionalised Variables” in the hands of French Mathematician and Mining Engineer George Matheron, of the Paris School of Mines (Clark, (2001)).

In the early 1960’s, Matheron developed a general solution to the problems of local estimation that built upon an empirical solution developed by the South African Mining

Engineer Danie G. Krige. To honour Krige's pioneering contribution in the field, Matheron named the new technique he developed "Kriging" (Vann, (2008)).

The application of Geostatistics to the estimation of ore reserves of an *in situ* ore body, in mining, is probably its most well-known use. However, it has been emphasized time and again that the estimation technique can be used wherever a measure (sampling) is made on a continuous variable at a particular location in space (or time); that is, where a sample value is expected to be affected by its position and its relationships with the surrounding or neighbours (Clark, (2001)).

Geostatistics may not be used in situations when there is no meaningful value of a variable at every point in space, within the region or area of interest. That is, when points or samples represent merely the presence of events or a phenomenon (e.g. crime, random people, volcanoes, buildings etc.). In these instances standard spatial estimation methodology of interpolation (estimation) might not be appropriate (Brusilovskiy, (2009)).

Geostatistics has evolved from statistics and it was developed to solve or enhance estimation of *in situ* gold grade: The application of statistical methods to ore reserve problems was first attempted in the 1950's and 60's in South Africa. The problem was that of predicting the grades within an area to be mined from a limited number of peripheral samples in development drives in the gold mines. Witwatersrand gold values are notoriously erratic, and when plotted in the form of a histogram, show highly skewed distribution with a very long tail into the rich grades. Normal (Gaussian) statistics theory does not handle such distributions unless a transformation is applied first. H.S. Sichel applied a log-normal distribution to the gold grades and achieved encouraging results. He then published formulae and tables to enable accurate calculations of local averages for log-normal variables, and confidence limits on those local averages. Three major drawbacks exist in the application of Sichel's 't' estimator:

- The 'background' probability distribution must be log-normal,
- The samples must be independent,
- There is no consideration taken of the position of the samples – all are equally important (Clark, (2001)).

However, the technique proved very useful in the gold mines, especially since some measure of reliability of the estimator was provided. It also laid the base for further statistical work by providing the conceptual framework necessary, that is, by assuming the sample values came from some probability distribution. At this stage, it was assumed that all the samples (in a specific area) came from the same probability distribution – a log-normal one – and this assumption is known in ordinary statistics as 'stationarity' (Clark, (2001)).

Subsequently, attempts were made to incorporate positions and spatial relationships into the estimation procedure. Two things seemed sensible: there should be 'rich' and 'poor' areas within a deposit; and there should be some sort of relationship between blocks or samples reasonably close to each other. To be able to deal with this relationship, methods such as 'Trend Surface Analysis' and 'Polynomial Trend Surface' were developed. Both methods have one thing in common – the basic assumptions about the statistical characteristics of the deposit. These assumptions have been extended from the 'stationarity' one, by stating that the sample value is expected to vary from area to area in the deposit. Some areas are expected to be rich, some to be poor. This expectation can be expressed as a reasonably smooth variation, either by a smoothed map or a relatively simple equation. Round about this trend there is expected to be random variation. That is, the value at any point in the deposit is supposed to comprise:

- A fixed component of the trend (which is probably unknown), and
- A random component following one specific distribution.

Thus, the stationarity has been shifted one step; the expected grade may vary slowly, but the random component is stationary (Clark, (2001)).

The two aspects (components) of the regionalised variable were further explained graphically in Figure 2:

- Structured or regionalised aspect – that accounts for large-scale tendencies of the variable (solid line),
- And a random aspect – that accounts for local, small-scale irregularities (dotted line) (Vann, (2008)).

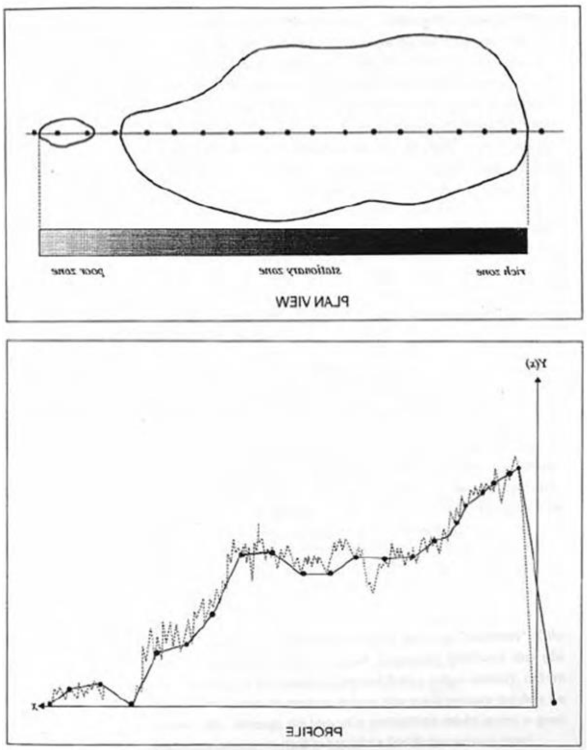

Figure 2: Random and structured aspects of a regionalised variable (Vann, (2008)).

It is evident from the above background and literature review that the development of estimation techniques or geostatistics was directed mainly to *in situ* ore bodies. Gold reefs in the Witwatersrand basin (super group) in South Africa were the initial focal point. Many practitioners still loosely translate or associate resource estimation (geostatistics) to *in situ* resources.

Furthermore, resource estimation is defined as the process of creating a three-dimensional reflection of *in situ* mineralisation based on sparse samples, as well as understanding of the geological extent in which estimation is being performed, together with geostatistical principles (Coombes, (2008)).

What goes together with estimating resources of an *in situ* ore body is geological understanding of that ore body. Geological processes that deposit mineralised reefs, pipes or fissures are medium to large scale phenomena that will mostly be consistent in regional characterisation of that specific ore body. Geological modelling, on the other hand, deals with local characterisation and delineation of the ore body into portions that are similar or closely related to each other, for accurately estimating grade of the ore body. Some of the critical features in the characterisation of the ore body into geological model generally relate to geometry: stratigraphic contacts, folding, location of faults and discontinuities, identification of vein orientations etc. Knowledge of a genetic link between assayed elements (Au, Pt, As, Pb, Ag etc.) may also be a very useful part of the model (Vann, (2008)).

One of the most important assumptions made in the estimation process is that of 'stationarity'. In geological modelling, stationarity decisions involve sub-setting (or regrouping) data such that the resultant data sets are correctly used for statistical analysis and estimation. The practical consequence is to build geological models for the purposes of grade estimation by paying attention not only to the geology, but also to the grade distribution. This is borne from the fact that the continuity of mineralisation maybe quite distinct from continuity of its geometry ("geological continuity") (Vann, (2008)).

The above explanation of "stationarity" emphasizes the importance of geological model and how critical it is in resource estimation.

It is further alluded that geology is the foundation of any accurate resource model. It is viewed to contribute as high as 90% of the accuracy of a resource estimate by some Mineral Resource Practitioners. The better the geological model, the simpler the resource estimation, and the simpler the mathematics required to generate a representative resource model (Coombes, (2008)).

The importance and relevance of geology in resource estimation, is that grades of a block are estimated based on the nearest sample data. If the grade of that block is to be relevant, then the samples it is based on must be relevant. In other words, there is a crucial need to identify the populations of interest that are relevant to the block that needs to be estimated. This means the population boundaries that constrain the samples need to be understood (Coombes, (2008)).

Building geological domains means:

- using one's understanding of the geological controls on mineralisation to create the limits of each mineralisation population,
- using statistical tools to validate interpretation of the mineralised populations,
- and understanding and defining domains before creating three dimensional envelopes of the populations (Coombes, (2008)).

The understanding of the background and development of geostatistical evaluation techniques regarding an *in situ* ore body, and the importance of a good geological model (geological controls) has been thoroughly assessed and put into proper context for the Case Study tailings dam in this research.

1.4. Hypothesis

Variography conducted on Elsa Tailings Project revealed that decent spatial continuity exists in Gold, Silver and Zinc grades (SRK Consulting, (2010)). Moreover, Platinum ounces currently being produced by Anglo American Platinum in Table 2 (Section 1.2) are mined from two tailings dams (different from Case Study tailings dam), where geostatistical evaluation techniques were used to perform evaluation of the tailings resource (Anglo American Platinum Internal Report, (2010)).

Therefore, it is evident that companies that are mining or considering tailings as resources are evaluating those tailings dams through geostatistical techniques because mineralisation continuity (spatial correlation) of a reasonable degree exists in tailings dams.

Further investigation of this hypothesis will determine or affirm whether it is appropriate to assess a platinum tailings resource using geostatistical resource estimation. Should it be appropriate, it is anticipated that a measure of the confidence associated with the tailings resource estimates can be determined; which would assist with tailings mineral resource classification as with any estimation method conducted on an *in situ* ore body.

Drilling and sampling to gather relevant grade information is the most practical and reasonable way to investigate the hypothesis, and to determine whether the application of geostatistical estimation techniques, specifically kriging, is an appropriate evaluation method on tailings resources.

Once accepted as appropriate and relevant, grade and tonnage estimates of any low grade man-made ore body, such as tailings dams, low grade rock dumps and muck piles can be derived using the methodology of this research.

1.5. Key Questions

Undertaking this research answered the key questions related to the understanding of tailings resource estimation; research objectives were also formulated in line with these key questions:

1.5.1. Application of the Principles and Theories of Geostatistics on Tailings Deposits

To test the hypothesis a primary question is: "Are the general principles and theories of geostatistics relevant and appropriate to tailings deposits without significant changes needed to these principles?" Basically, what are the constraints or freedom of applying geostatistical estimation techniques on tailings dams, from drilling and sampling to geological modelling, and grade estimation?

1.5.2. Impact of Feeding Selectivity on the Random Component of the Data

What role does the feeding strategy of plant play in the grade profiling of the tailings deposit? The feeding strategy can involve: selecting only high-grade ore; strategic blending of very low-grade and high-grade ore; feeding similar types of rocks or mineralogy, or even blending oxidised (weathered or altered) rocks with fresh rock or ore. It is preferable to feed plant with ore of related metallurgical or geometallurgical properties at a given time. Therefore, does this selective feeding of ore to the plant affect, and to what extent, the random component of grade distribution of the tailings dam? What level of bias gets introduced in the distribution of tailings Platinum Group Metals (PGMs)?

1.5.3. Impact or Influence of Mechanical Deposition of Tailings on Grade Continuity

What impact does mechanical deposition of tailings have on grade continuity? Do tailings have sufficient spatial correlation for accurate interpolation of grade? Variography analysis will provide answers to this question.

1.5.4. Impact of Processing on Grade distribution

Crushing and milling of the ore involves homogenisation of one or more rock types, different grade profiles and possibly different geological facies. The original PGM distributions and small scale (micro) variation of the ore body is smoothed out (erased). The resultant grade profile in the tailings dam would not necessarily reflect the original profile that existed in the *in situ* ore body. The different *in situ* grades can possibly be altered during the metallurgical extraction process to a new and highly uniform grade profile.

Table 4 below is an illustration of a possible scenario, wherein several samples could be collected from the tailings and have almost identical grade of 0.7 g/t, even though those tailings were sourced or milled from a wide range of geological facies and *in situ* grades. The similarity in tailings grade being a function of plant recovery. Therefore, what is the impact of that likelihood – and significance of that impact – on the distribution of grade in tailings? Furthermore, is the grade continuity affected or influenced in anyway by this phenomenon?

Table 4: Illustration of the possibility that different *in situ* grades can result similar tailings grades.

***In situ* grade (g/t)**	**Recovery %**	**Recovered (g/t)**	**Tailings grade (g/t)**
3.55	80.00	2.84	0.71
4.80	85.40	4.10	0.70
6.02	88.13	5.31	0.71
1.92	63.15	1.21	0.71
1.50	53.33	0.80	0.70
1.22	42.66	0.52	0.70
1.10	36.36	0.40	0.70

1.5.5. Overall Accuracy and Reliability of Estimated Results

Does the accuracy of a geostatistical evaluation of tailings dam compare or even better any other method of estimating the average grade and tonnage of a tailings dam? Is the accuracy within reasonable and acceptable statistical margins?

Modern mines have records of average grade and tonnage of their tailings dams as recorded by metallurgists, in this case the question will be to determine which figure is more reliable and comprehensive: metallurgical accounting or geostatistical estimated results based on drilling of the tailings dam? The ultimate question would be: is the geostatistical evaluation of tailings dams of acceptable quality and accuracy for confident business decision making?

1.6. Research Objectives

This research will specifically address platinum tailings dam estimation; which is essentially a man-made ore body. A Tailings Dam from Anglo Platinum has been investigated as a case study, but its actual identity and exact location have not been disclosed due to the confidentiality agreed upon. Consideration of this case study will enhance the understanding of the differences (or similarities) in estimation methodology between an *in situ* platinum resource and a tailings dam. The research will therefore focus on challenges and limitations experienced during tailings dam estimation, as well as reliability and accuracy of the estimated results. The objectives will be addressed by the following:

- Identification and confirmation of drilling and sampling best practices for optimal estimation and evaluation of tailings dam;
- Analysis of deposition of the material within the tailings dam setting and the subsequent establishment of geological zones;
- Analysis of the impact or influence of mechanical deposition on mineralisation and grade continuity:
  - Variogram ranges,
  - Nugget effect and its implication, and
- Assessment as well as rating the accuracy (quality) and reliability of Geostatistical estimation on tailings dam; an unconventional and man-made ore body.

CHAPTER TWO: RESEARCH METHODOLOGY

The methodology that was followed to conduct the research was a standard procedure that is practiced when resource estimation is performed: drilling, sampling, assaying, statistical and variography analysis, grade estimation, and resource (grade) model validation.

Drilling of this case study project was supervised by the author between September 2011 and April 2012. Sampling was also carried out solely by the author of this research report. Samples were sent to the laboratory then, thus, assay results (data) were ready and available for analysis during the proposal stage of this research. Chapters on drilling and sampling are a reflection of the steps taken and followed by the author during data collection.

2.1. Sonic Drilling

Sonic drilling was chosen as a suitable technique to conduct the drilling. It is a drilling technique which is widely used in geotechnical and structural engineering investigations. It has been adopted successfully as a better exploration tool for tailings dam analysis or evaluation. It is a rapid soil penetration technique which relies on the Law of Inertia and Bingham's Law of fluidizing porous and unconsolidated material (Ewing, (2015)).

Sonic drilling is a dry drilling method which was preferred over other dry drilling techniques, such as auger, because of its proven ability to provide good coring recovery of a tailing profile. This drilling method, therefore, can be practised without any drilling fluids as opposed to the conventional core drilling where fluids are a prerequisite.

In Sonic drilling rods are vibrated at a frequency range of between 100 Hz and 200 Hz to liquefy soil particles immediately around the rods, thereby reducing friction to allow penetration of the rods. Vibration motion is supplemented by a rotation movement to give an additional cutting power to the rods, when needed (Ewing, (2015)).

In this case study, 3273 metres were drilled from 77 boreholes. The depths of the boreholes (tailings dam thickness) averaged 42 m and were drilled on a grid pattern of 100m x 100m. In cases where 100 m could not fit, the grid was slightly adjusted or reduced to between 90 m and 95 m. All the boreholes we drilled at -90° inclination. Furthermore, no deviation was experienced in these short boreholes. Therefore, downhole survey was not needed.

2.2. Logging and Sampling

Logging and sampling are closely dependent and related activities that are conducted instantly and, to a degree, concurrently in sonic drilling. Logging and sampling in the Case study project, as such, were conducted at the site immediately after drilling run was completed and recovered, before samples were delivered to the core yard for dispatch to the laboratory.

Drilling was conducted with three metre rods, thus three metre drilling runs. The core barrel was slightly longer than three metres (± 4.5 m) to accommodate three metres being drilled and material caving from the side wall (contamination). The inside diameter of the core barrel, same as that of drill bit, was 9.9 cm. The diameter of the sampling bag or tubing, however, was required to be bigger than that of the core barrel due to the nature of collecting tailings in sonic drilling.

2.2.1. Sampling

Sample collection was performed by sliding plastic tubing around the core barrel (Figure 3), and then releasing the sample into the bag, with bottom end of the bag sealed. The average diameter of sampling bags used was 13.2 cm.

Figure 3: Demonstration of sample collection and logging preparations

The difference in diameter of the sampling bag to that of the core barrel meant that three metres drilled in each run would be collected to make short, thicker sample. This difference necessitated the use of a formula to calculate the new length of the bigger sampling bag.

The formula that was used to reconcile (account) for sample recovery is:

$$Volume = \pi r^2 \mathrm{L}$$

Core barrel (inside) diameter = 9.9 cm,

Radius (r) = 4.95 cm

$\pi = 3.14159$

Length of 3 m drill run (L) = 300 cm

Volume drilled is therefore:

$$Volume\ (V) = \pi r^2 \mathrm{L}$$

$$= 3.14159 \text{ x } (4.95)^2 \text{ x } 300$$

$$= 23093 \text{ cm}^3$$

The volume drilled as calculated above was then used to calculate the length expected in a sample bag with a 13.2 cm diameter.

Sample bag diameter (d) = 13.2 cm

Radius (r) = 6.6 cm

Therefore, from $V = \pi r^2 \mathrm{L}$

$$\mathrm{L} = \frac{V}{\pi r^2}$$

$$= \frac{23093.06}{3.14159 \text{ x } (6.6)^2}$$

$$= 169 \text{ cm}$$

The calculation confirmed that 3 m drilled by a core barrel of 9.9 cm in diameter will be collected into 169 cm long sample bag with a diameter of 13.2 cm.

Therefore, the average expected new length of one sample was 169 cm/3 = 56 cm

The above calculation was crucial and necessary to ascertain recovery performances. That is, material that slid back from the core barrel (loss) and material that were gained due to previous run losses (recovered losses/gains), or material that caved in from the side wall (contamination).

2.2.2. Logging

Immediately after the sample was extracted into one long sample bag, the next step was to establish the amount of contamination, if any, that had to be discarded. The following guidelines or parameters were assessed to determine the contamination:

- **Compaction of material**

  The *in situ* material was often found to be highly compacted whereas the contamination comprising material shaved from the sidewalls of the borehole or caved into the hole was loose.

- **Colour contrast between *in situ* and the contaminated material**

  As the hole progressed deeper, the colour of the tailings was observed to change from a very light grey to a dark grey. Therefore, the actual sample and contamination would, from time to time, have different colour contrast.

- **Material distribution**

  The tailings material was usually homogeneous or layered. The top most contaminated portion of each run was often mixed and lacked these properties.

- **Moisture content**

  As drilling progressed deeper, especially in the wet portion of the dam, the moisture content difference between deeper *in situ* sample and contamination from shallow and dry material would be visible. Furthermore, liquefied slurry on top of solid tailings being drilled in a wet zone was often due to water accumulation; as such it was treated as contamination.

2.3. Assay Analyses

Assay analyses have been carried out by the appointed laboratory that met the requirements and specifications, and results were ready and available for analysis during the drafting of the proposal for this research project.

Rh was only conditionally analysed in cases when the 3E (Pt, Pd and Au) grade was greater than 1.5 g/t. At the lab, samples were homogenized and separated into four splits (A, B, C, and D). The latter two splits were returned for storage or future reference.

Analyses carried out on the first two sample splits include:

1. Assaying for Platinum group metals (PGMs) and Au using –
   (a) Ag prill for Pt, Pd and Au;
   (b) Pd prill for Rh.
2. Base metals (Cu and Ni) using XRF method.
3. Particle size distribution (PSD).
4. Moisture content.
5. Weight measurement: wet and dry
6. And Specific Gravity (Density) determined by a gas Pycnometer.

2.4. Data Validation and Statistical Analyses

2.4.1. Data Validation

Data validation had been carried out to establish and remove outliers in the data. Collar positions were also validated to ensure that all boreholes plot on the correct position and within the drilled grid, which in turn promotes accurate grade estimation.

Data validation was concluded by confirming the following:

- Collar elevation, inclination and bearing;
- Logging consistency and errors;
- As well as sampling overlaps and consistency.

2.4.2. Statistical Analyses

Statistical analyses were conducted on the grades of all six variables of interest – 3E (2 PGEs + Au), Pt, Pd, Au, Cu and Ni – to establish underlying distribution of the grade. Analysis was also extended to the seventh variable, namely density measurements; a crucial parameter used during tonnage estimation.

Sampling was carried out in 1m intervals, but few samples were slightly longer that 1m, especially the deepest sample in some boreholes. Borehole data was composited into 1m composites before statistical analyses were carried out, to ensure that those few samples longer than 1m are all standardised.

It is essential to perform downhole compositing to ensure samples have comparable influence on the statistics. Samples that are collected over variable lengths have a risk of introducing bias into the data being analysed. The effect of compositing is to weight sample grades according to the corresponding interval length (Coombes, (2008)).

Histograms, probability plots, and tables with set of statistical parameters were generated during statistical analyses to understand distribution of each variable in the data sets.

2.5. Geological Modelling

Geological modelling is a computer aided exercise performed to try and delineate an ore body into blocks of similar or identical characteristics. It is a critical step, before geostatistical analysis and estimation, to improve the accuracy of the estimation.

Three variables that can assist with identifying geological zones underlying the tailings resource were analysed, namely: lithology (tailings material), PGE grade (3E) and particle size distribution (PSD). A fourth variable, water content, which does not necessarily have any relationship with grade, was also analysed for comparison and guidance purposes.

2.6. Analysis of Grade Continuity (Variography)

Experimental semi-variograms and subsequent variogram modelling (models) were performed through Snowden Visor (Supervisor) software for all seven variables (six metals and density). Variography analysis is one of the prerequisite steps before grade estimation can be carried out. Moreover, the accuracy (margin of error) of estimation is partly a function of variogram model. The descriptive data analysis together with the variography exercise, led to the understanding of the distribution and continuity of mineralisation of the Case Study tailings dam.

The fundamental question of this project was: whether "mineralisation continuity" in the Case Study tailings dam, which is man-made and mechanically deposited, meets the minimum requirements for geostatistical evaluation? What is the relationship between values at sampled locations to values at unsampled locations? Where $Z(x)$ is the value of the sample at location x, does the assumption for stationarity hold for tailing dams? That is, the probability distribution of: $Z(x_1), Z(x_2), Z(x_3), \dots, Z(x_n)$ similar to the probability distribution of, $Z(x_1+h), Z(x_2+h), Z(x_3+h), \dots, Z(x_n+h)$?

To understand spatial correlation of all seven grade variables at very short ranges in the tailings dam, attention was also given to the nugget effects of all the variables of interest.

2.7. Grade (Resource) Estimation

Descriptive data analysis and variography carried out prior to grade estimation confirmed Ordinary Kriging as the appropriate geostatistical method to perform estimation, because there was no indication of any trends in the data. Grade estimation was carried out using Datamine software package. Grades were interpolated into a 5m x 5m x 5m block model, and all seven grade variables of interest were interpolated into each cell.

2.8. Grade Model Validation

Model validation was the final step (exercise) to answer a key question: whether geostatistical evaluation of the Case Study tailings dam would be reliable and accurate such that the final resource model can be used without any reservation?

The estimated values (model) were validated against known samples – that is comparison of drillholes (input) versus results – to determine the accuracy and reliability of the grade model. To standardise support sizes, borehole data was composited into 5m composites before it was loaded, bringing it in line with the 5m x 5m x 5m resource model.

Validation was performed in 4 steps:

- **Cross-validation** – It is a tool to test and optimise parameters that are set to conduct estimation. The process involved removing each original grade value, one at a time, and estimating it with the set parameters and the remaining/neighbouring samples. The difference between the two set of values was a forecast or presumption of the accuracy achievable from the set parameters.
- **Visual validation** – Grade profiles between boreholes (5 m composites) and resource model, particularly cells around borehole traces, were visually verified to confirm similarities between the two data sets.
- **QQ Plots** – The behaviour of two data sets was also verified through QQ Plot graphs.
- **Relative Histograms** – The similarity in the shapes of the distribution of the estimated model and original data was further validated through relative histograms.

CHAPTER THREE: TAILINGS DAM SETTING

3.1. General Tailings Dam Overview

Tailings dams are man-made structures designed as a storage site for tailings produced during processing of ore. Certain safety protocols guided by environmental legislations and structural integrity are observed during the design and construction of tailings dams. Poor design and/or construction can lead to failure or bursts of the dam which could result in mud rushes or landslides. Consequences of which could vary from environmental contaminations (pollution), damage to properties and serious injuries or even death.

Construction of a tailings dam is carried out in a series of benches (Figure 4) which are pumped over time, in some cases throughout the life of mine, and are laid on top of one another. The bench below (underlying) is always strategically longer and bigger than the overlying bench on top.

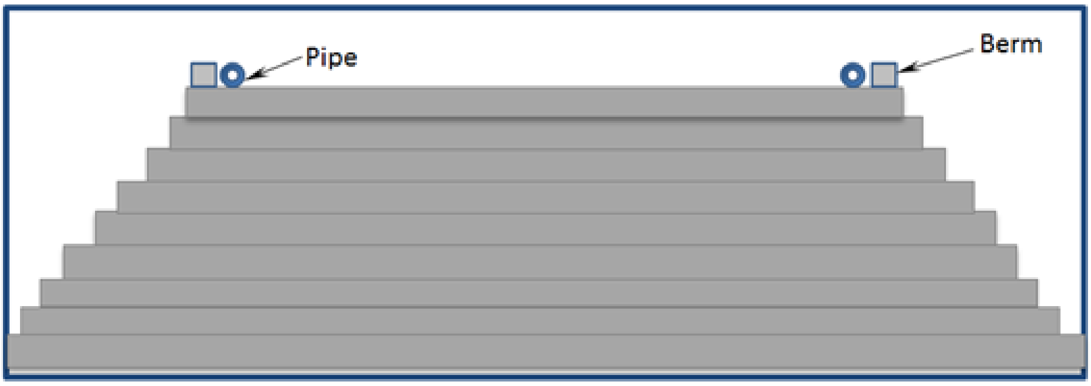

Figure 4: Cross-sectional view of tailings dam benches, berm and pipes.

Each bench, which could be as thick as several metres, represents a major pumping (construction) cycle. A berm is constructed at the edge of each bench to contain and direct tailings to flow inward towards the centre (penstock). Pipes feeding the dam are laid around the outer edge (berm) of the dam, with several valve openings faced inward to the centre of the dam. Depending on the size of the dam or the bench being pumped, another set of pipes are usually connected and laid transversely to feed the middle portion of the dam (Figure 5).

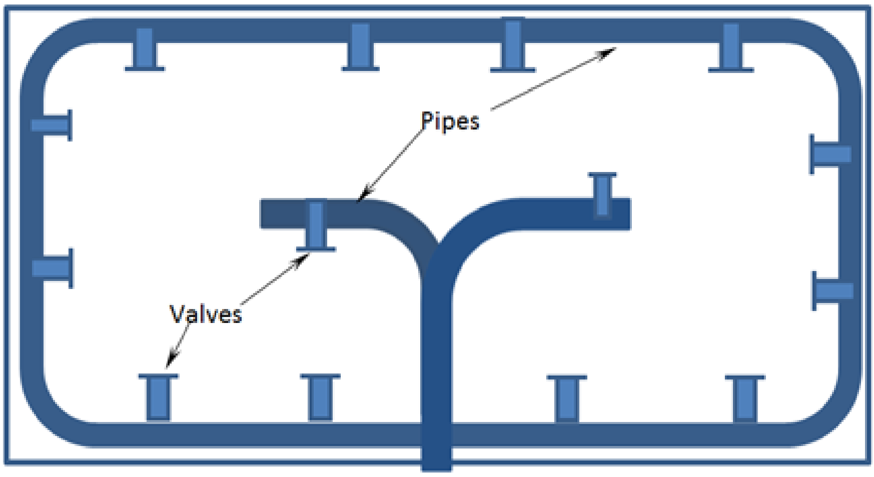

Figure 5: Plan view of pipe layout pumping a bench.

3.2. Case Study Tailings Dam

The Case Study Tailings dam (Figure 6) was also constructed in a series of benches laid on top of one another, and is similar to the design described above.

Figure 6: Arial view of the Case Study tailings dam

The dam has reached its operational life (designed capacity) and no further pumping of new benches is occurring or expected to ever happen in the future. The average dimensions at the base of the dam are 1200 m north-south (length) and 1050 m east-west (breadth). The top of the dam has the shortest span that averages at 970 m long and 910 m wide.

The design resulted in the cross-sectional view of the dam strongly resembling a "Trapezoid" (Isosceles) shape; two parallel sides of which the longest corresponds to the bottom of the dam (wide base) and the shorter side to the flat top (narrow top). The angled (jagged) lateral sides represent retreating benches, see Figure 4.

The thickness (height) of tailings varies from 39 m in the south to over 46 m in the north. This variation can be attributed to the variation in topography of the footprint, with the overall difference reaching 7 m in the north. The elevation varies between 974 m and 981 m above sea level.

3.2.1. Composition

The composition of tailings material on the Case Study tailings dam was directly influenced by the type and lithology of the ore that was milled at the plant. Certain properties (features) of the dam, such as grain size distribution and layering, got influenced by external factors such as underlying design of the dam, layout of the pipes, pressure and rate at which tailings were pumped at the time of deposition.

Layering experienced in the dam occurred because of pumping cycles and pulses that were repeated several times during construction of the dam. Moreover, sorting (particle size distribution) experienced is attributed to transportation of tailings from the plant and deposited into the dam as mineral particles-carrying liquid, which was subjected to natural laws of physics of separating coarse from fine particles. Sorting is therefore a direct result of the ability of coarser and heavier particles to resist stronger currents that carried them (existed) immediately after discharge, thereby settling quickly and much earlier next to the discharging pipes, before they could be carried further away to the middle of the dam. On the other hand, finer particles are easily moved by the dissipating and weaker currents. These remnant currents were able to carry the finer particles to the middle of the dam where the water became standstill. Drying up overtime then afforded these remaining particles to settle down and get deposited.

To understand local setting of Case Study Tailings dam, five characteristics that relate to composition and material distribution where assessed:

Lithology (Tailings Material) – For consistency Logging was conducted using the nomenclature defined in Table 5 as a reference (Anglo American Platinum, (2011)).

Table 5: Logging reference table for tailings material (lithology).

Lithology Type	
CH	Clay of High Plasticity liquid limit >50%
CL	Inorganic clay liquid limit <50%
GC	Clayey Gravel >12% fines
GM	Silty gravel >12% fines
GP	Poorly graded Gravel <12% fines
GRAVEL	Gravel 50% of coarse fraction >4.75mm
GW	Well graded gravel
MH	Silt of High Plasticity elastic silt, liquid limit >50%
ML	Inorganic silt liquid limit <50%
OL	Organic silt, organic clay liquid limit <50%
OTHER	Other Inserted QC
PT	Highly organic soils
SC	Clayey Sand >0.075mm and <4.75mm >12% clay fines
SCREE	Loose surface scree
SM	Silty sand with >12% silt fines
SP	Poorly graded sand <12% fines
SW	Well graded fine to coarse sand

From the perimeter of the dam where pipes were discharging, tailings were found to be mostly silty sand material (SM) with more than 10% silt fines. It gradually increases its clay content to clayey sand material (SC) or much finer silty sand, with considerable amount made of clay fines, towards the penstock area (middle) of the dam. The lithology distribution does also vary vertically, largely based on the composition of the tailings that was being pumped at a specific time or bench.

Material distribution (sorting) – Sorting was logged according to the nomenclature defined in Table 6. The deposition of tailings on different benches, with each bench made by multiple discharges of tailings at different times, resulted in the dam composed largely of layers of tailings with varying thicknesses. Although *in situ* compaction and layering was not always preserved during drilling and sample collection, individual layers were observed to vary from millimetres to between 5 cm and 10 cm from different holes at different depth. Tailings which were very loose and failed to preserve any layering were often classified as homogenous.

Table 6: Logging reference table for tailings sorting (Anglo American Platinum, (2011)).

Material Distribution (Sorting)	
FINESDOWN	Grain size decreases downwards
FINESUP	Grain size decreases upwards
HETEROGENEOUS	Large clasts in a finer grained matrix
HOMOGENOUS	Homogeneous - similar composition and grain size
LAYERED	Layered - alternating grain size or composition different.

Colour – Tailings in the dam are mostly grey in colour of varying shades. It varies between light grey, grey and dark grey. The light grey was usually associated with SC material and often found in the top 8 m to 15 m metres, at times slightly deeper in holes closer to the centre of the dam. A slightly different coloured SC material exists at the very bottom of the dam, between 38 m and 45 m. It has a unique light greyish-greenish colour, which could also be classified as yellowish green. Rarely, very thin layers of tailings were observed to be black. This was attributed to the UG2 tailings, which were intermittently and rarely pumped into Merensky-dominated tailings. SM material was usually grey to dark grey. It was also associated or commonly found in moist and wet zones.

Particle size distribution – was found to correlate to the pattern and trend observed on lithology distribution. Particle size is much coarser on the perimeter and becomes finest towards the centre of the dam. The reason for this behaviour, as for lithology, has been attributed to the ability of the coarser material to resist strong current that existed immediately after discharge, thereby settling early next to the discharging pipes, before finer particles are carried away to the middle of the dam.

Water content – The project was conducted over three of the four climate seasons of the year, and seasons have an influence on the level of the water table or moisture content of the tailings in general. Drilling started in spring, through summer and ended in autumn. Water content of tailings also varied slightly from location to location; holes that were drilled next to the perimeter generally have a thin wet zone and those drilled around the penstock had a thicker wet zone.

The water content of the dam was generally divided and classified into three:

- **Dry zone**: this is the top most zone where no moisture or very little and insignificant moisture was present (less than 15%), it started from surface and end at depth between 11 m and 18 m, depending on the position of the hole and season of the year.
- **Moist zone**: this zone starts from depth of between 11 m and 18 m and end at depth of between 33 m and 38 m. It also depends on the position of the hole and season. The water content in this zone was measured (assigned) to be between 15% and 50%. The highest water content (50%) was found at depth approaching the wet zone.
- **Wet zone**: this is the zone that existed below the water table and the tailings on this zone were completely saturated and submerged under water. This zone also varied from location to location and fluctuated with water table in different seasons. It started from depths of between 33 m and 38 m and extended to the floor of the dam, which is up to 47 m at the northern side of the dam.

3.2.2. Dam floor

The bottom of the Case Study tailings dam is primarily lined and protected by dark-brown clayey soil generally referred to as the “black turf”. The black turf helps to prevent water from the tailings percolating down into the natural water table and, together with drainage systems, prevent water with floatation chemicals and salts to seep into the surrounded environment. Whilst drilling, the black turf helped plugging in the tailings, which gave rise to good recovery in wet tailings by preventing it from falling (sliding) back into the hole.

Most boreholes that intersected the full length of the black turf, revealed that thickness generally vary between 30 cm and 180 cm, shown as CL in Figure 7. Two boreholes TD75 and TD77, in the middle of the south-eastern side intersected black turf with thicknesses of less than 30 cm. Two other boreholes, TD37 and TD42 on the north-western side, intersected the thickest portion of the floor lining (Figure 8). The thickness of black turf in that area was more than 2.2 m, yet those boreholes were stopped before the bottom contact could be intersected or exposed.

Black turf is generally underlain by a weathered norite or anorthosite of the bedrock. The thickness or the depth to which that bedrock is weathered varies between 5 cm and 30 cm.

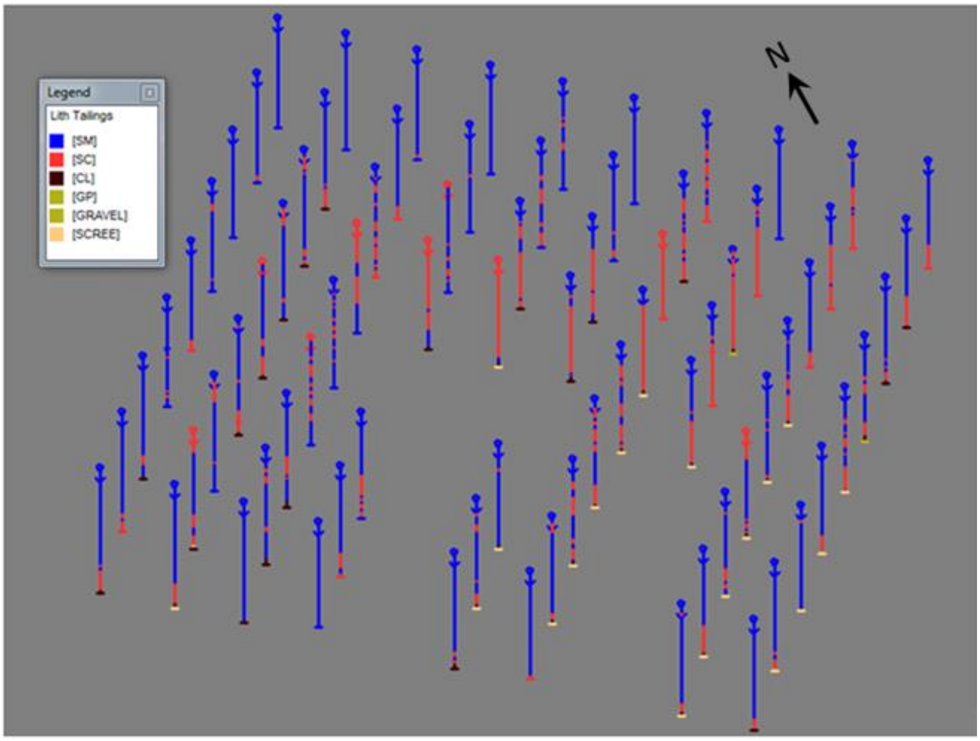

Figure 7: A 3D view of lithology across the dam. CL represents the black turf

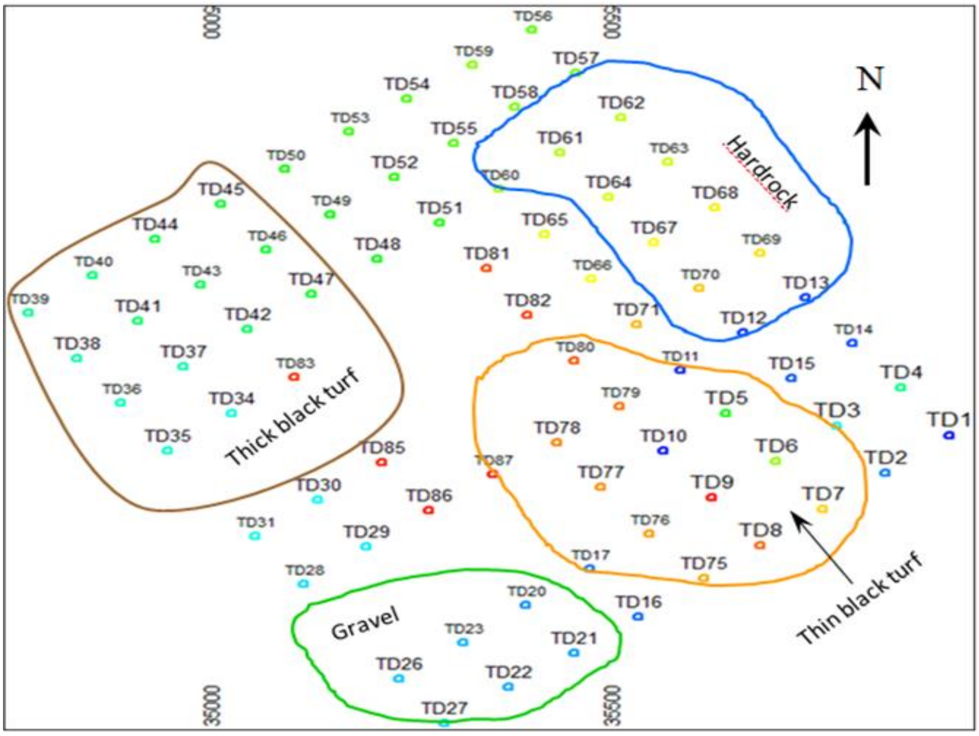

Figure 8: An aerial view of boreholes location with the Black turf thickness areas.

There are several holes that did not intersect any black turf or weathered norite but went and stopped on the hard bedrock. The machine could not penetrate any further because of the hard rock, yet the core barrel was not plugged by any of the materials, black turf or weathered norite. The area where the floor of only hard bedrock existed was observed in a zone at the middle and close to the edge on the eastern side of the dam, stretching from TD60 to TD62 and TD69 to TD70 (Figure 8).

Some of the boreholes, TD21 and TD20 (Figure 8), intersected gravel or very course sand at the bottom of the tailings, referred to as scree in Figure 7. This material has a distinct black colour which did not shine to the extreme of chromitite; it could possibly be road making gravel and tar, or granulated UG2 which is high in feldspars as internal waste.

The topography, hence the floor of the dam, is undulating and steeping to the north, its elevation varies from 975 meters above sea level (masl) on the southern end to about 981 masl at the northern end. That gave rise to the thickness of tailings to also vary from 39 m on the south to over 46 m on the north.

CHAPTER FOUR: DATA VALIDATION AND ANALYSIS

4.1. Data Validation

Data validation was performed on collar positions to verify and confirm that all boreholes are plotting on the correct position and within the drilled grid (Figure 9).

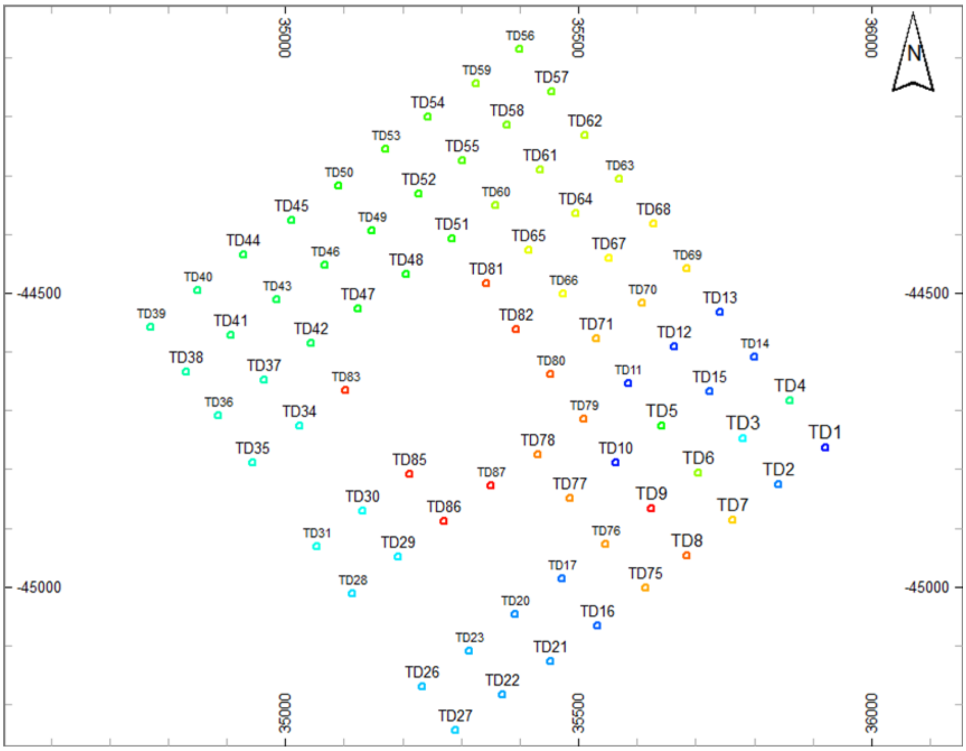

Figure 9: Confirmation of borehole collar positions and drilling pattern.

Validation was also performed to confirm the following:

- Collar elevation, inclination and bearing.
- Gaps in the borehole logs to ensure that they correspond to core losses.
- Stratigraphy and lithology was checked to ensure that tailings are not given to footwall and black turf (footwall) not incorporated into tailings.
- QAQC standards were inserted after every 10th sample and they all returned with their certified expected values or within acceptable ranges, as such sample assay results have been certified to be true representative of the tailings resource.
- Assay data were also verified to ensure that QAQC standards were not incorporated and used for grade estimation.

Further validation was performed on all grade and density values to identify outliers that might impact on the estimation results. Assay values that are extremely low and possibly below detection limit were identified and removed. Likewise, density values that are too light to suggest tailings being lighter than water or that imply tailings to be as dense as a solid rock was also identified and removed as outliers during the validation process. Table 7 below is a list of all samples that where flagged and removed as outliers.

These samples flagged as outliers were of insignificant quantity and therefore were not returned to the laboratory for re-analysis. These outliers represent less than 0.5% (10/3225) of assays on any given grade variable, but their impact on the mean and skewness was significantly higher. Mean grade for Pt (g/t), for example, was noted to change from 0.478 g/t to 0.564 g/t, which represent close to 20% movement of the mean by the 0.16% (5/3225) flagged samples.

Table 7(a): PGMs and Gold values flagged as outliers.

Variable	BHID	Sample ID	Grade (g/t)	Comment
4E	TD14	TD14/21	2.235	Too high
	TD43	TD43/40	2.755	Too high
	TD55	TD55/17	2.065	Too high
	TD85	TD85/23	7.175	Too high
	TD85	TD85/24	2.37	Too high
Pt	TD14	TD14/21	1.85	Too high
	TD43	TD43/40	2.11	Too high
	TD54	TD54/17	1.665	Too high
	TD85	TD85/23	6.165	Too high
	TD85	TD85/24	1.92	Too high
Pd	TD85	TD85/23	0.76	Too high
Au	TD36	TD36/2	0.005	Too small
	TD44	TD44/2	0.0075	Too small
	TD44	TD44/4	0.0075	Too small
	TD45	TD45/3	0.0075	Too small
	TD45	TD45/4	0.0075	Too small
	TD45	TD45/5	0.005	Too small
	TD50	TD50/2	0.0075	Too small
	TD53	TD53/12	0.0075	Too small
	TD53	TD53/3	0.0075	Too small
	TD56	TD56/2	0.005	Too small

Table 7(b): Base Metals values flagged as outliers.

Variable	BHID	Sample ID	Grade (%)	Comment
Cu	TD14	TD14/9	0.0047	Too small
	TD47	TD47/30	0.1291	Too high
	TD69	TD69/24	0.0035	Too small
	TD85	TD85/23	0.1963	Too high
	TD85	TD85/24	0.0454	Too small
	TD87	TD87/2	0.00485	Too small
	TD9	TD9/13	0.0045	Too small
Ni	TD47	TD47/30	0.186	Too high
	TD69	TD69/24	0.0617	Too small
	TD85	TD85/23	0.1373	Too high

Table 7(c): Density values flagged as outliers.

BHID	Sample ID	Density (g/cm^3)	Comment
TD38	TD38/12	1.008	Too small
TD38	TD38/30	1.006	Too small
TD42	TD42/1	0.931	Too small
TD42	TD42/3	0.882	Too small
TD45	TD45/2	2.621	Too high
TD53	TD53/6	2.683	Too high

4.2. Data (Statistical) Analysis

Statistical analyses were performed on all seven variables of interest that were to be estimated by geostatistical techniques (kriging), prior to data being used for variography and estimation processes. These analyses were essential to understand the behaviour of the data, most importantly to establish the underlying distribution of each variable. It also established the number of underlying populations in the data. Three analyses in the form of relative histograms, summary statistics (tables) and cumulative less than and greater than frequency distribution graphs were performed or plotted, results of which are discussed in the sections below.

4.2.1. Platinum (Pt) Statistical Analysis

The grade distribution of Pt is non-symmetrical, and displays a slight positively skewed distribution. The weight just slightly shifted to positive distribution due to the tail that intermittently stretches between 0.8 and 1.0 g/t, as can be seen from the relative frequency plot below (Figure 10).

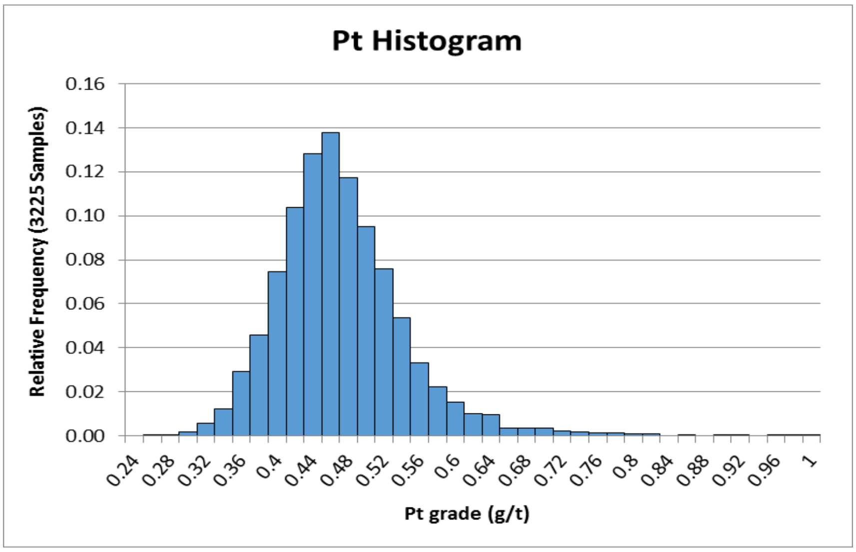

Figure 10: Platinum grade (Pt in g/t) Histogram

The classical statistics summary for Pt is presented in Table 8 below. The log-normal estimate is 0.4777 g/t, which is close to the average derived from the untransformed data 0.4783 g/t. The Skewness of 4 confirms the asymmetry. The kurtosis at 35 is lepto-kurtic, and confirms the high peak observed in the histogram; more than one third of the data is distributed around the centre.

Table 8: Platinum (Pt) Classical/Summary Statistics.

No of Assays	3225
Min (g/t)	0.25
Max (g/t)	1.565
Range (g/t)	1.315
Average (g/t)	0.4783
Variance $(g/t)^2$	0.008
Standard Deviation (g/t)	0.091
Coefficient of Variation(CV)	0.19
Skewness	4
Kurtosis	35
Log Average [ln(g/t)]	-0.752
Log Variance $[\ln(g/t)]^2$	0.026
Log-normal Estimate (g/t)	0.4777

The difference between the average Platinum grade calculated from the samples and the lognormal estimate is 0.0006 g/t or 0.125% which is considered negligible

The Pt cumulative distribution frequency as shown in Figure 11, confirms the conclusions reached from the summary statistics and histogram. The narrow short curve on lower grade side indicates a short tail; a single slope in the middle indicates that data possibly comes from a single population, and slightly wider or longer curve on the high-grade end corresponds to a slightly longer tail as reflected by relative frequency histogram.

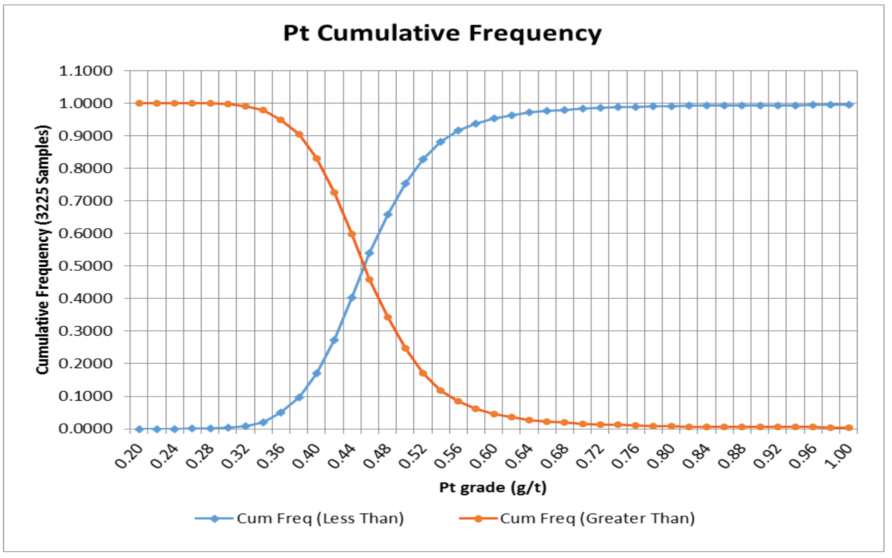

Figure 11: Platinum (Pt g/t) Cumulative Less than & Greater than Frequency curves.

4.2.2. Palladium (Pd) Statistical Analysis

The grade distribution of the Pd g/t values shown in Figure 12 is similar to that of Pt; it is also non-symmetrical, however, with a higher degree of positive skewness. The tail on high-grade end is slightly thicker compared to Pt (Figure 10). This tail on Pd comprises roughly 7% of the data, as opposed to 3% on Pt.

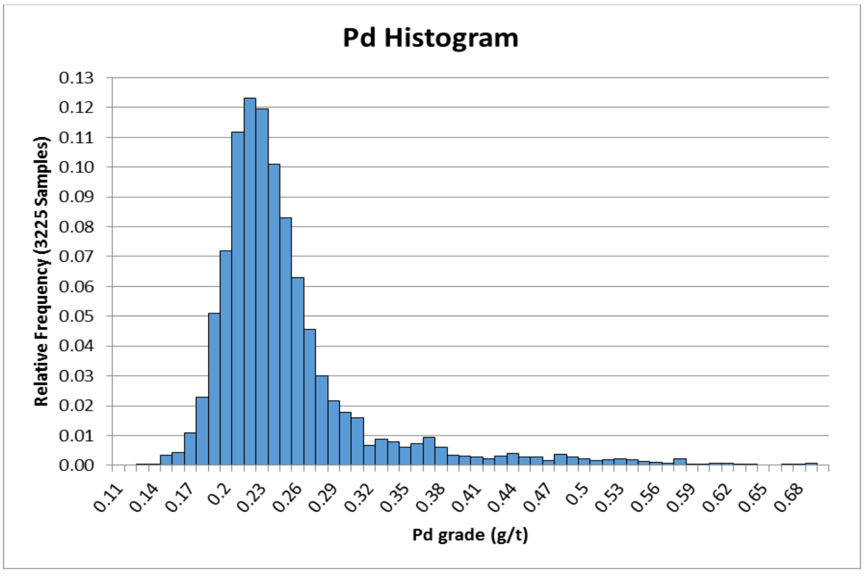

Figure 12: Palladium (Pd g/t) Relative Frequency Histogram.

The classical statistics summary for Pd is presented in Table 9 below. The Log-normal estimate of 0.252 g/t; is 0.001g/t or 0.31% lower than the average 0.253 g/t of the raw data. As expected, this difference is fractionally bigger than that observed in Pt because the Coefficient of Variation (CV) is higher. Also, the tail on the higher-grade end is slightly thicker. The skewness of 3 and Kurtosis of 12 confirm the data to have a non-symmetrical and Lepto-kurtic shape, with roughly 34% of the data in the middle (three highest columns in the histogram).

Table 9: Palladium (Pd) Classical/Summary Statistics.

No of Assay	3225
Min (g/t)	0.125
Max (g/t)	0.695
Range (g/t)	0.570
Average (g/t)	0.253
Variance $(g/t)^2$	0.005
Standard Deviation (g/t)	0.068
Coefficient of Variation (CV)	0.27
Skewness	3
Kurtosis	12
Log Average [ln(g/t)]	-1.402
Log Variance $[\ln(g/t)]^2$	0.049
Log-normal Estimate (g/t)	0.252

The difference between the Palladium average grade calculated from the samples and the lognormal estimate is 0.001 g/t or 0.31%, which is considered negligible

The Pd cumulative relative frequency distributions are shown in Figure 13, as expected these curves reflect the behaviour observed in the Pd histogram above: The middle section has a single slope, and therefore it is concluded that the data is coming from a single population.

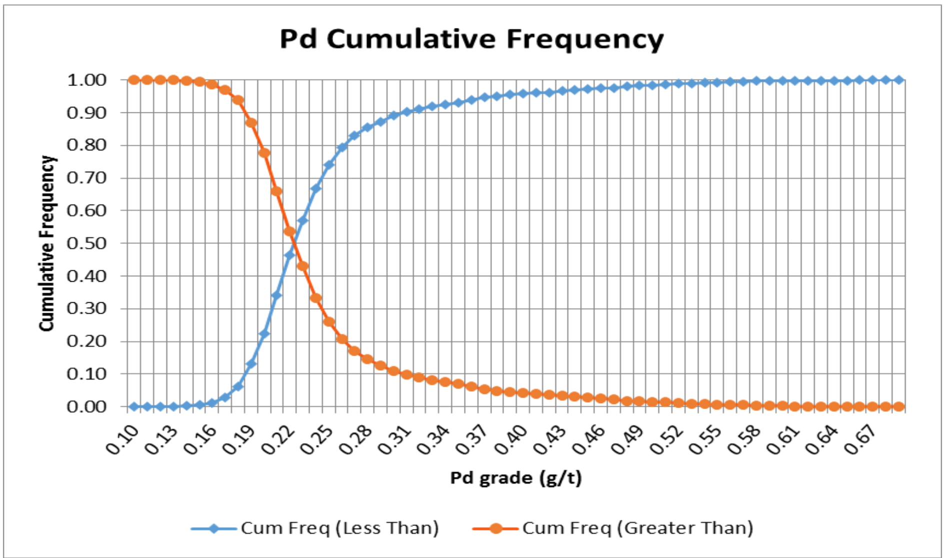

Figure 13: Palladium (Pd g/t) Cumulative Less than & Greater than Frequency curves.

4.2.3. Gold (Au) Statistical Analysis

The Au grade distribution portrays a different shape from both Pt and Pd. The histogram suggests that the data possibly comes from 2 or 3 populations; see the red, yellow and blue models drawn in on the Au histogram in Figure 14. A tail is also observed on the right, but it is less pronounced than for Pt and Pd.

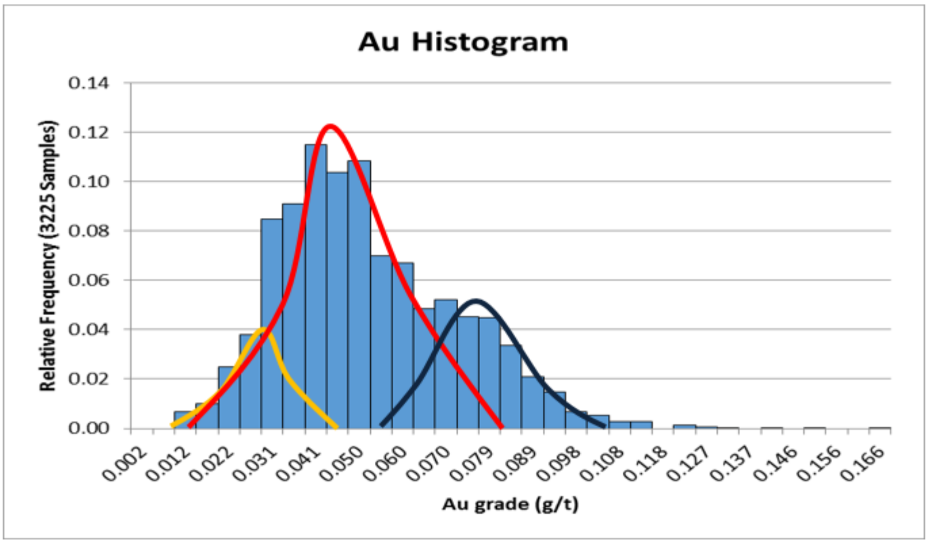

Figure 14: Gold (Au) Histogram showing possible source population divisions.

The distribution is asymmetrical, and the skewness (Table 10) is 1, because of that tail in the high-grade end. Kurtosis shows data to be meso-kurtic. The Log-normal estimate returned 0.053 g/t, which is 0.54% above the mean derived from untransformed data. Although the difference is negligible, it is the highest percentage diffidence of all elements analysed, possibly due to the combination of populations in the data, as noted in the histogram.

Table 10: Gold (Au) Classical/Summary Statistics.

No of Assay	3225
Min (g/t)	0.010
Max (g/t)	0.165
Range (g/t)	0.155
Average (g/t)	0.0521
Variance $(g/t)^2$	0.0004
Standard Deviation (g/t)	0.021
Coefficient of Variation (CV)	0.39
Skewness	1
Kurtosis	3
Log Average [ln(g/t)]	-3.035
Log Variance $[ln(g/t)]^2$	0.173
Log-normal Estimate (g/t)	0.0524

The difference between the average Au grade and the lognormal estimate is -0.0003 g/t or -0.54%, which is considered negligible

Moreover, the three black slope lines drawn on the cumulative less than Au frequency distribution in Figure 15 coincide with the three possible populations revealed by the Au histogram. The first slope is noted on the lower 10% of the data, and the second slope exists between 10% and 60%, with the third change happening between 60% and 90%.

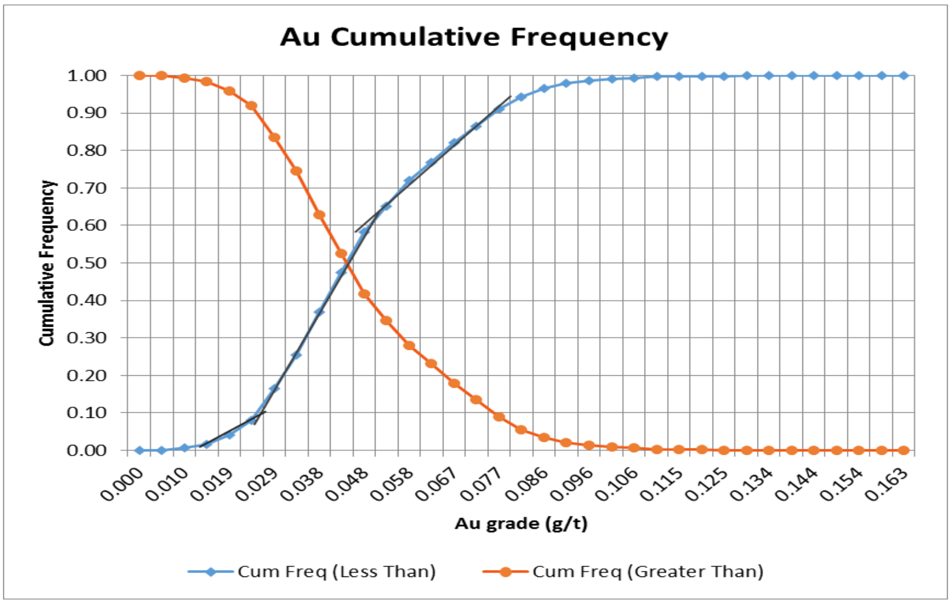

Figure 15: Gold (Au) Cumulative Less than & Greater than Frequency curves.

4.2.4. 3E (Pt+Pd+Au) Statistical Analysis

The 4E grade – a combination of Platinum, Palladium, Rhodium and Gold – is an important grade value (variable) that many Platinum companies require for different considerations (inputs). On the Case Study tailings dam, Rhodium (Rh) has been of insignificant quantity and only a few samples were analysed for Rh. Rh is only analysed on samples that have a 3E (Pt+Pd+Au) grade greater than 1.5 g/t. In this case, only 11 out of 3225 samples (0.34%) triggered the Rh analysis threshold. Therefore, Rh had been excluded from any analysis, as such 3E was statistically analysed instead.

The 3E grade distribution is therefore a combination of the three elements that comprised it. Gold (Au) has the least influence because of its marginal grade values. As observed on the individual Pt and Pd, the distribution of 3E (Figure 16) is also non-symmetrical, but the degree of positive skewness is relatively low. Furthermore, the histogram suggests that 3E distribution is characterised by a single population, possibly due to subdued influence from marginal Au grade values, which was particularly noted to have more than one population.

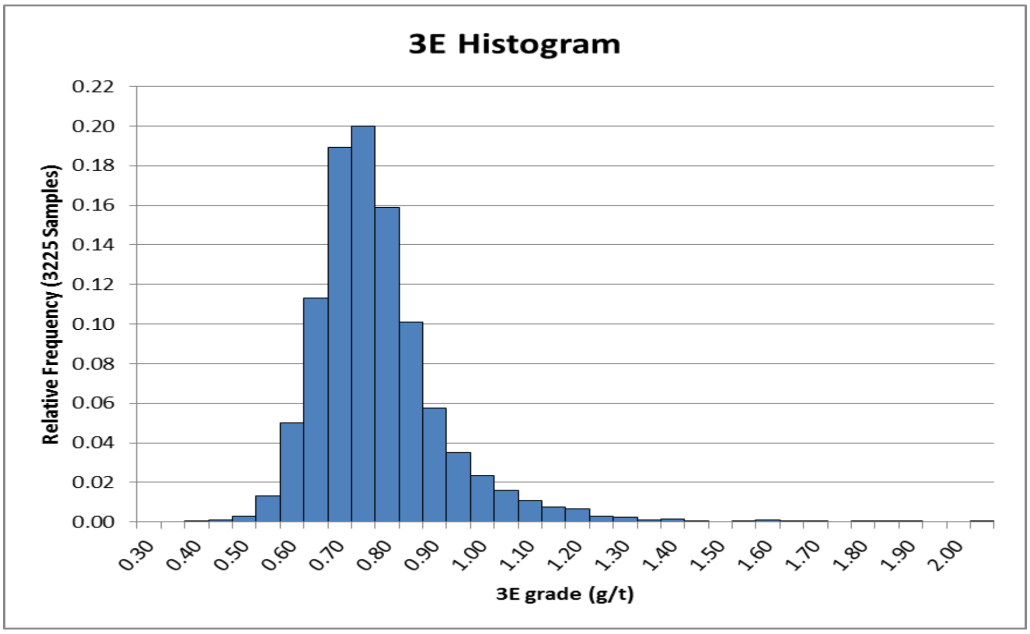

Figure 16: 3E (Pt, Pd & Au) Relative Histogram.

Table 11 below contains the 3E summary statistics. The Log-normal estimate is virtually identical to the average derived from the raw data. The difference of 0.08% is negligible. A skewness of 2 confirms that data is asymmetrical and the kurtosis value of 14 implies it is lepto-kurtic.

Table 11: 3E (Pt, Pd & Au) Classical/Summary Statistics.

No of Assay	3225
Min (g/t)	0.385
Max (g/t)	2.065
Range (g/t)	1.68
Average (g/t)	0.7833
Variance $(g/t)^2$	0.020
Standard Deviation (g/t)	0.143
Coefficient of Variation (CV)	0.18
Skewness	2
Kurtosis	14
Log Average [ln(g/t)]	-0.258
Log Variance $[\ln(g/t)]^2$	0.027
Log-normal Estimate (g/t)	0.7827

The difference between the average 3E grade and the lognormal estimate is 0.0006 g/t or 0.08%, which is considered negligible

Likewise, cumulative distribution frequency (Figure 17) for 3E resembles that of its two main underlying elements, Pt and Pd. It is dominated by a single slope, confirmation of a possible single population as deduced in the histogram.

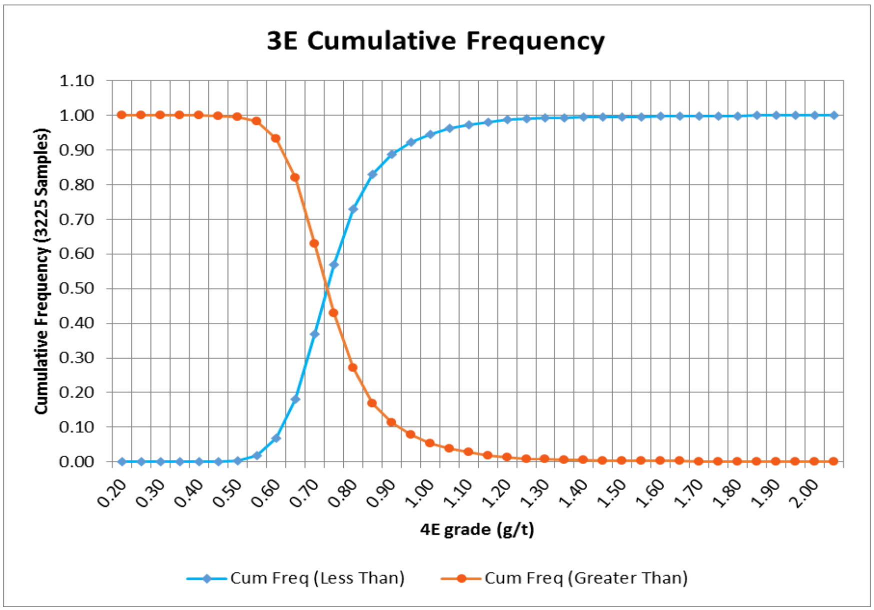

Figure 17: 3E (g/t) Cumulative Less than & Greater than Frequency curves.

4.2.5. Nickel (%Ni) Statistical Analysis

The Nickel is the first of the two base metals that have been analysed. The histogram suggests that the Ni grade (%) data possibly comes from two populations as shown in Figure 18, even though the second population looks much smaller than the first. The Distribution is non-symmetrical, possibly due to the existence of the small population and the short tail that exists on the high-grade side.

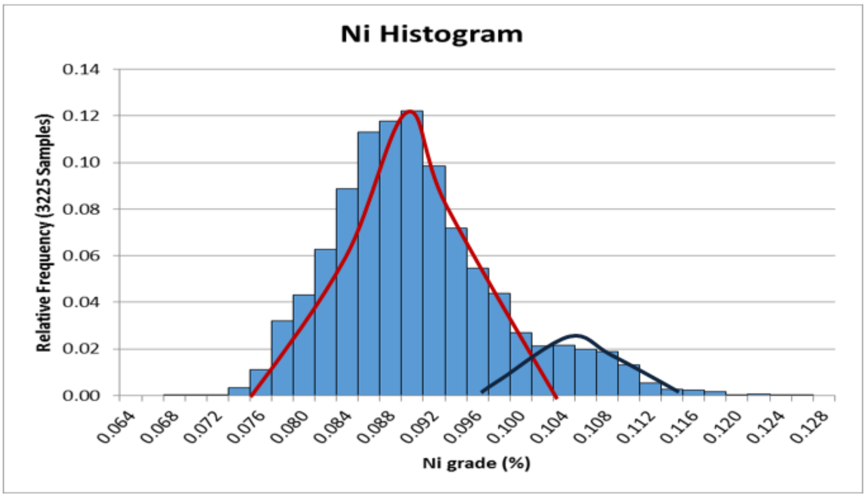

Figure 18: Nickel (% Ni) Histogram.

The Nickel grade statistics have been summarised in Table 12 below. The skewness parameter confirms the asymmetry observed in the histogram. Likewise, the kurtosis value of 4 confirms the lepto-kurtic histogram shape. Furthermore, the log-normal estimate is identical to the raw data average at 0.0905%.

Table 12: Nickel (%Ni) Classical/Summary Statistics.

No of Assay	3225	
Min (%)	0.0681	
Max (%)	0.127	
Range (%)	0.0589	
Average (%)	0.0905	
Variance $(\%)^2$	0.00006	The average Ni % of the raw data and the lognormal estimate Ni % are identical
Standard Deviation (%)	0.0079	
Coefficient of Variation (CV)	0.09	
Skewness	1	
Kurtosis	4	
Log Average [ln (%)]	-2.406	
Log Variance $[\ln (\%)]^2$	0.007	
Log-normal Estimate (%)	0.0905	

A closer look at Ni cumulative distribution frequency in Figure 19, highlights the possible existence of multiple populations, identified by the changes in the slope on the graph. Also, the curve on the high-grade end is wider, which corresponds to the thicker tail (small population) observed on histogram.

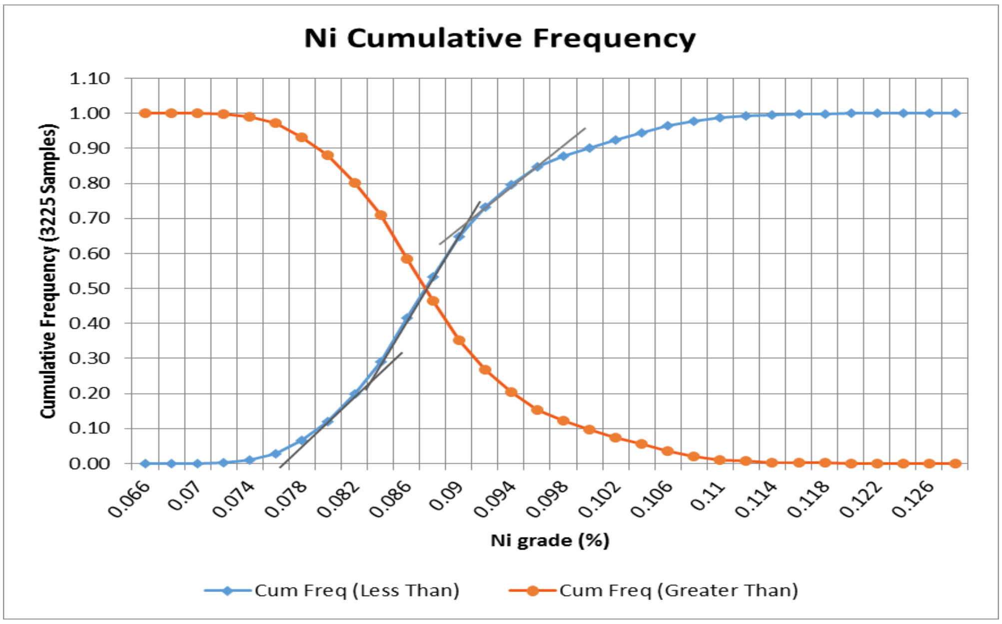

Figure 19: Nickel (% Ni) Cumulative Less than & Greater Frequency curves.

4.2.6. Copper (%Cu) Statistical Analysis

The distribution of Copper, the second of the two base metal elements, has the longest tail on high grade end of all the elements analysed. The histogram suggests, although not strongly, the possibility of at least two populations, as illustrated by distribution models drawn in Figure 20.

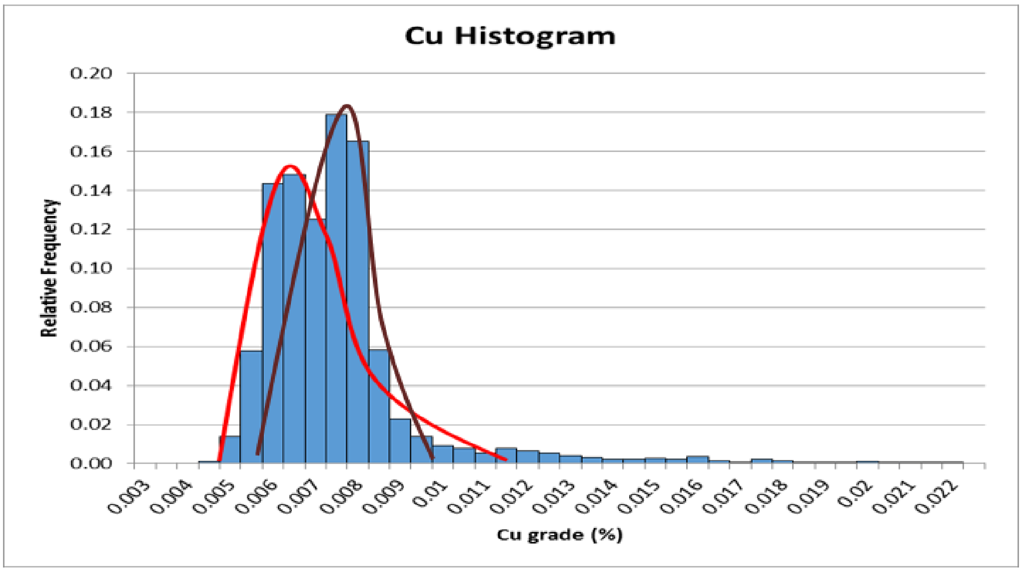

Figure 20: Copper (%Cu) Histogram.

Cu summary statistics is presented in Table 13 below. The Lognormal estimate is 0.0078% Cu, and it is virtually the same as the untransformed average grade of 0.00782% Cu. The difference is a negligible 0.26%. The positive skewness of the distribution is confirmed by the skewness parameter value of 3. The kurtosis value of 16 confirms the lepto-kurtic nature of the distribution.

Table 13: Copper (%Cu) Classical/Summary Statistics.

No of Assay	3225
Min (%)	0.005
Max (%)	0.0225
Range (%)	0.0175
Average (%)	0.00782
Variance $(\%)^2$	0.000004
Standard Deviation (%)	0.0019
Coefficient of Variation (CV)	0.25
Skewness	3
Kurtosis	16
Log Average [ln (%)]	-4.875
Log Variance $[\ln (\%)]^2$	0.042
Log-normal Estimate (%)	0.00780

The difference between the average Cu grade and the lognormal estimate is 0.00002 or 0.26%, which is negligible

The Cumulative Cu Grade Frequency Distribution (Figure 21) confirms multiple slopes in the middle section, an occurrence being attributed to an existence of a possible second population from which the data is coming from.

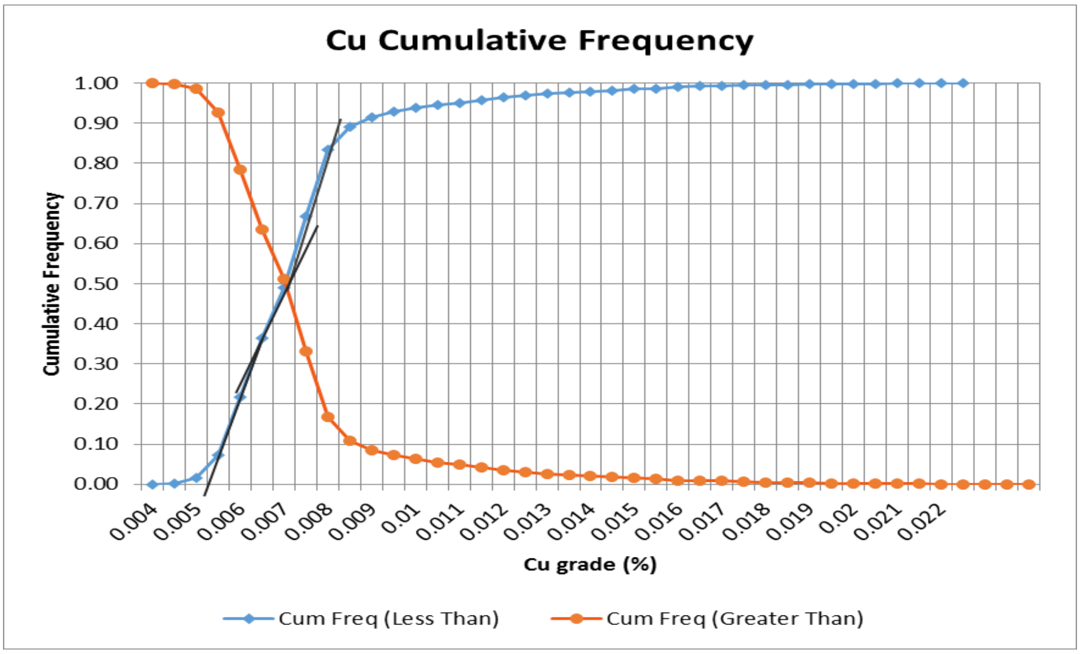

Figure 21: Copper (%Cu) Cumulative Less than & Greater Frequency curves.

4.2.7. Density Statistical Analysis

The seventh and last variable analysed is density. It is a very important variable as it used to calculate tonnages of the resources being evaluated, which in turn is used to calculate (estimate) the actual metal content – through average grade – and financial valuation of the deposit.

Data from *in situ* tailings density comes from two populations, as indicated by the histogram shown in Figure 22. The second population is slightly smaller and exists on the higher end of the density grade (values). The underlying distribution of each individual population is interpreted to be symmetrical. Visually, these distributions are the closest to a normal distribution of any variable (elements) analysed.

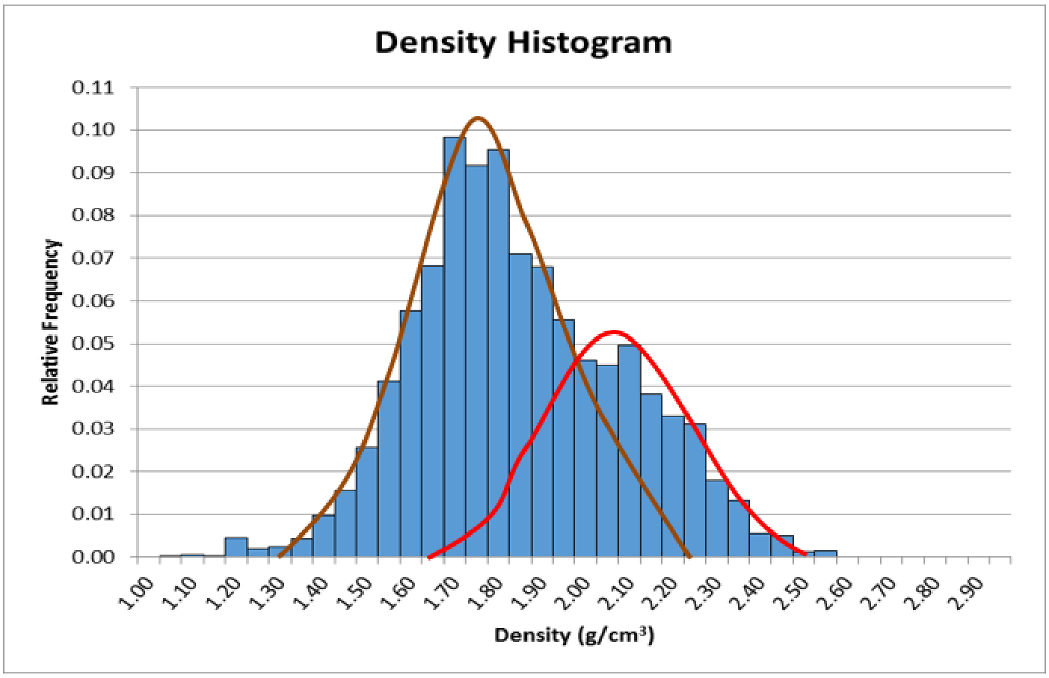

Figure 22: Density Histogram.

The Density summary statistics in Table 14 below clearly shows a skewness parameter of 0 indicating symmetry and a kurtosis of 3, indicating a meso-kurtic shape, both are characteristics of a normal distribution. Also, the log-normal estimate is a lowly 0.0001 g/cm^3 (0.00%) above mean calculated from untransformed data.

Table 14: Density Classical/Summary Statistics.

No of Assay	3225
Min (g/cm^3)	1.065
Max (g/cm^3)	2.575
Range (g/cm^3)	1.510
Average (g/cm^3)	1.8728
Variance (g/cm^3)^2	0.057
Std Dev (g/cm^3)	0.238
Coefficient of Variation (CV)	0.13
Skewness	0
Kurtosis	3
Log Average [ln(g/cm^3)]	0.619
Log Variance [ln(g/cm^3)]^2	0.016
Log-normal Estimate (g/cm^3)	1.8729

The difference between the average density value and the lognormal estimate is 0.0001 or 0.00%, which is virtually identical

The symmetry around the crossover of the two cumulative frequency plots shown in Figure 23, confirm the likelihood of a normal distribution for the density data. The slight change in the slope on the right confirms the existence of a second and smaller population on the high-grade side.

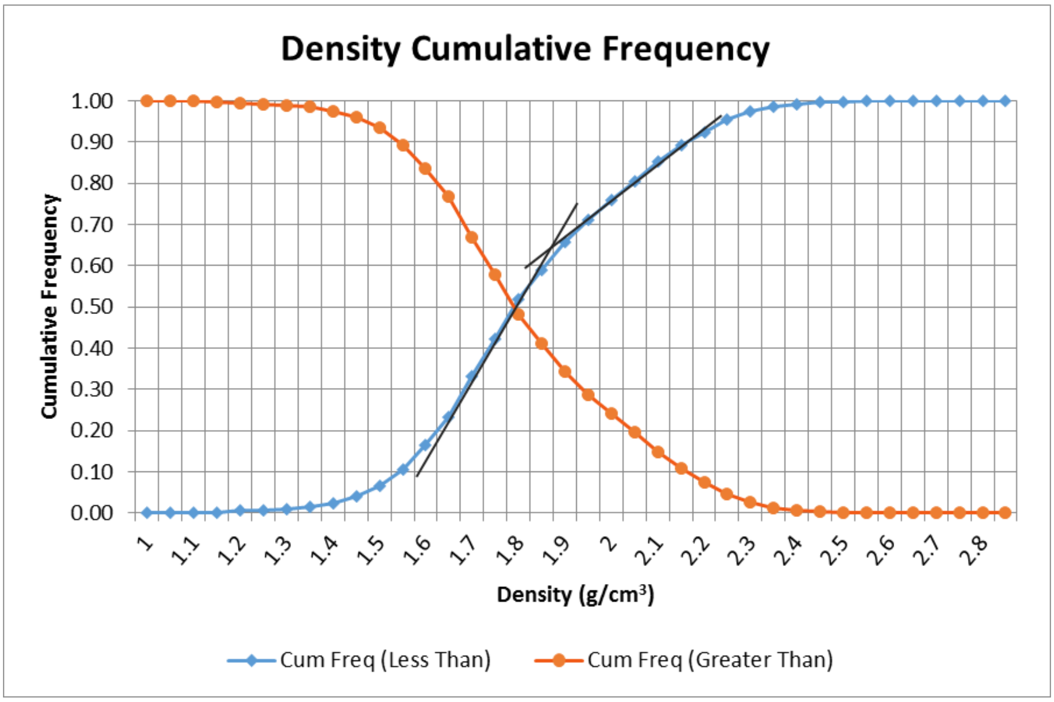

Figure 23: Density Cumulative Less than & Greater Frequency curves.

4.3. Summary and General Discussion on Statistical Analysis

The distributions of the grades in the Case Study tailings dam, presented by histograms and cumulative frequency graphs, appear to be mainly asymmetrical and positively skewed to varying degrees. Although the data is skewed, it is just a slight positive skewness for almost all the variables. Furthermore, distributions of some of these elements (Au, Cu, Ni and Density) come from more than one population, and as such histograms and averages of such elements alone cannot be used to conclusively deduce the underlying distribution without turning to other statistical parameters.

Some of the statistical parameters were selected and compiled into one table (Table 15) to be able to draw some additional conclusions. In terms of skewness, density is the only variable that returns 0 for this parameter; therefore, it is the only distribution that is outright normal. Two other variables (Au and Ni) have skewness of 1. The rest of the variables vary between 2 and 4 which correspond to the slightly longer tail on the high-grade end.

Table 15: Comparison of some of the parameters from variables' summary statistics.

Variable	Skewness	Std Dev	Mean	Log-normal Estimate	Mean vs Log-normal Estimate		Coefficient of Variation
					Diff	% Diff	
3E	2	0.143	0.783	0.783	0.0006	0.08	0.18
Pt	4	0.091	0.478	0.478	0.0006	0.12	0.19
Pd	3	0.068	0.253	0.252	0.0008	0.31	0.27
Au	1	0.021	0.052	0.052	-0.0003	-0.54	0.39
Ni	1	0.0079	0.091	0.091	0.000005	0.01	0.09
Cu	3	0.0019	0.0078	0.0078	0.00002	0.26	0.25
Density	0	0.238	1.873	1.873	-0.0001	0.00	0.13

The Coefficient of Variation (CV) was also looked at; this parameter provides an indication of the variability of the variable (element) being studied; the higher the CV the greater the variability and the smaller the less the variation in the data. Variability is concluded to be minimal in the Case Study tailings dam, as confirmed by the small CVs. For Au the CV = 0.39, the Pd CV is 0.27 and for Cu the CV = 0.25, whilst the CV is even smaller for the rest of the variables.

A conclusion can therefore be drawn based on the interpretation of a combination of parameters, particularly related to skewness and the differences (%) between the untransformed mean and log-normal estimate. Log-normal estimates calculated returned values virtually identical to the arithmetic mean in all variables.

The biggest difference is -0.54% for Au, followed by 0.31% and 0.26% in Pd and Cu respectively. The rest of the variables have almost zero differences. Log transformation is therefore not viewed as a prerequisite before kriging, since no substantial improvement is expected to be realised on the accuracy of the estimated results. As such the Case Study tailings dam was kriged from raw and untransformed data.

Out of interest and for comparison purposes, three variables were selected and kriged on both raw and log transformed data. The three variables are: Density, 3E and Cu (see Chapter 7).

CHAPTER FIVE: GEOLOGICAL MODELLING

Geological modelling is a computer aided exercise, and it is a critical step that is performed to try and delineate the ore body into areas of similar or identical characteristics, to improve the accuracy of the estimation.

The geological model that is produced needs to be geometrically consistent and adequately capture key geometric factors that influence the distribution of potential economic grades. Moreover, attention must not only be given to geology, grade distribution and profile must also be considered for building a comprehensive geological model for accurate grade estimation (Vann, (2008)).

Geological modelling of the Case Study tailings resources was also performed to try and establish geological zones or facies that might have an impact or influence on grade distribution and continuity. The following three variables, that might reveal geological zones, were studied and assessed: lithology (tailings material), PGE grade (3E) and particle size distribution (PSD). The fourth variable, water content - does not necessarily have any relationship with grade, since Platinum tailings are not comprised of soluble minerals that can dissolve and get mobilised by water. Nevertheless, water content was also considered for reference purposes.

5.1. Lithology (Tailings Material)

Crushing and grinding of ore during processing deforms or mixes the petrological and mineralogical signature of each individual rock type. For this reason, the tailings lithology was largely classified based on clayey content and grain size. These two physical parameters in turn, are considerably influenced by the tailings deposition process. Sandy and coarser materials are more likely to settle adjacent to feeding pipes laid around the edge of the dam, while fine and clayey particles are transported further away from the pipes towards the centre of the dam.

A 3D view of the lithologies observed in the boreholes is shown in Figure 24 below. The dam is mainly comprised of two main tailings lithologies, namely: Silty Sand (SM) and Clayey Sand (SC). SM is generally medium to fine grained. SC material, in the other hand, is very fine to ultra-fine grained.

A lithological trend, although weak and discontinuous, exists in the dam. Generally, the finer and clayey SC material occurs commonly, and in much thicker proportions, in boreholes drilled in and around the centre of the dam (Figure 24).

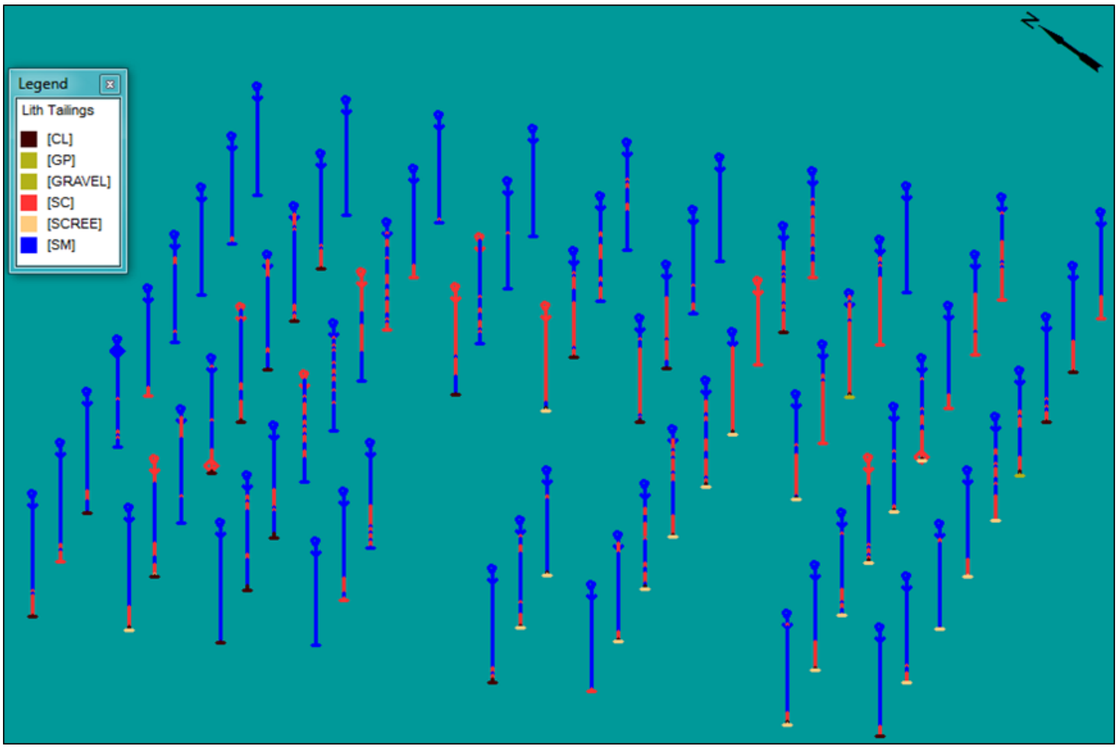

Figure 24: A 3D view of lithology on all boreholes across the dam.

Silty (sandy) SM material, on the other hand, is commonly found to comprise boreholes drilled around the perimeter of the dam. These boreholes – drilled around the perimeter – usually only intersect finer SC material, towards the bottom of the dam, although insignificant SC patches are also found occasionally and erratically spreading throughout those boreholes. Similarly, the topmost material on SC dominated boreholes drilled around the centre of the dam, is often comprised of SM material.

Despite the localised patterns noted above, the continuity and consistency of lithological distribution is not strong or large enough to facilitate any subdivision of the dam into geological zones based on the lithology. In addition, significant interlayering of SC and SM was noted throughout the dam. The overall impression is that lithology does not have definite and large scale pattern that can result in the dam being subdivided into practical geological zones.

Based on these lithological distributions, the geological model of the Case Study tailings dam was generated comprising only a single zone that covers the entire dam.

5.2. PGEs (3E) Grade

The 3E grade (Figure 25) was also assessed as another variable that can possibly be used to establish geological zones. A relation was found to exist between lithology and 3E grade distribution: Generally, high grade was often found to be associated with finer SC material, whereas coarser SM material was often not as rich as SC material. Furthermore, high 3E grade – and much thicker high grade profile – was often found in boreholes drilled in and around the middle of the dam, in line with the trend noticed on SC lithology. Boreholes drilled around the perimeter tend to have good grade values towards the bottom of the dam, where SC material is also found.

What is unique about the grade distribution is that: interlayering of low and high grade is found to be more prominent in 3E grade values than what was observed in lithology. As was the case in lithology, 3E grades do not carry and display consistent and large scale trends that could be used to facilitate any practical subdivision of the dam into geological zones. Therefore, a concept of a single zone that was created, as mentioned under lithology, is still applicable since 3E grade assessment did not affect (introduce) any changes on the zone.

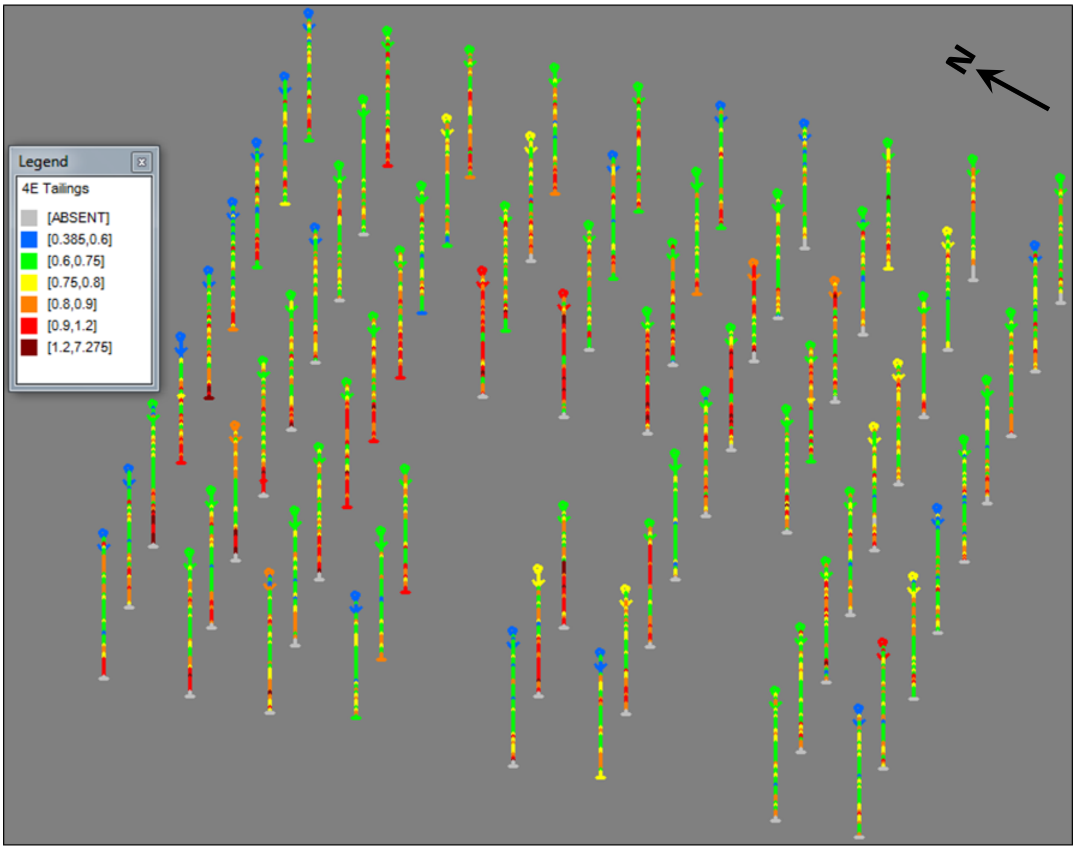

Figure 25: A 3D view of spread of 3E grade on all boreholes across the dam.

5.3. Particle Size Distribution (PSD)

Particle size distribution (PSD) or grain size was also measured by the lab as part of sample analyses. Seven sieves were used to measure PSD; namely 250 Micron (µm), 212µm, 180µm, 150µm, 125µm, 75µm and 45µm. Six boreholes on the mid-western section (coloured red in Figure 26) did not have comprehensive PSD results, and as such were not considered. To perform PSD analysed on only two sizes; 250µm, 212µm, 180µm and 150µm sieves were combined to represent greater than (>) 150µm (Figure 26). Likewise, 125µm, 75µm and 45µm were also combined to represent less than (<) 150µm (Figure 27).

The PSD trend observed is that boreholes dominated by particle size greater than 150µm are located on the south-eastern corner of the dam (Figure 26). Also, the last row of boreholes on the northern-end comprises at least 50% particles greater that 150µm. the rest of the borehole have fewer and erratic spreading of particles bigger than 150µm throughout.

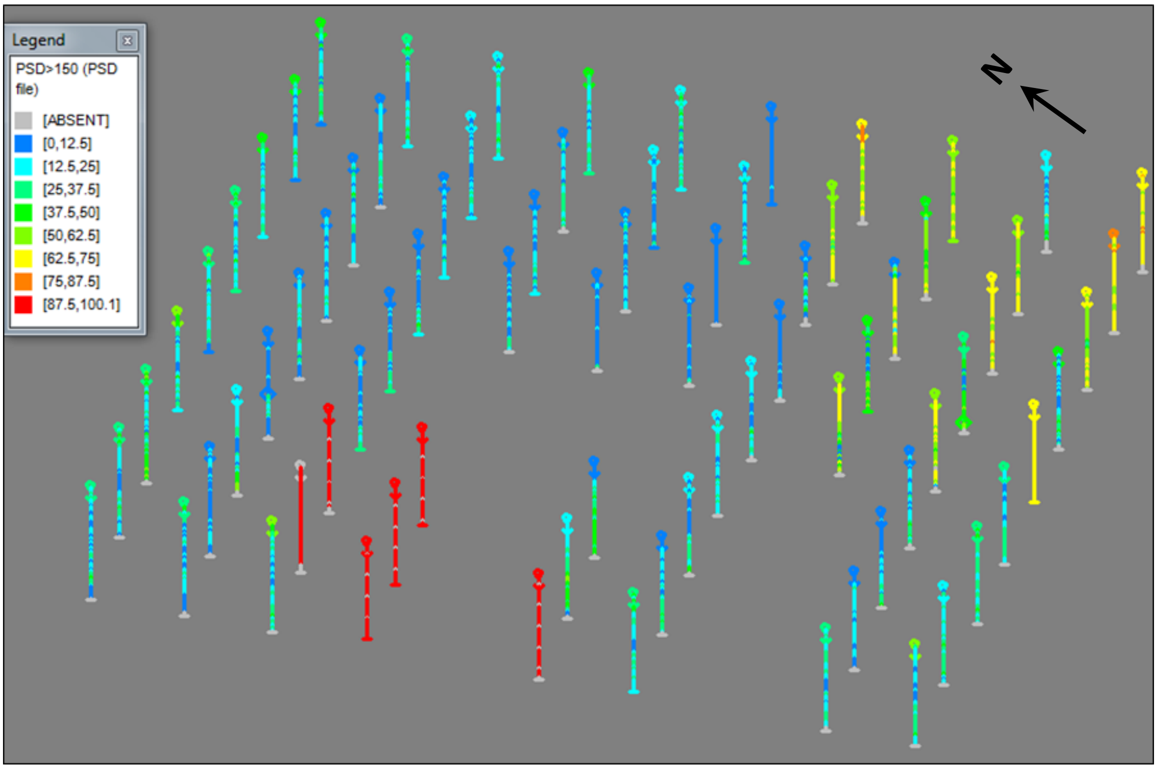

Figure 26: A 3D view of grain size > 150µm (%) on all boreholes across the dam.

Particle sizes less than 150µm, in turn, is the size that is more prominent or dominant in the dam (Figure 27). Boreholes drilled in and around the centre of the dam are overwhelmingly dominated by particle sizes < 150µm. There is also a significant interlayering between the two PSD sizes (groups) on all the boreholes. Furthermore, PSD analysis revealed that correlation exists between PSD, lithology and grade. Small sized sieves correspond to SC material, which in turn is often associated with high grade.

Similar to the 3E and lithology, the PSD trend was also found to be inconsistent and discontinuous on a larger scale, or along similar elevation across boreholes, to warrant the establishment of geological zones based on PSD sizes.

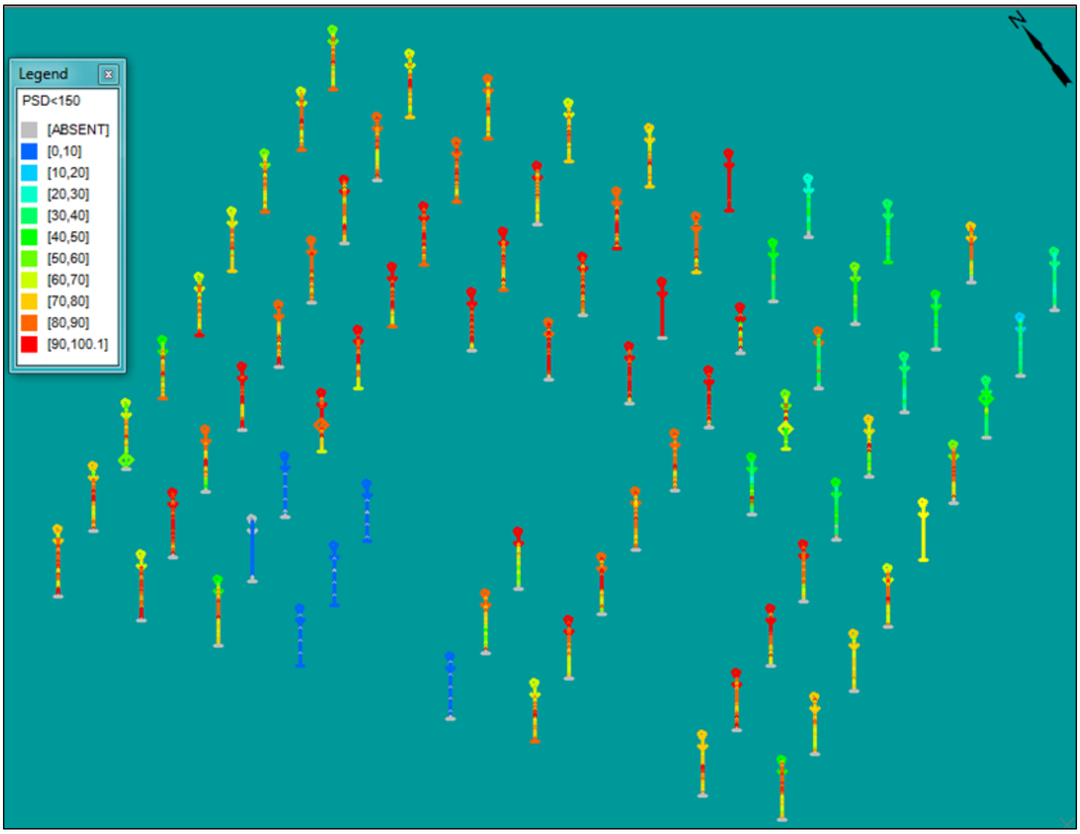

Figure 27: A 3D view of grain size < 150µm (%) on all boreholes across the dam.

5.4. Water Content

Moisture content is the only variable that was found to be consistent throughout the dam (Figure 28). All boreholes are comprised of three levels (zones) of moisture content: dry, partly wet (Moist) and wet. The moisture content trend is consistent throughout all the boreholes, dry is the topmost zone, followed by partly wet in the middle, and wet as the bottom-most zone of each borehole. The only exception was encountered at the top when drilling was conducted soon after rain. In which case, patches of moist and wet material were encountered at the top.

Platinum tailings are not comprised of soluble minerals that can dissolve and get mobilised by water, thus water content does not have any impact on grade continuity or lithology. Consequently, geological zones were not modelled based on water content. Water content was only considered as a benchmark or indication of what the consistency and continuity, that could be modelled, looks like. It is also a good indication on drilling recovery, that is where or when precaution would need to be exercised to improve recovery, should more drilling ever decided in the future.

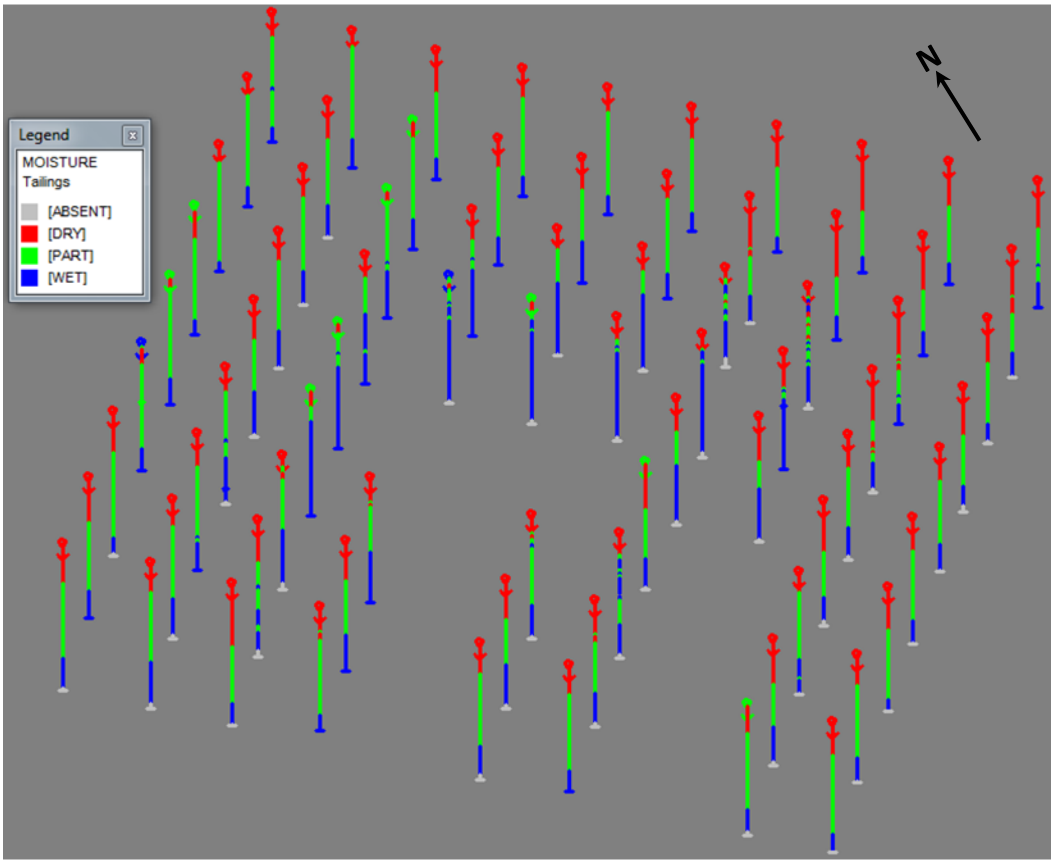

Figure 28: A 3D view of moisture content on all boreholes across the dam.

5.5. Summary on geological modelling

The three main variables analysed above, that are relevant to grade distribution and continuity, were found to be inconsistent and largely discontinuous for any practical geological modelling to be performed on a tailings dam scale. A reasonable correlation exists between lithology, grain size and grade, but this correlation was not useful in facilitating geological modelling. The correlation across these variables is that SC material is the finest (PSD < 150µm) and it is typically where highest grade is found. It is also the material that is found to have been transported the furthest away from discharging pipes (perimeter). The opposite is the case with SM material, which is the coarser, relatively poor in grade and deposited usually along the perimeter.

Figure 29a, 29b and 29c are examples of sections used to assess correlation between lithology and grades (3E and the base metals). In these figures the string in the middle depicts the actual borehole trace and the colour is based on the lithology legend. The rainbow bar on the right represents 3E g/t, the brown bar on the left is Copper %, and light blue string also on the left is Nickel %.

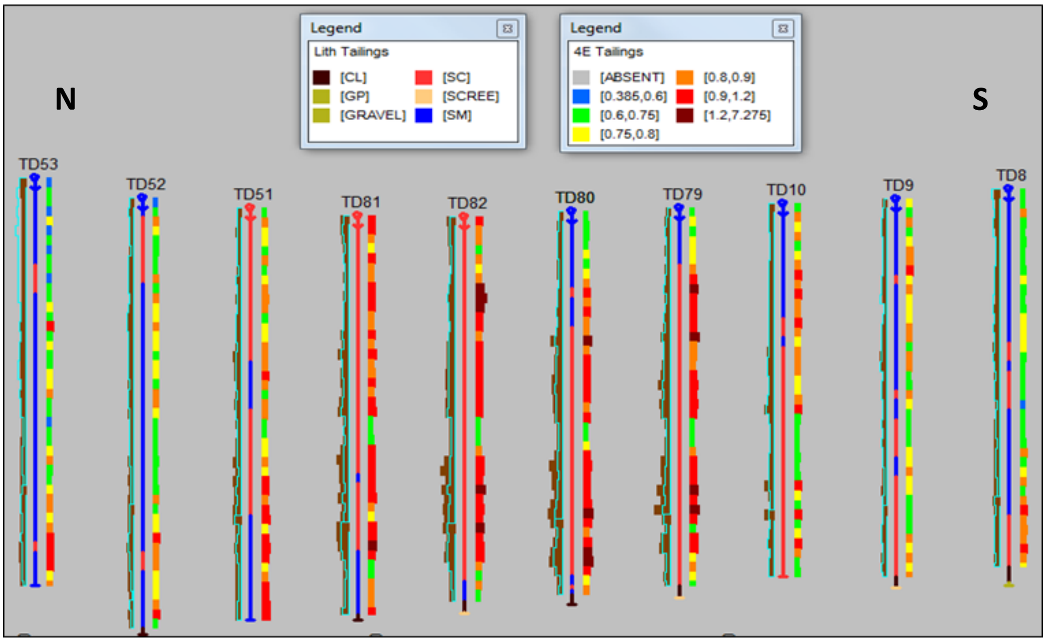

Figure 29(a): A cross-section view correlating lithology and grades (3E, Copper and Nickel).

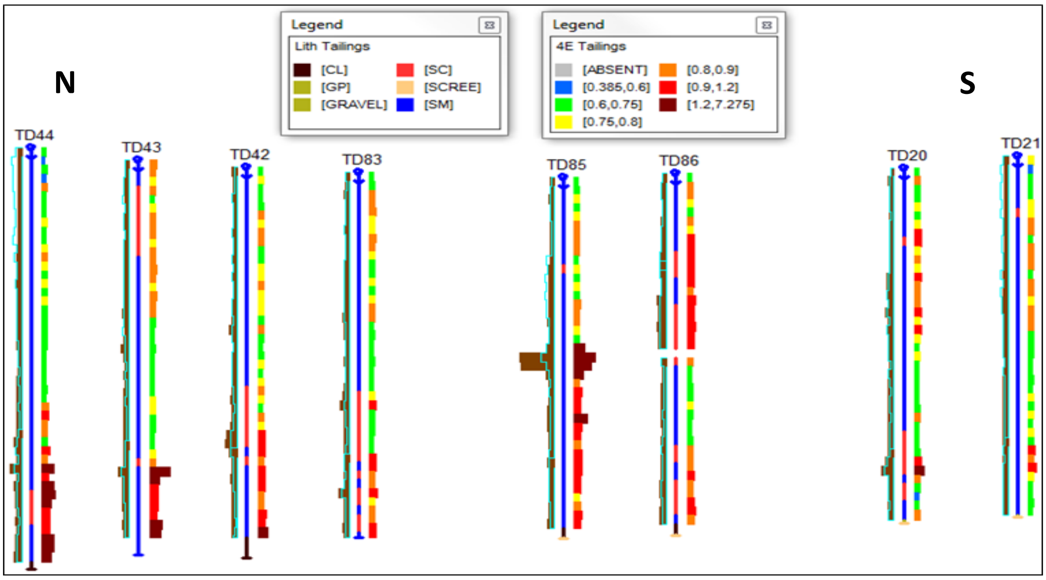

Figure 29(b): A cross-section view correlating lithology and grades (3E, Copper and Nickel).

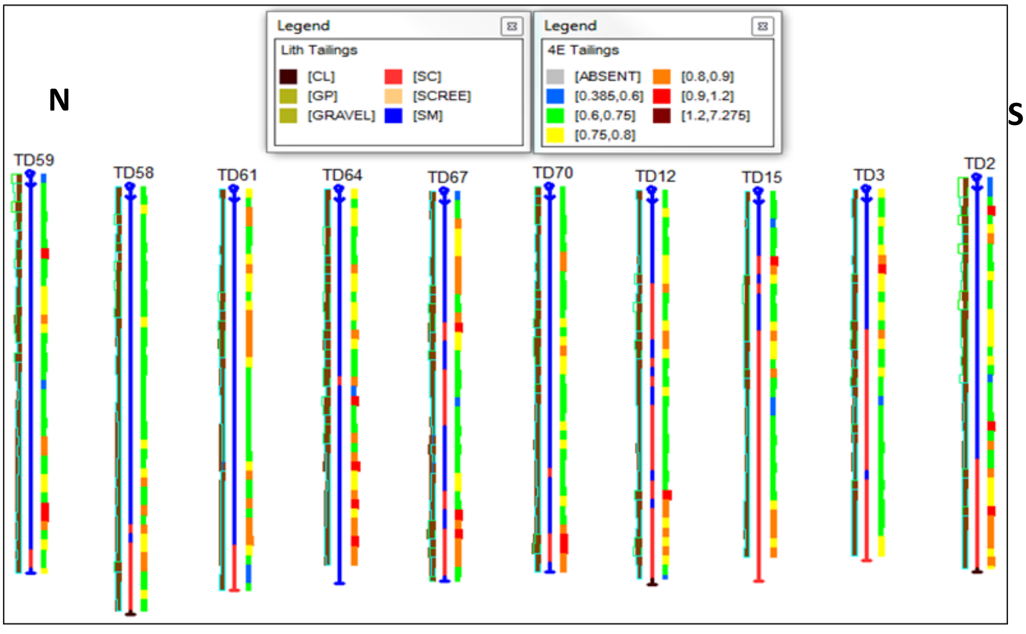

Figure 29(c): A cross-section view correlating lithology and grades (3E, Copper and Nickel).

In summary, the centre of the dam is mostly comprised of SC material and is very fine grained. It generally has good or thicker high grade (profile) accompanied by thick wet zone. Whereas, the perimeter is mostly comprised of SM material, fine to medium grained and grade not as high as in the middle part of the dam.

Furthermore, the topmost portion of several boreholes is generally comprised of SM material, and grade is often in the lower ranges. Similarly, the bottom portion of most boreholes is comprised of the much finer SC material and grade is in the upper ranges.

The inconsistent and small scale continuity of the three variables relevant to grade continuity did not result into meaningful subdivision of the dam into geological zones, which led to only a single zone being modelled to represent the entire dam (Figure 30).

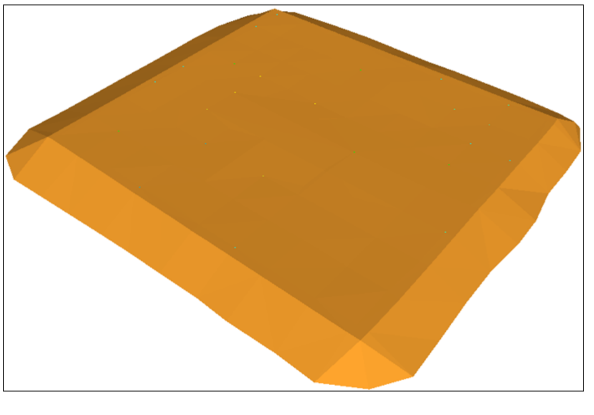

Figure 30: A 3D view of wireframe model for the whole tailings dam (proportional scale).

The area at the top of the dam (910m x 970m) is more than 20 times the average thickness of the dam (42 m). For this reason, the elevation (z coordinate) was exaggerated 10 times to allow easier analysis and presentation (Figure 31).

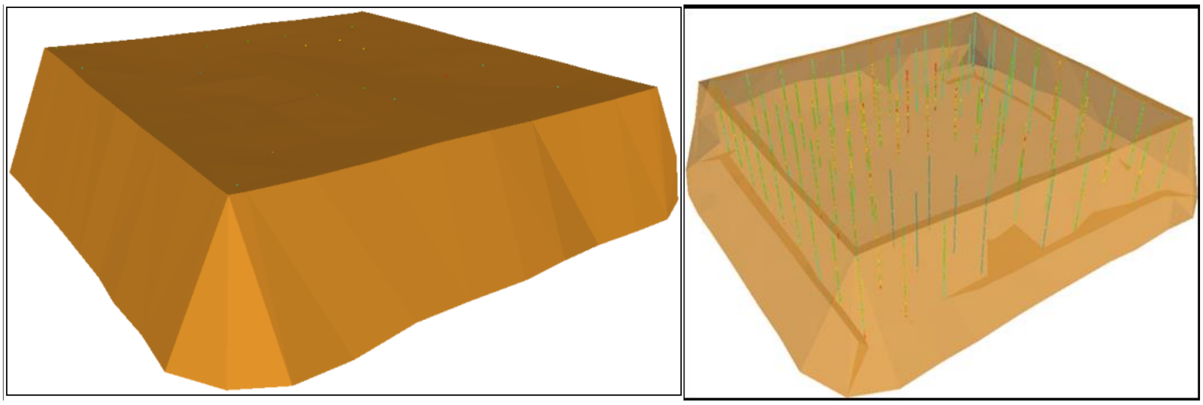

Figure 31: A 3D views of wireframe model with exaggerated z coordinate.

The wireframe generated from the single geological zone representing the Case Study tailings dam was used to create a 5m x 5m x 5m block model (Figure 32) into which grades were interpolated for the final resource model.

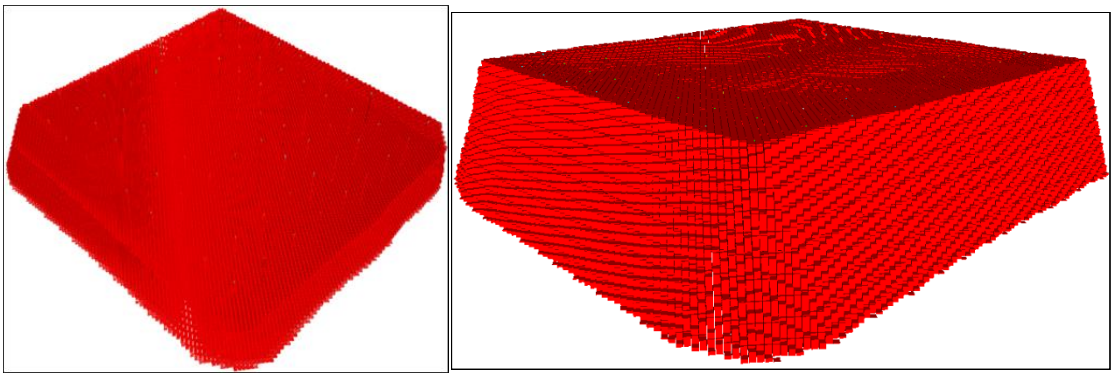

Figure 32: A 3D views of 5m x 5m x 5m block model with exaggerated z coordinate.

CHAPTER SIX: ANALYSIS OF GRADE CONTINUITY (VARIOGRAPHY)

Variography is a critical step in geostatistics, and it is a prerequisite for Kriging. It is a technique that is used to analyse spatial correlation of the grades in the ore body. Sample grade variability is analysed in relation to the distance and orientation between them.

A key assumption being that the grade values of a variable being analysed for spatial correlation come from the same population and that the differences among the grade values (samples) depend only on their relative separation (Coombes, (2008)).

Experimental semi-variograms are calculated from sample data as part of variography, and variogram models are subsequently fitted (modelled) from that fragmentary information gathered from drilling and sampling, to draw a meaningful conclusion concerning the unsampled portion of the ore body.
There are four common variogram models available for modelling in geostatistics:

1. Spherical
2. Power
3. Exponential
4. Gaussian

Additional models are also available in the two software packages that were used to conduct spatial modelling and estimation:

1. De Wijsian (Datamine)
2. Hole effect (Visor)
3. Dampened hole effect (Visor)

6.1. Case Study tailings dam variography

The Case Study tailings dam variography – including variogram models – was performed using Snowden Supervisor (Visor). A lag of 100 m, which correspond to drilling grid of 100m x 100m, was used to calculate experimental semi-variograms. Normalised variograms were generated and corresponding models were fitted on all the variables of economic interest. All seven variables were found to have a preferred orientation and therefore anisotropic variogram models were fitted. The strike of the preferred orientation is noted to be closely related among the elements; in this case the preferred orientation is typically aligned to a NW-SE or NE-SW directions, which is generally in line with the drilling pattern or grid (Figure 33).

Furthermore, mineralisation continuity of all variables revealed an underlying nugget effect, which has been calculated on a downhole direction, where lag distances as small as 1m can be considered.

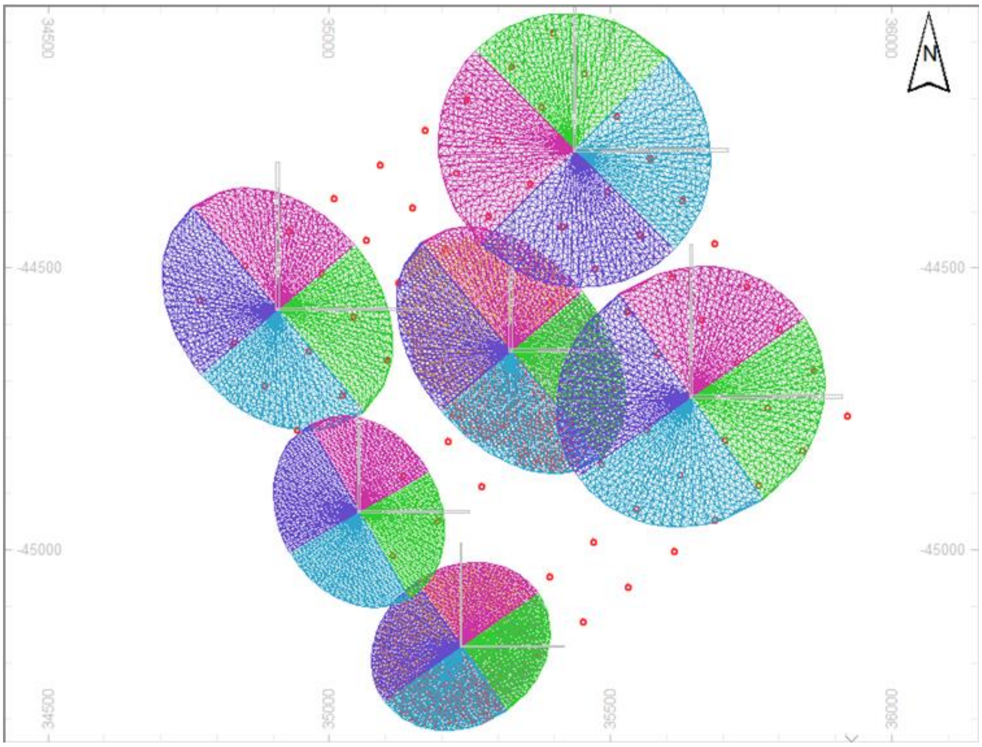

Figure 33: Search ellipse to show preferred orientation of some of the variogram models.

The Nugget effect is a discontinuous behaviour near the origin and it reflects a highly irregular behaviour (movement) of the regionalised variable at short distances, in which case variogram [$\gamma(h)$] does not tend to zero as h tends to zero (Vann, (2008)).

Variogram models for the main direction are the only ones presented in this section. Downhole and secondary directions are presented in Appendix A for information.

6.1.1. 3E Variogram

The 3E grade (the combination of 3 metals – Pt, Pd & Au) has a reasonable level of spatial correlation (Figure 34). Its normalised semi-variogram shows considerable grade continuity that goes comfortably beyond 300 m, which is equivalent to three rows of drilling (3 x 100m). This continuity reaches the sill at 500 m along the preferred main direction. The nugget effect (variance) is considered high at 0.45.

The spatial variance (normalised) calculated on the first lag [$\gamma(100)$] plots just above 0.7. Nugget (0.45) +0.25 = 0.7. The implication is that the actual continuity modelled from the 100m lag onwards only accounts for a third of the total variance (sill). It further implies that considerable spatial variability exists between 0 and 100 m, and it accounts 70% of the total sill. In the 100 m to 200 m range, the spatial variability changes from 0.7 to 0.9 (0.2), or 20% of the total sill, with the remaining 10% of the variability in the 200 m to 500 m range.

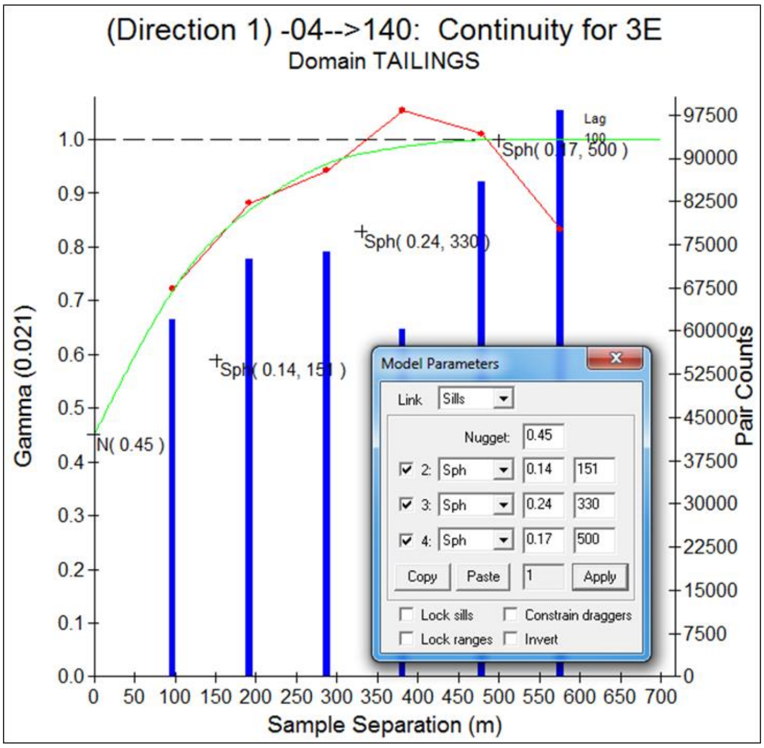

Figure 34: Anisotropic variogram model for 3E.

The 3E variogram model is a nested structure comprised of three spherical models. Nested structures or intermeshed structures indicate the presence of spatial correlation at different scales or at different ranges. These different scales of variation are superimposed (Vann, (2008)).

6.1.2. Platinum (Pt) Variogram

Platinum is the biggest contributor to 3E grade and for this reason the semi-variogram of the two elements behave in a similar manner (Figure 35). The Pt semi-variogram, likewise, has good spatial correlation that extends up to the third row of drilling (300 m). The variogram model, similarly, is a nested structure comprised of three spherical models.

The nugget variance for Pt is 0.52 and it is the highest of all variables. Normalised spatial variance calculated on the first lag [γ(100)] is also the highest at approximately 0.77.

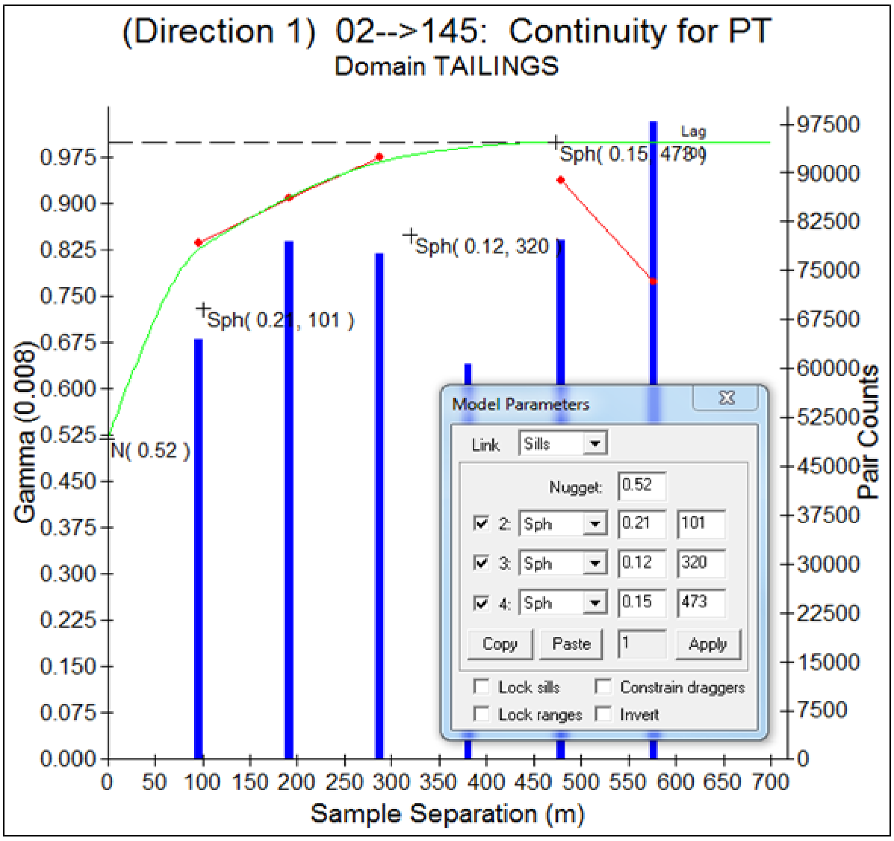

Figure 35: Anisotropic variogram model for Pt.

6.1.3. Palladium (Pd) Variogram

The Palladium variogram model is also a nested structure. Likewise, is comprised of three spherical models (Figure 36).

Pd grade continuity has one of the shortest ranges, and the experimental semi-variogram reaches the sill by the third row of drilling (300 m). The variogram has been modelled to a range of 390 m. The Nugget variance is considered high at 0.42, and the spatial variance calculated on the first lag distance [γ(100)] is about 0.7.

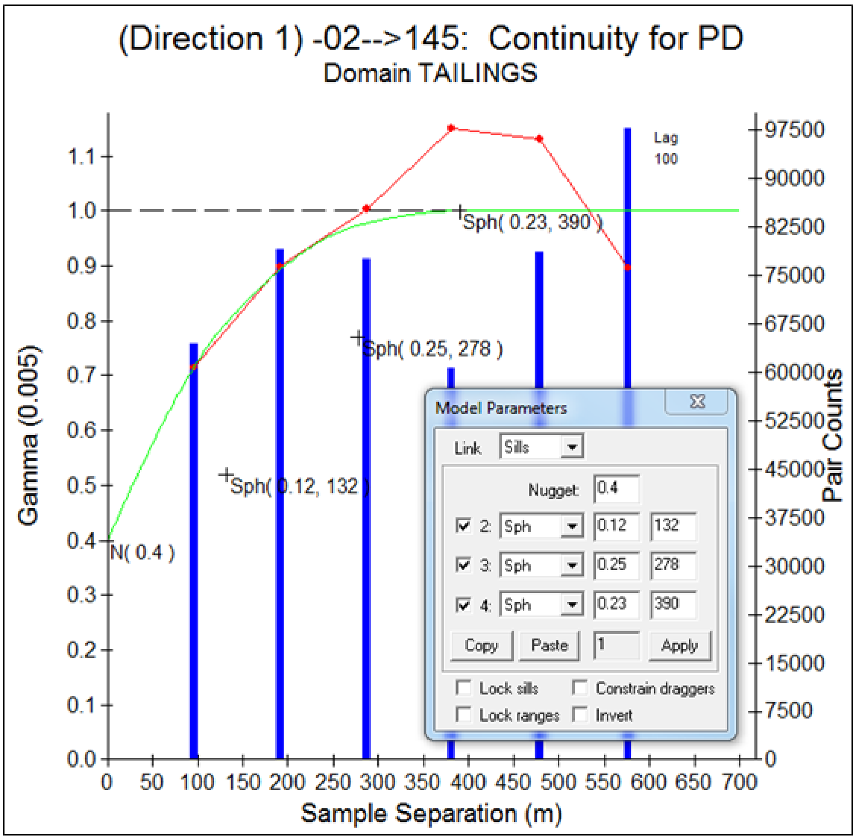

Figure 36: Anisotropic variogram model for Pd.

6.1.4. Gold (Au) Variogram

Gold has one of the best spatial correlation of all elements analysed (Figure 37). The Nugget variance is the lowest at 0.1. Also, the normalised spatial variance from the first lag [γ(100)] is the lowest at 0.38, which is even lower than nugget variance of most other elements. Therefore, the spatial variance modelled by the Au variogram from 100m and beyond (100-500 m) is 0.65, indicating better spatial correlation than in other elements.

The variogram model is a nested structure comprised of two spherical models. The total range of mineralisation continuity is a reasonable 535 m.

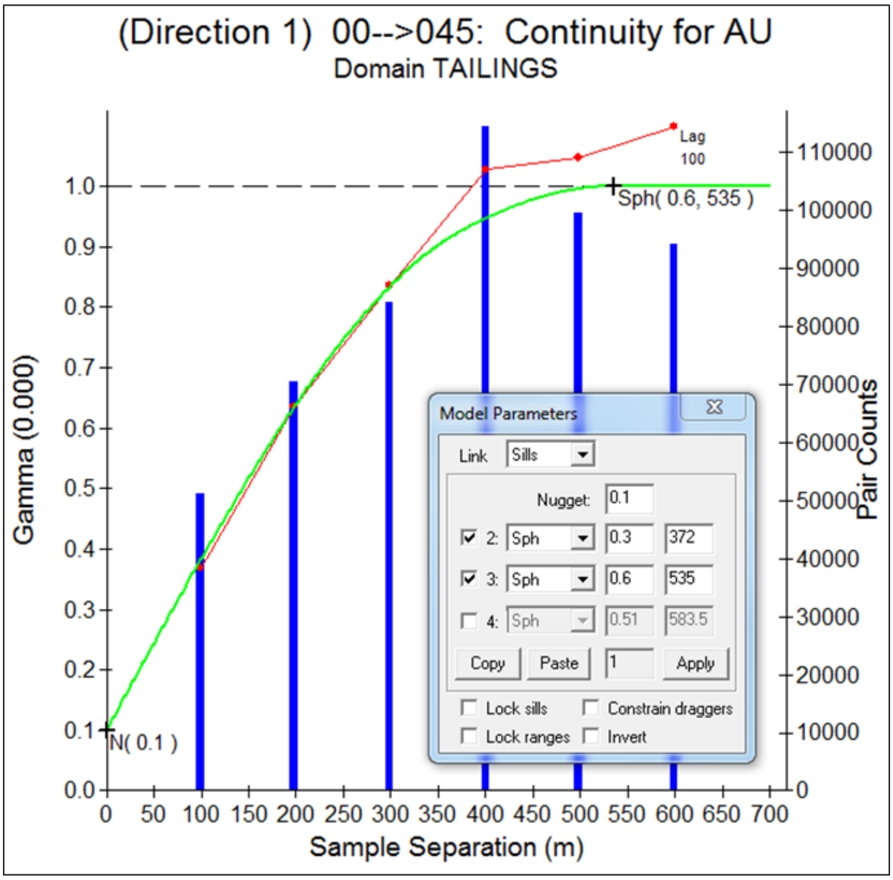

Figure 37: Anisotropic variogram model for Au.

6.1.5. Base Metals (Cu and Ni) Variogram

Copper (Cu)

Copper, like the PGEs, has reasonable mineralisation continuity (Figure 38). The experimental semi-variogram reaches the sill just after 300 m (equivalent to three drilling rows away); the rest of the range is reasonably extended to 428 m by the variogram model fitted. The fitted model is also a nested structure comprised of only two spherical models. Nugget variance is a reasonable 0.44 and the normalised spatial variance from the first pairs of the selected lag [γ(100)] is 0.65.

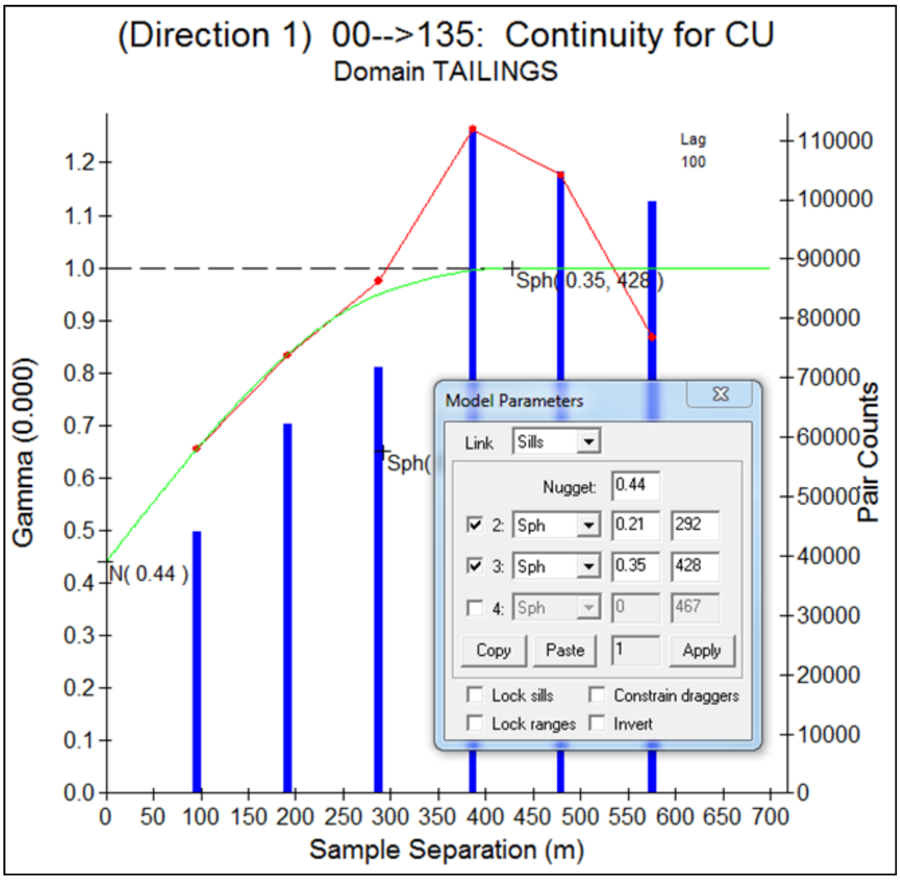

Figure 38: Anisotropic variogram model for Cu.

Nickel (Ni)

Nickel has a slightly different behaviour in the experimental semi-variogram after 300 m; the variogram drops off and has a sill (roughly at 0.95) below the normalised total sill (1), see Figure 39 below. The variogram fluctuate below 1 before it eventually rises above the total sill after 750m.

The variogram model is a nested structure and it is comprised of three spherical models. The nugget variance is comparatively high at 0.49, and the spatial variance calculated from the first lag [γ(100)] is approximately 0.75.

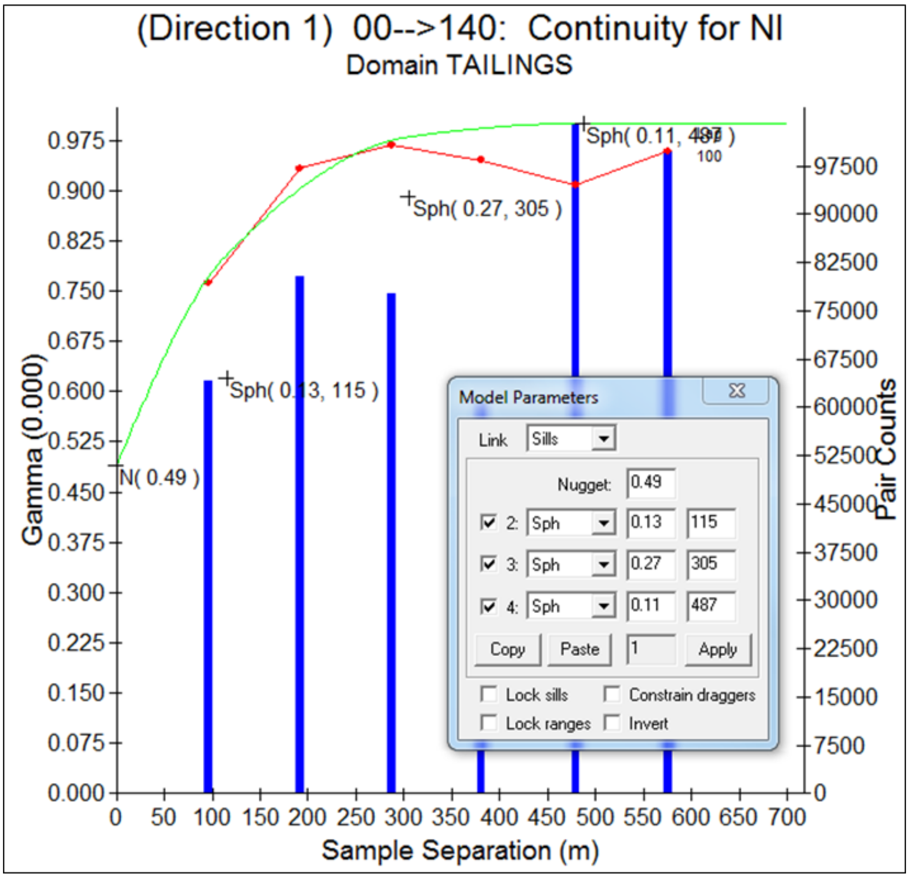

Figure 39: Anisotropic variogram model for Ni.

6.1.6. Density Variogram

The tailings dam, as expected, does not have constant or uniform density because of different compactions in tailings materials that exist from top to bottom at the dam. Furthermore, different rock types with different specific gravity have been milled at the plant. Therefore, for an accurate estimation of tonnages in the dam, density would be estimated for each cell in the block model; as such variogram is also calculated for density.

Density produced a variogram closely similar to the PGEs and Base Metals contained in the dam (Figure 40). Nugget variance is relatively low at 0.33. The normalised spatial variance calculated on the first lag [$\gamma(100)$], similarly is slightly better at 0.66. The variogram model is also a nested structure that is comprised of three spherical models.

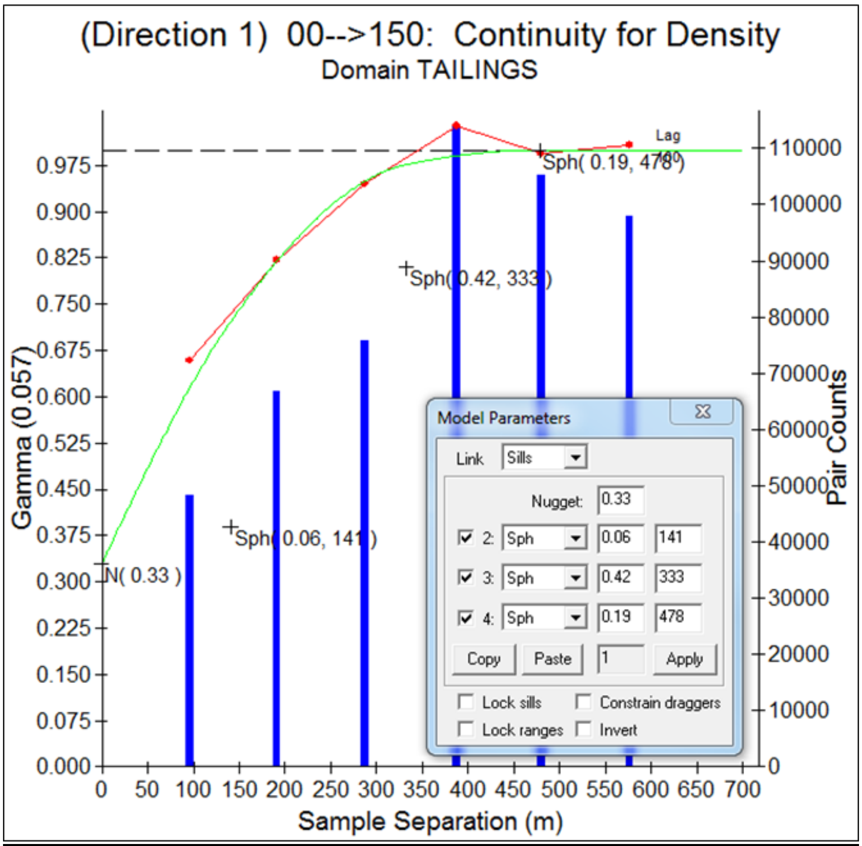

Figure 40: Anisotropic variogram model for Density.

6.2. Variogram Summary and Discussion

Surprisingly, a decent spatial correlation has been found to exist in the Case Study Tailings Dam. All seven variables have reasonable spatial grade continuity, which translates to the possibility of using kriging to perform estimation of the tailings resources. Nugget effects being realised in the dam are considered to be in line with the deposition processes that gave rise to high lithological variability. These nugget variances were derived from downhole direction.

The experimental semi-variograms calculated from sample data acquired from the dam are anisotropic and display a preferred orientation. The impact of the data limitation from undrilled portions in the dam, due to budget constraints, is not established, particularly with reference to the direction of the anisotropic variograms and nugget effect. Middle areas (penstock) of the dam, as well as two rows on the western side of the dam were not drilled due to the budget constraint (Figure 41). The orientation preferred by most Variograms is generally aligned to drilling rows, either in NE-SW or NW-SE direction. It has been noted that all variogram directions do not intersect or approach the drilling pattern obliquely, to maximise drilling information available.

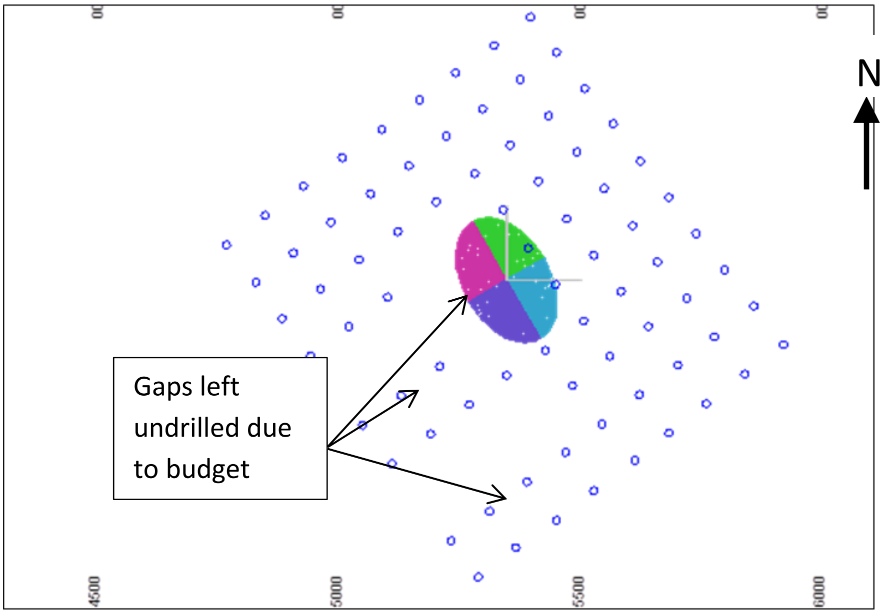

Figure 41: Undrilled portions as well as search volume superimposed on drilling grid.

The spatial variance calculated on first lag [γ(100)] is generally quite high for most of the Variograms, on average being about 0.7. Therefore, the variance modelled from 100 m to the maximum ranges accounts for less than 30% (one third) of the total sill. Due to the spacing of 100m x 100m, there is a level of unknown behaviour that exists in the variogram between 0 and 100 m, where up to 70% of the total variogram sill exists in that range. If the level of accuracy and certainty takes precedence over budget constraints, a drilling grid of 50m x 50m would have been ideal for this tailings dam to confirm estimated variogram structures at short ranges, which were modelled without any actual experimental variogram values within that range.

Variograms presented and discussed above are for the main directions of all the seven variables of interest. Secondary direction and plunge (downhole direction) were also modelled and used for estimation, Graphs (Figures) of which are presented in Appendix A. Variogram models generated on Snowden Supervisor were exported into spreadsheet, with all three directions, in a Datamine compatible format (Table 16).

Table 16: Variogram model parameters.

VDESC	VREFNUM	VANGLE1	VANGLE2	VANGLE3	VAXIS1	VAXIS2	VAXIS3	NUGGET	ST1	ST1PAR1	ST1PAR2	ST1PAR3	ST1PAR4	ST2	ST2PAR1	ST2PAR2	ST2PAR3	ST2PAR4	ST3	ST3PAR1	ST3PAR2	ST3PAR3	ST3PAR4
3E	1	100	5	-50	3	1	3	0.4	1	151	100	11	0.17	1	330	231	21	0.16	1	500	415	25	0.27
PT	2	35	5	20	3	1	3	0.52	1	118	97	8	0.22	1	320	232	16	0.11	1	473	495	25	0.15
PD	3	-145	5	-160	3	1	3	0.4	1	136	120	14	0.14	1	277	175	24	0.21	1	390	417	35	0.25
AU	4	0	0	-45	3	1	3	0.1	1	372	405	19	0.3	1	535	549	31	0.6	-	-	-	-	-
CU	5	0	0	45	3	1	3	0.44	1	292	271	17	0.21	1	428	344	30	0.35	-	-	-	-	-
NI	6	0	0	50	3	1	3	0.49	1	115	153	10	0.13	1	305	251	17	0.27	1	487	428	29	0.11
Density	7	0	0	60	3	1	3	0.33	1	141	199	15	0.06	1	333	296	25	0.42	1	478	442	31	0.19

Analysis of grade continuity and the spatial correlations that have been revealed is not only a pre-requisite to kriging estimation that will be carried out and discussed shortly, it has also answered one of the key questions of the research about mineralisation continuity in the Case Study tailings dam.

Mechanical deposition of tailings made it impractical to model geological zones (domains). Brief trends, short lenses and intermixing of different tailings materials, as well as alternating high and low grade prevented the establishment of multiple geological zones in the dam. Regardless of this difficulty experienced during geological modelling, grade continuity has been established through experimental semi-variogram calculations and subsequent variogram modelling. Mechanical deposition processes that affected geological sub-zoning, together with inherent sampling error, might have given rise to the comparatively high nugget variances being realised in the dam.

Overall reasonable spatial correlation exists for all the variables considered in the Case Study tailings dam and therefore estimation of grade through the geostatistics technique, Ordinary Kriging in this case, would be performed and will be discussed in the following chapter.

The accuracy of the resultant model estimates would also be established through model validation shortly, a critical step that addressed some of the remaining questions or objectives of the project.

CHAPTER SEVEN: GRADE ESTIMATION

7.1. Case Study Tailings Dam Grade Estimation

Grade estimation is the process whereby grade of a variable of interest is estimated through Kriging – or any other geostatistical technique – into each cell of the geological block model created during geological modelling. The estimated resource model is taken through rigorous validation process before it is subsequently used for resource classification and evaluation.

Estimation of the Case Study tailings resources was carried out using Datamine. An interactive "ESTIMATE" menu in Datamine was used to perform the estimation process. Ordinary Kriging is the geostatistical technique selected as the appropriate estimation method in the "ESTIMATE" menu, which requires the following input files to run the process:

1. Sample (grades) file – A de-surveyed and 1m-composited samples was prepared from borehole data for the process.
2. Block model – 5m x 5m x 5m block model file was generated during geological modelling process. 5 m block size was selected deliberately to standardise support size as close as possible to 5 m composites for later validation purposes.
3. Variogram models – Variogram modelling was performed using Snowden Visor (Supervisor) and imported as Datamine compatible file.
4. Search volume – A file that defines search volume ellipsoid (search or kriging neighbourhood) for each variable being estimated. Axis rotation and spatial ranges on this file were imported from Visor Variogram models. A minimum of 1 sample and a maximum of 20 samples were also specified on the file.
5. Estimation Parameters – the file that specified the estimation methods, input files and output files was referenced.

Apart from axis rotation and spatial ranges (search radii) imported from Visor Variogram models, kriging neighbourhood analysis was not conducted, therefore the block size as well as minimum and maximum number of samples were selected intuitively and on a trial basis. In retrospect, larger block size, such as 50x50x5, would have been more appropriate, as elaborated on in the Conclusion and Recommendation section.

Prior to running final grade estimation processes, the estimation parameters specified (set) in the input files were tested and assessed through Cross-validation (“XVALID”), for the optimal performance and to reveal the possible level of accuracy of the estimation results.

Cross-validation, also known as *jack-knifing* or *point kriging*, is a statistics-based method which can be used to optimise variogram models, as well as search and estimation parameters (CAE Datamine, (2013)).

In the cross-validation process, every original (actual) value in the sample file is removed and an associated new value is estimated from the remaining neighbouring samples, so that the original and the estimated values are compared for their differences.

Cross-validation results for all variables are presented in Figure 42 below: Records 1 to 7 are the cross-validation results of variables based on the final estimation parameters, which are the parameters that were applied in the actual kriging of the Case Study tailings dam. Records 11 to 17, on the other hand, represent the cross-validation results of the seven variables derived from the original (initial) estimation parameters before minor adjustment were effected.

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RECORD	ID (N)	VALUE IN...	VALUE O...	IMETH...	SRE...	VREF...	P...	NUM...	NU...	ACT MEAN ...	EST MEAN (N)	DIFF (N)	PC DIF...	M...	ACT VAR ...	EST VAR ...	COR COE...	KV VMOD (N)
1	1	3E	3E	3	1	1	2	3232	0	0.78439645	0.78439645	0	0	0	0.00942217	0.00942217	1	0.53258836
2	2	PT	PT	3	2	2	2	3232	0	0.47881666	0.47881666	0	0	0	0.00334964	0.00334964	0.999	0.68581669
3	3	PD	PD	3	3	3	2	3232	0	0.25315804	0.25315804	0	0	0	0.00241765	0.00241765	1	0.51529002
4	4	AU	AU	3	4	4	2	3232	0	0.05218859	0.05218859	0	0	0	0.00034812	0.00034812	0.999	0.18550361
5	5	CU	CU	3	5	5	2	3232	0	0.00784518	0.00784518	0	0	0	0.00000209	0.00000209	1	0.55566698
6	6	NI	NI	3	6	6	2	3232	0	0.0905203	0.0905203	0	0	0	0.00003301	0.00003301	1	0.63602396
7	7	density2	density2	3	7	7	2	3232	0	1.87342617	1.87342617	0	0	0	0.03503354	0.03503354	1	0.43685355
8	8	3E	3ELG	3	1	1	2	3232	0	0.78439645	1.01280873	0.22841228	-29.119	1228	0.00942217	0.01455599	0.997	0.23558696
9	9	CU	CULG	3	5	5	2	3232	0	0.00784518	0.01021987	-0.0023747	-30.269	3747	0.00000209	0.00000316	0.997	0.00002058
10	10	density2	densLG	3	7	7	2	3232	0	1.87342617	2.3312843	0.45785813	-24.439	5813	0.03503354	0.05488616	0.998	1.0897235
11	11	3E	3E	3	1	1	2	3232	0	0.78415794	0.78415794	0	0	0	0.00914475	0.00914475	1	0.57585239
12	12	PT	PT	3	2	2	2	3232	0	0.47843277	0.47843277	0	0	0	0.0033625	0.0033625	1	0.68570719
13	13	PD	PD	3	3	3	2	3232	0	0.25317192	0.25317192	0	0	0	0.00234889	0.00234889	1	0.60609235
14	14	AU	AU	3	4	4	2	3232	0	0.05210063	0.05210063	0	0	0	0.00034631	0.00034631	0.999	0.31166338
15	15	CU	CU	3	5	5	2	3232	0	0.00784148	0.00784148	0	0	0	0.00000208	0.00000208	1	0.60663209
16	16	NI	NI	3	6	6	2	3232	0	0.09053959	0.09053959	0	0	0	0.00003568	0.00003568	1	0.54635764
17	17	density2	density2	3	7	7	2	3232	0	1.87384948	1.87384948	0	0	0	0.03665105	0.03665105	1	0.45512109

Figure 42: Cross-validation results.

These results in Figure 42 above suggest that the input parameters are sufficient and optimally set for ordinary kriging of the raw (untransformed) data. The actual and estimated means of the seven variables (first seven records) are identical. The same hold true for the variances: actual and estimated variances are also identical for all seven variables (the first seven records). Therefore, Ordinary Kriging of this tailings resource is generally expected to be an unbiased and accurate estimation method.

The three variables (3E, Cu and Density) that have been selected for log-normal testing also underwent cross-validation process. These variables would be estimated twice: raw (untransformed) as well as log normal data. Cross-validation results of the three log-transformed variables are presented by Records 8 to 10 (Figure 42). There is a huge difference between the mean calculated from original data and that from log-transformed grades. Lognormal values are considerably higher than the actual mean, which are 29, 30 and 24% higher in 3E, Cu and Density respectively. It was concluded that this is an inappropriate estimation method.

Final grade estimation into geological block model was performed for all seven variables (six metals and Density) through kriging of untransformed data.

A brief review of the estimated resource (block) model was carried out to ascertain whether the model is preliminarily acceptable. It was also performed to establish whether gaps or cells with blank estimates exist in the resource model (Figure 43). 3D view of the model confirmed that all cells along the perimeter of the resource model were estimated because no gaps (cells coloured in grey) could be noted.

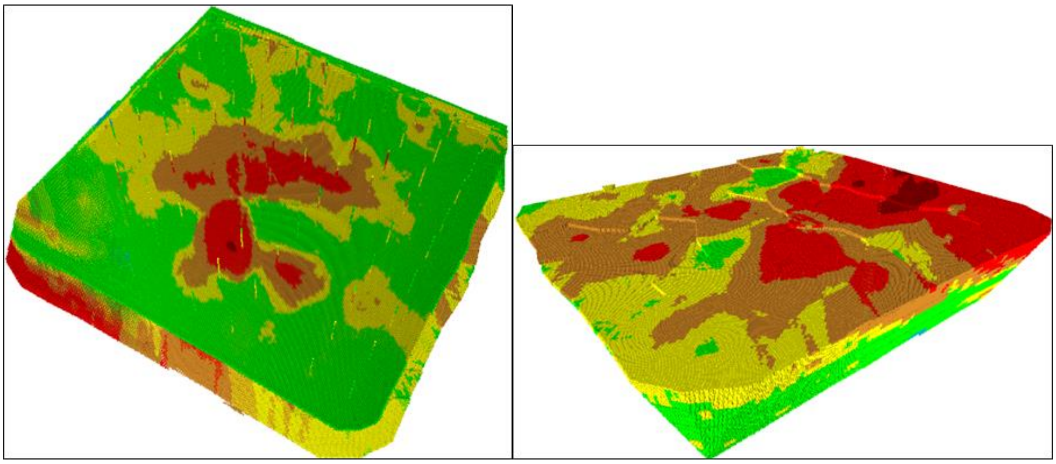

Figure 43: Top view and bottom view of the full resource model.

Furthermore, 5m composited boreholes were loaded with the resource model to review the estimated cells against borehole data. Sections were cut along the drilled pattern to confirm that the model is representative of the sample data (Figure 44(a) to 44(d)).

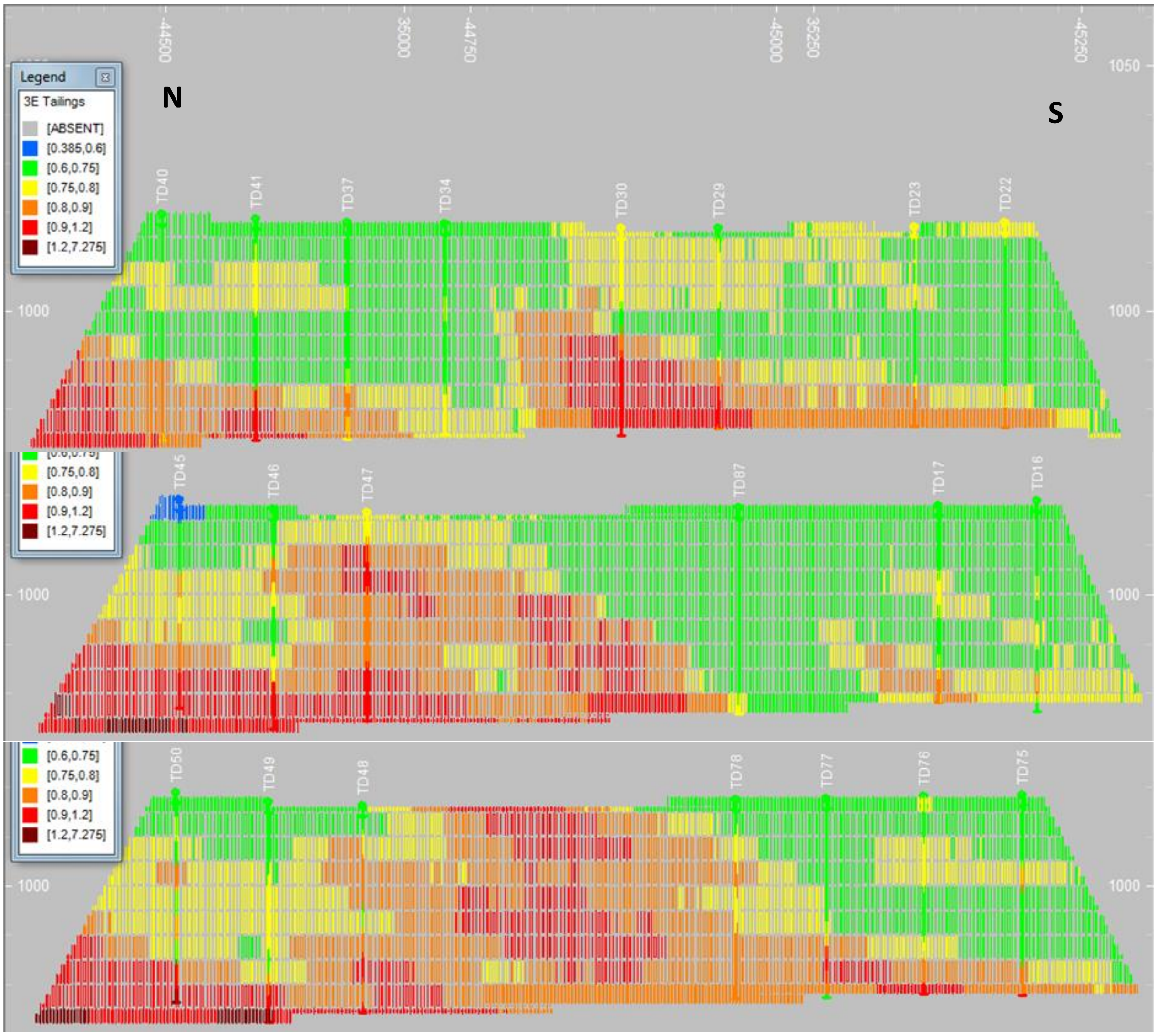

Figure 44(a): N-S sections cut across the resource model along drilled rows.

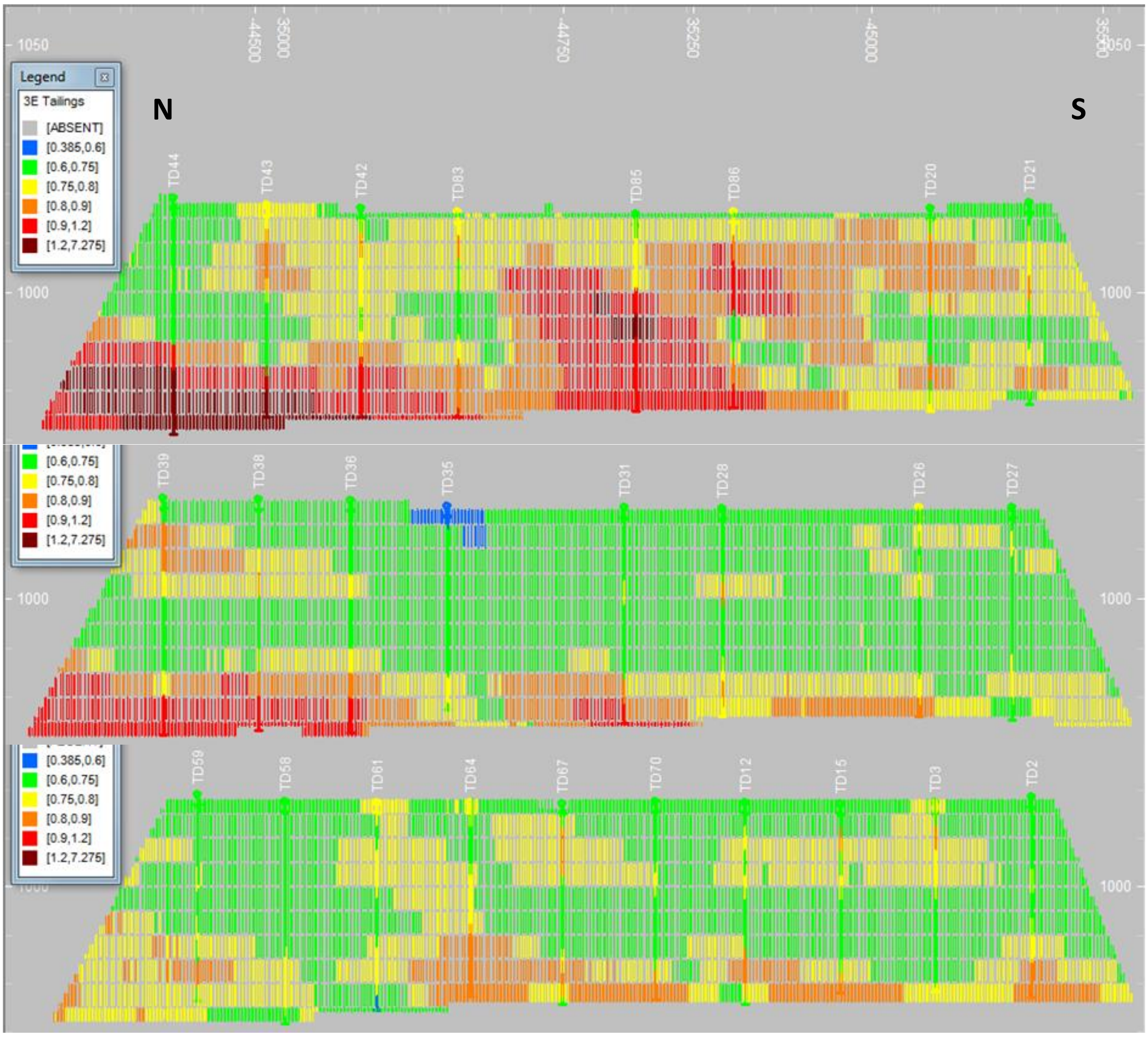

Figure 44(b): N-S sections across the resource model along drilling rows.

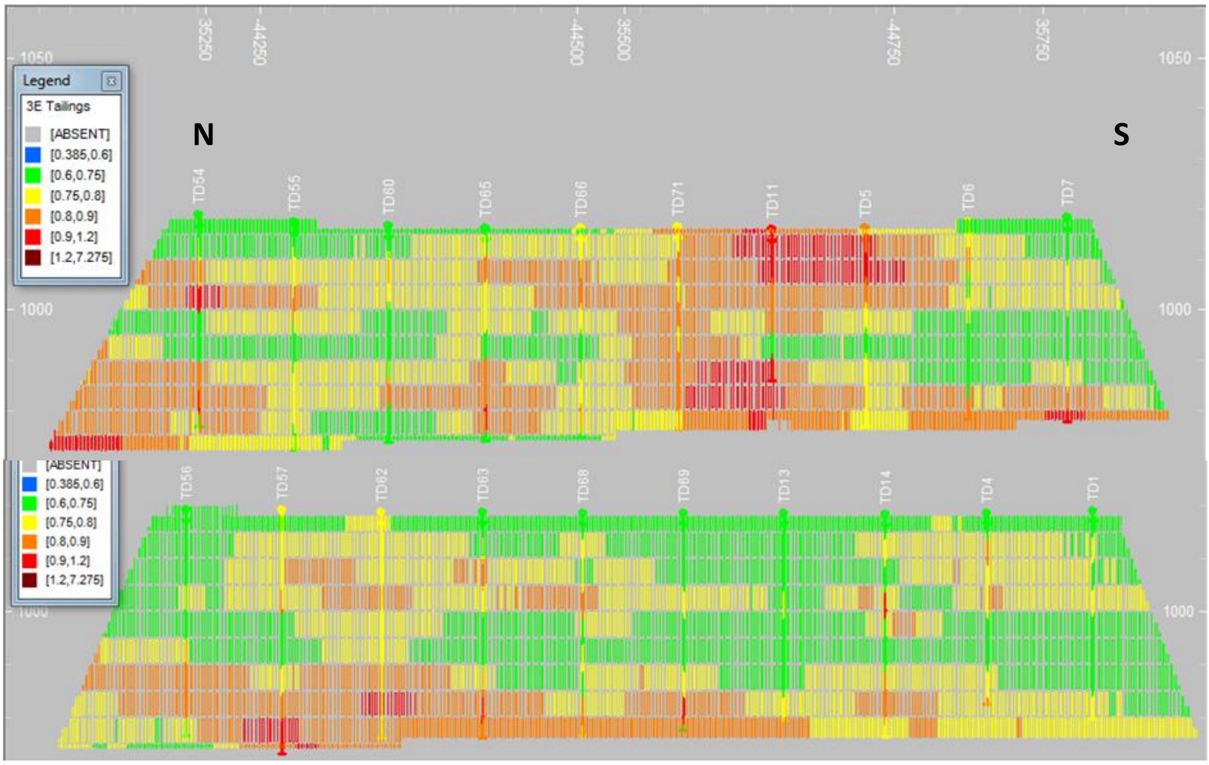

Figure 44(c): N-S sections across the resource model along drilling rows.

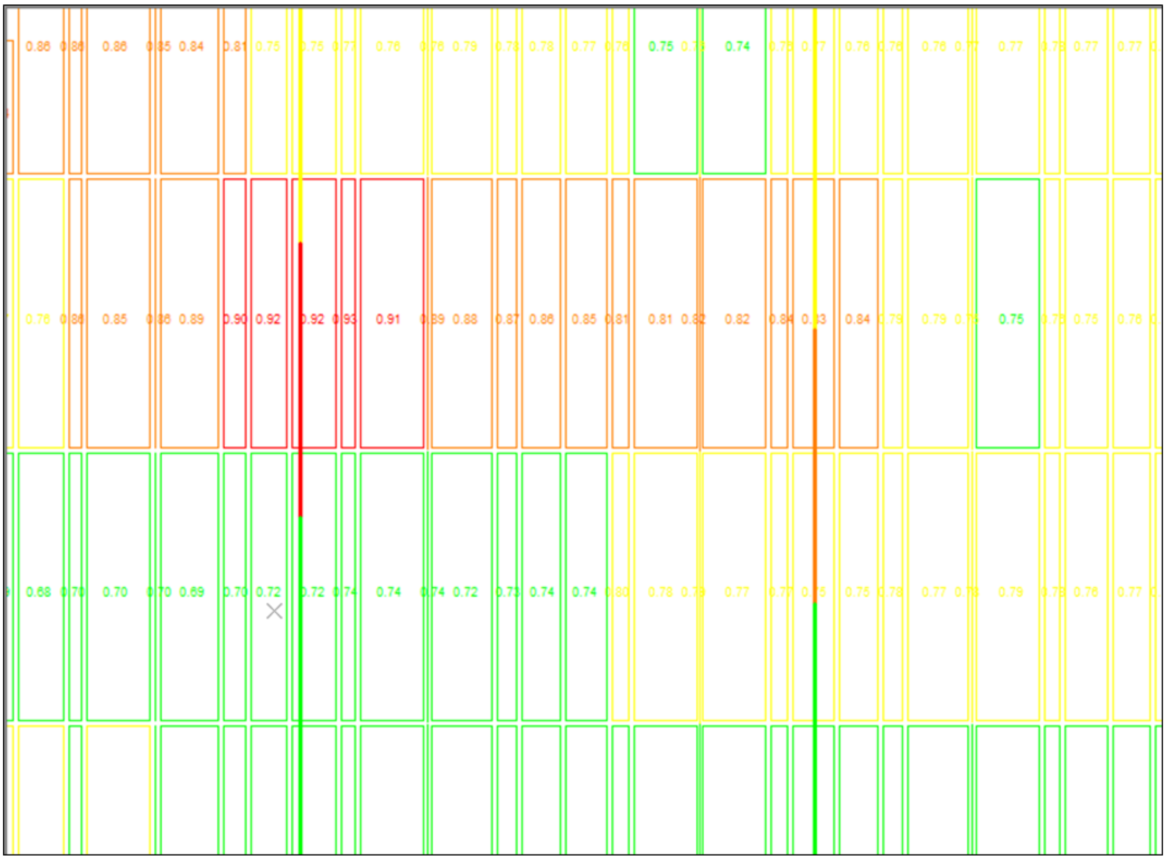

Figure 44(d): N-S section across the resource model along drilling rows, with actual grade values.

A review of the resource model confirmed that estimation carried out was successful and all cells were estimated for all seven variables of interest. Furthermore, grade profile plotted or loaded from resource model matched that from the 5 m composited borehole data very well. Therefore, the resource model passed or satisfied the initial review, and more rigorous validation process then followed, which is discussed in the next chapter.

CHAPTER EIGHT: MODEL VALIDATION

Validation is the process that is undertaken to confirm whether the resource estimation that has been carried out produced a resource model that reasonably reflects and compares to the original data (sample data) that was used in the estimation process.

In the Case Study tailings resource estimation, the model validation is the rigorous and comprehensive quality assurance that was carried out in addition to the two brief reviews undertaken previously, which are:

- visual inspection of grade profiles between resource model and borehole profiles;
- and the pre-emptive cross-validation process that was performed to optimise estimation parameters before the actual estimation was carried out.

The borehole data used in the validation process was 5 m long composites, to bring its support size in line with cells of the 5m x 5m x 5m block model used for estimation.

Validation was carried out through summary statistics, histograms and QQ Plots:

8.1. Summary Statistics.

Statistical validation involved comparing key statistical parameters of borehole data against resource model (Table 17). Margin of the differences, between parameters of the two data sets were used to give an indication on quality of the estimated model.

Table 17: Resource model versus 5m BH composites summary statistics.

VARIABLE	TYPE	NSAMPLES	MIN	MAX	RANGE	MEAN	% DIFF OF MEAN	VAR	STD DEV	STD ERR	SKEWNESS	KURTOSIS
3E	Borehole	681	0.523	1.458	0.935	0.786	-1.668	0.01351	0.116	0.00445	2.0	6.55
	RES Model	635403	0.556	1.393	0.837	0.802		0.01196	0.109	0.00014	1.7	3.88
PT	Borehole	681	0.334	1.037	0.703	0.479	-0.805	0.00498	0.071	0.00270	3.3	19.55
	RES Model	635403	0.352	0.944	0.592	0.485		0.00360	0.060	0.00008	2.6	11.00
PD	Borehole	681	0.172	0.561	0.389	0.254	-0.932	0.00336	0.058	0.00222	2.3	6.42
	RES Model	635403	0.178	0.516	0.338	0.257		0.00243	0.049	0.00006	1.8	3.49
AU	Borehole	681	0.013	0.115	0.102	0.053	-1.049	0.00036	0.019	0.00073	0.4	-0.46
	RES Model	634932	0.014	0.110	0.096	0.054		0.00042	0.021	0.00003	0.0	-1.17
CU	Borehole	681	0.005	0.018	0.012	0.008	-0.755	0.000003	0.002	0.00006	2.4	8.62
	RES Model	634608	0.006	0.016	0.010	0.008		0.000003	0.002	0.00000	2.0	5.28
NI	Borehole	681	0.075	0.113	0.038	0.090	-0.233	0.00004	0.006	0.00025	0.7	0.67
	RES Model	634408	0.078	0.113	0.034	0.090		0.00003	0.006	0.00001	0.8	0.83
density	Borehole	681	1.183	2.429	1.246	1.886	-1.141	0.04442	0.211	0.00808	0.3	-0.38
	RES Model	635403	1.301	2.377	1.075	1.901		0.04553	0.213	0.00027	-0.1	-0.82

The main and critical statistical parameter assessed is the global average grade of the two data sets for all variables estimated; as such difference (in percentages) between these averages (means) are presented in Table 17 above. The highest difference (worst value) is still a very reasonable -1.67% for the 3E data sets. Negative value indicates that the estimated 3E average is 1.67% higher than the borehole data mean. The differences between averages of all grade variables suggest that a good and representative global resource model was produced from the Case Study tailings dam.

Individually, elements that comprise the 3E returned even smaller differences; Pt estimated global average is only 0.81% higher than the borehole data mean. Pd estimation also returned a very reasonable global average that is only 0.93% higher than the borehole data mean. Density has the second highest difference, yet the average is still reasonably 1.14% higher in estimated results than in the 5m composited boreholes. Au has the third highest difference in these global averages noted from the case study estimation, coming after 3E and Density; it is 1.05% higher in the resource model than the borehole average.

The remaining two grade variables (base metals), likewise, returned good and comparative global estimates. The Cu estimated average is only 0.76% higher than that from borehole data. Ni is the best estimated variable; the difference between averages of the two data sets is a mere -0.23%.

Statistical validation through comparison between estimated averages from resource model to that of borehole data give some assurance of the kriging estimation performed, therefore the results are considered reliable and representative of the resources contained in the Case Study tailings dam.

8.2. QQ Plots and Histograms

QQ-Plot is a graph of the quantiles of a grade of one data set (resource model) against the quantiles of another data set (boreholes). A quantile is defined as the fraction (or percent) of the number of data points below a given value. If the two data sets come from a population with the same distribution, the points should fall approximately along the 45-degree reference line (CAE Datamine, (2013)).

It must be noted that the QQ-plots used for validation purposes in this research have been generated from supports that are not entirely compatible, 5 m core (composites) vs 5m x 5m x 5m blocks. The technique, however, has practical benefits in establishing a reasonable indication or comparison of the distributions of the two data sets.

8.2.1. 3E

3E quantiles plot approximately along the 45-degree line for most data points (Figure 45). The only noticeable deviation from the reference line is noted on points starting approximately at 0.679 g/t and below, which accounts for less than 10% of the total data.

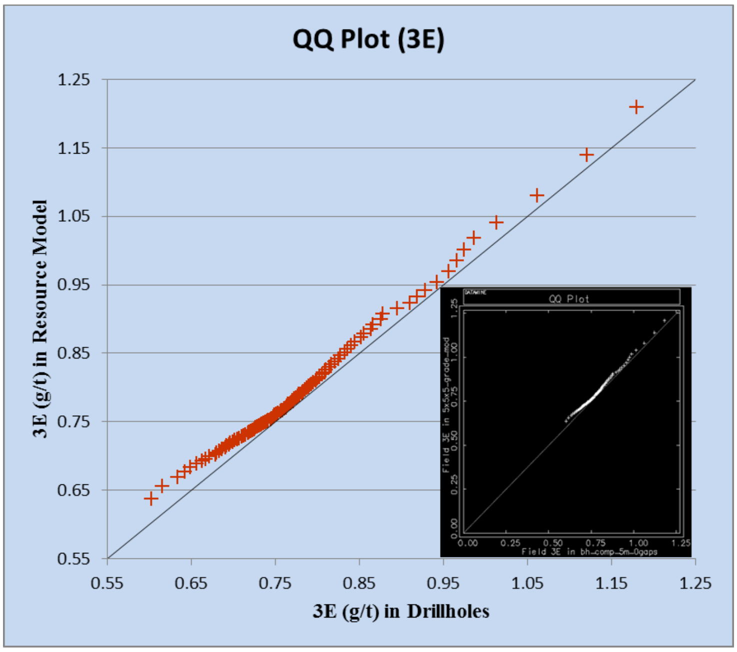

Figure 45: Reproduced 3E QQ Plot (with original Datamine plot insert).

The distribution of the two data sets, except for the deviations at the top and bottom ends, plots close to each other. The differences at the extreme end are because of the different supports, which is leading to slightly different variability in those grade values at both ends. Therefore, the conclusion from the QQ-plot above is that the two data sets come from similar shaped populations.

The histograms shown in Figure 46 support the conclusion drawn above concerning the 3E estimation. The effect of different support sizes is also evident in the figure, with the smallest support size (1m samples) having the biggest spread of the data.

The 3E validation confirms that Ordinary Kriged estimates closely resemble borehole data from which estimation was derived, and therefore geostatistical technique (Ordinary Kriging) is a good estimation method for a reliable and representative evaluation of the Case Study tailings dam.

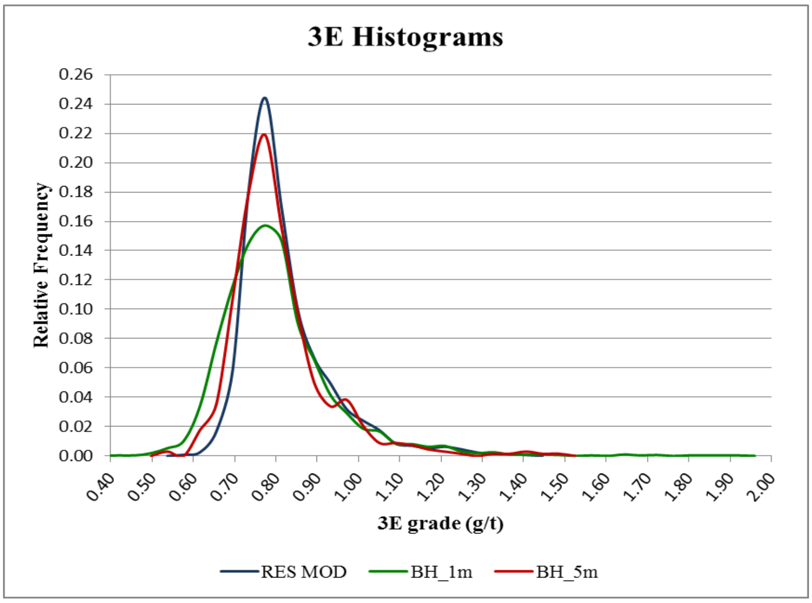

Figure 46: 3E Resource model histogram against boreholes (1 m and 5 m) Composites.

8.2.2. Platinum (Pt)

The Platinum QQ Plot resembles 3E very closely, with just some minor differences at the top quarter (Figure 47). Deviation from the reference (45°) line, likewise, is slightly noticeable at lower grade values. It is further noted that a gap exists between 0.62 and 0.79 g/t where there are no data points plotted. Only one point plots after the gap at approximately 0.8 g/t. The Pt QQ Plot therefore indicates very similar distribution between 5m composited boreholes data and block estimated results (resource model).

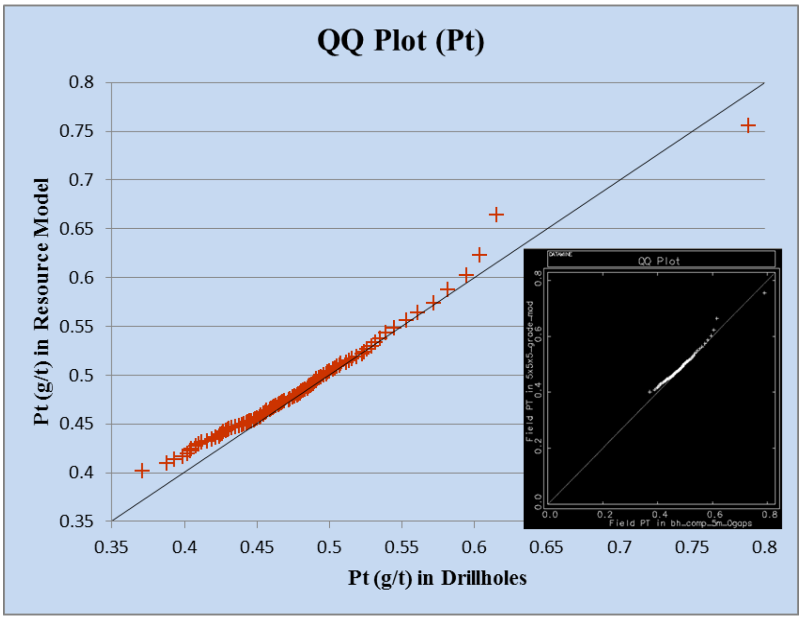

Figure 47: Reproduced Pt QQ Plot (with original Datamine plot insert).

Individual distributions of the three Pt data sets are plotted on histograms on Figure 48 below. The 5 m composited borehole data is distributed almost identical to the estimated resource model, the expected effect of the change of support is also evident in the tails of the distributions. Therefore, the QQ plot and histograms reasonably affirm positively the estimation carried out for Pt grade.

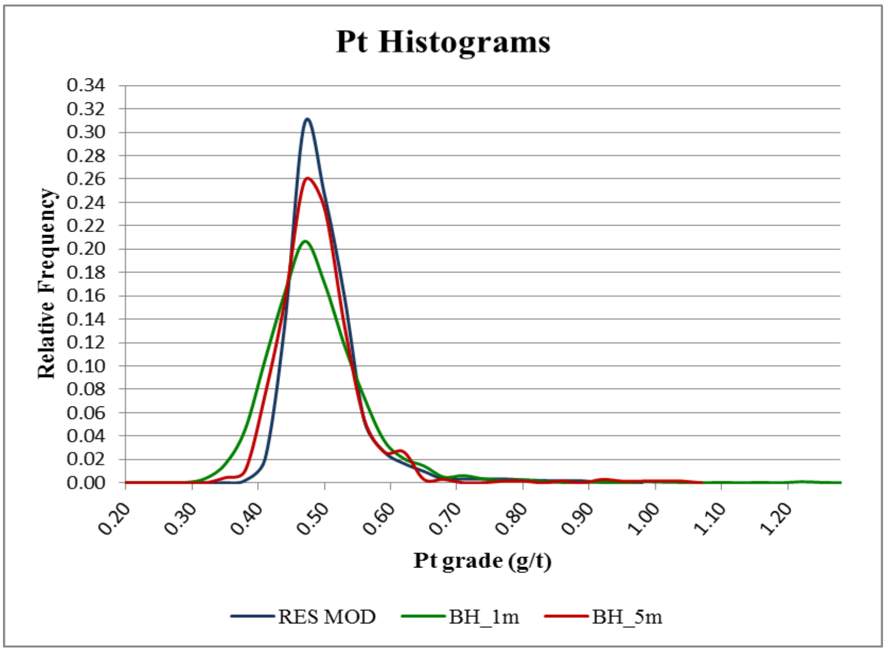

Figure 48: Pt Resource model histogram against boreholes (1 m and 5 m) Composites.

8.2.3. Palladium and Gold

Palladium (Pd)

The quantiles on the Palladium QQ Plot (Figure 49), like other PGEs, do not deviate by a significant margin from the reference line. The only noticeable deviation is on the three data points at the higher end of the grade spectrum, which accounts for less than 3% of the total data set.

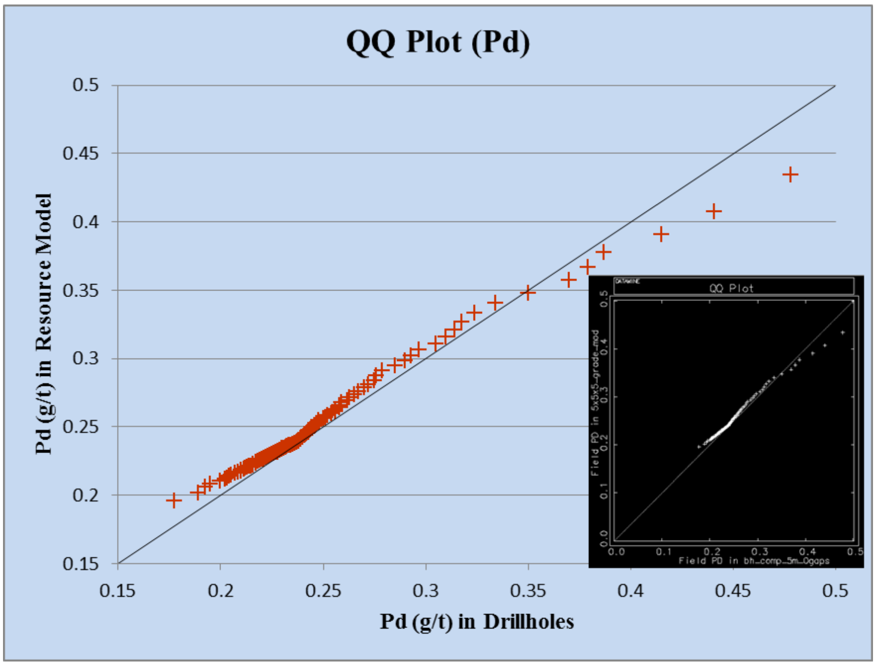

Figure 49: Reproduced Pd QQ Plot (with original Datamine plot insert).

The Histogram plots of the Pd data sets (Figure 50) are behaving as expected. The resource model grade distribution has assumed a shape very much in line with the distribution noted in borehole data. As expected, the spread is narrower in bigger support sizes and vice versa.

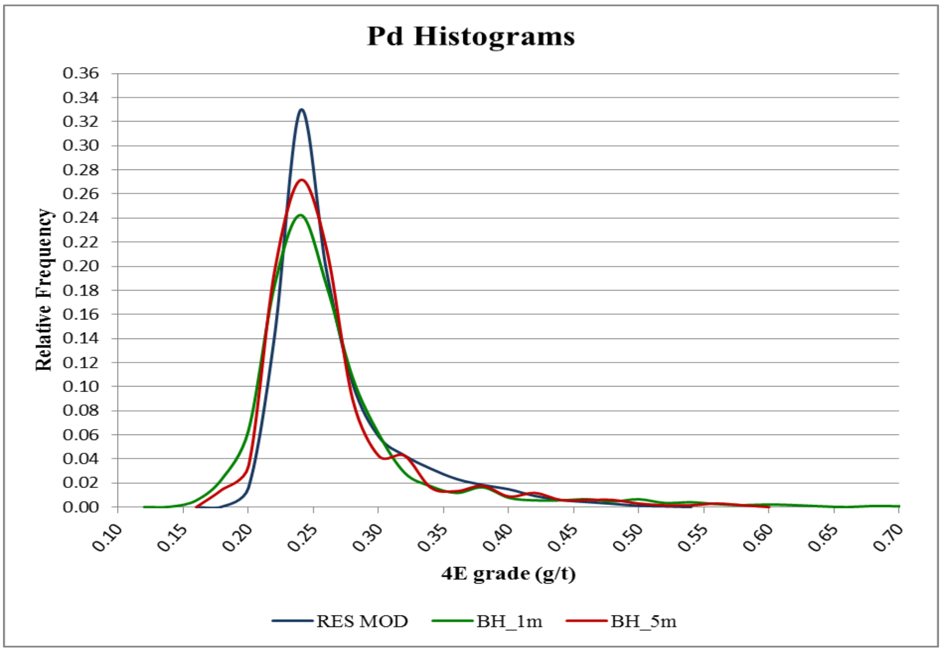

Figure 50: Pd Resource model histogram against boreholes (1 m and 5 m) Composites.

Gold (Au)

The Gold QQ plot is oscillating around the reference (45°) line (Figure 51), which is due to the possible multiple Au populations in the distribution, as identified earlier in Section 2.4.3. The minor slump at the bottom quarter, followed by bump or lifting from the reference line after mid-point, affirms this possibility.

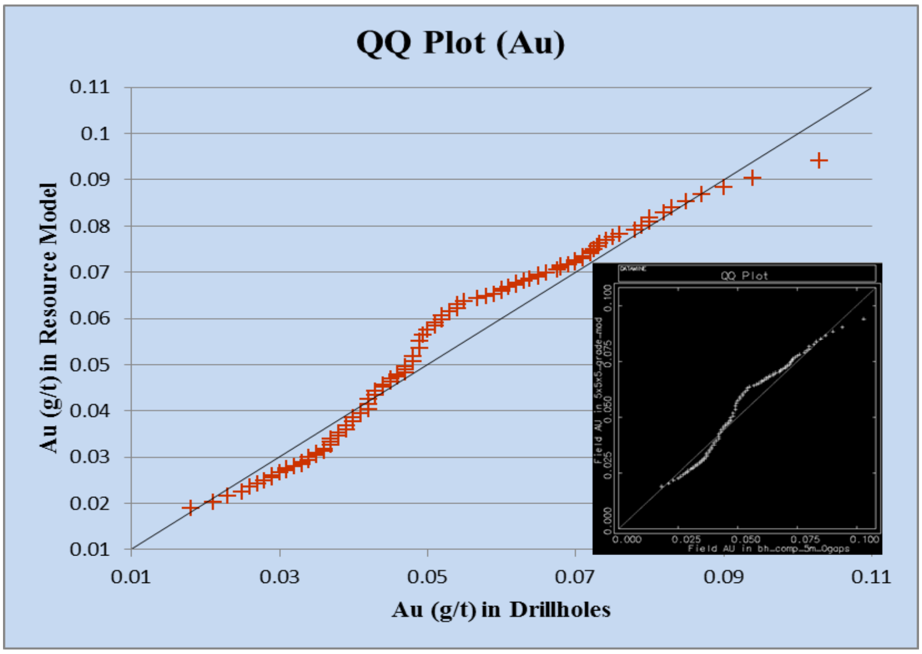

Figure 51: Reproduced Au QQ Plot (with original Datamine plot insert).

Histogram plotted from resource model and 5 m borehole composites evidently magnifies the existence of those populations in the data (Figure 52). Therefore, the bigger support size is clearly replicating the different populations that exist in the data. As such, Au grades from the Case Study tailings dam can be confidently declared as coming from at least two distribution populations. Overall, the Au validation confirms that the original data is reasonably represented or replicated by the resource model. Therefore, kriged estimates of Au data are acceptable.

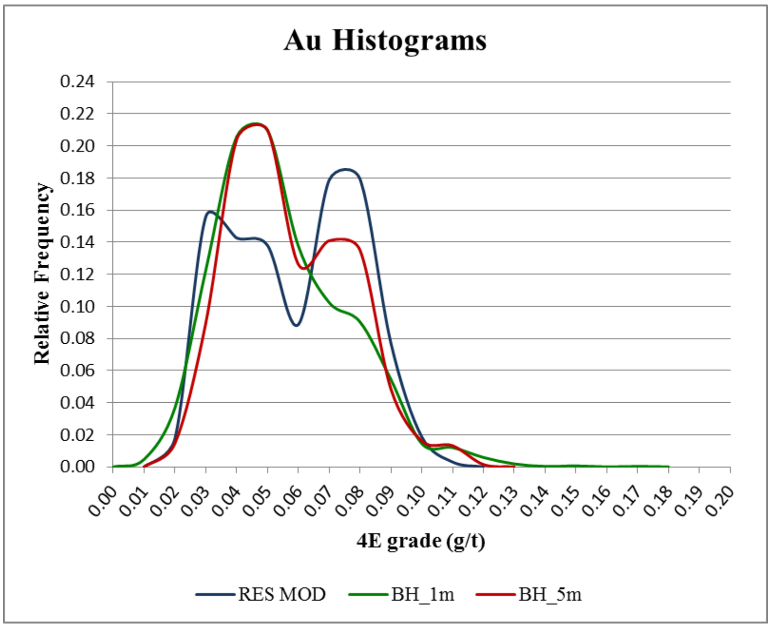

Figure 52: Au Resource model histogram against drillholes (1 m and 5 m) Composites.

8.2.4. Base metals (Cu and Ni)

Copper (Cu)

The QQ plot of Copper, as experienced in several other grade variables validated so far, is plotting tightly along the reference (45°) line (Figure 53). Furthermore, up to 80% of the data is concentrated at the bottom third of the graph, which is also where the data plots perfectly and touches the reference line. The slight deviation in the tails is due to the change of support and the possibility of more than one Cu distribution existing in the data.

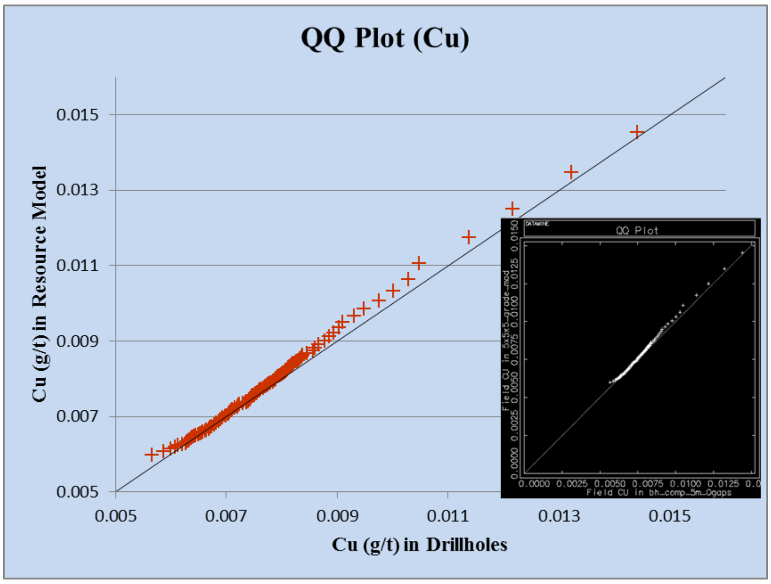

Figure 53: Reproduced Cu QQ Plot (with original Datamine plot insert).

Distribution of the three data sets, as plotted in Figure 54, is virtually identical. The slight and unusual difference is at the peak, unlike most other grades, 5 m composited boreholes peaks slightly higher than the resource model distribution. Overall the estimated results are concluded to be a good reflection of the boreholes data used to derive the estimation.

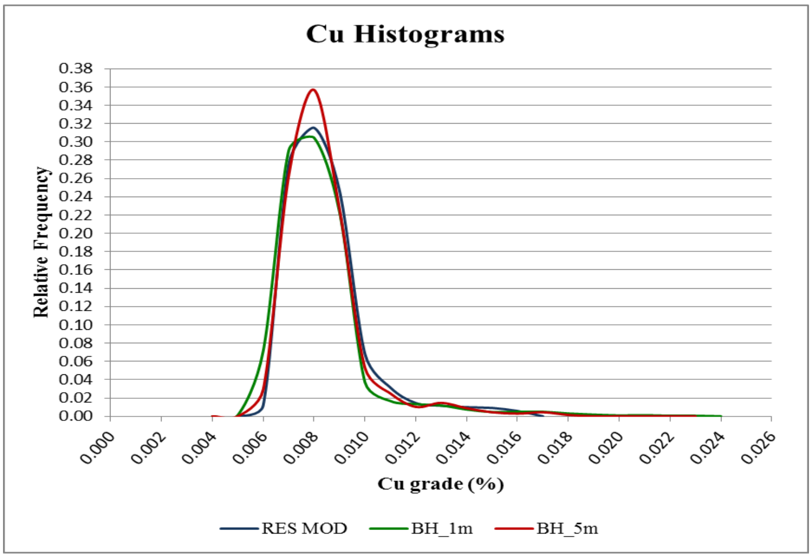

Figure 54: Cu Resource model histogram against boreholes (1 m and 5 m) Composites.

Nickel (Ni)

Nickel QQ Plot, as with other variables, confirms the similarity between boreholes composites and block estimates. The Quantile data plots very tightly along the reference line (Figure 55).

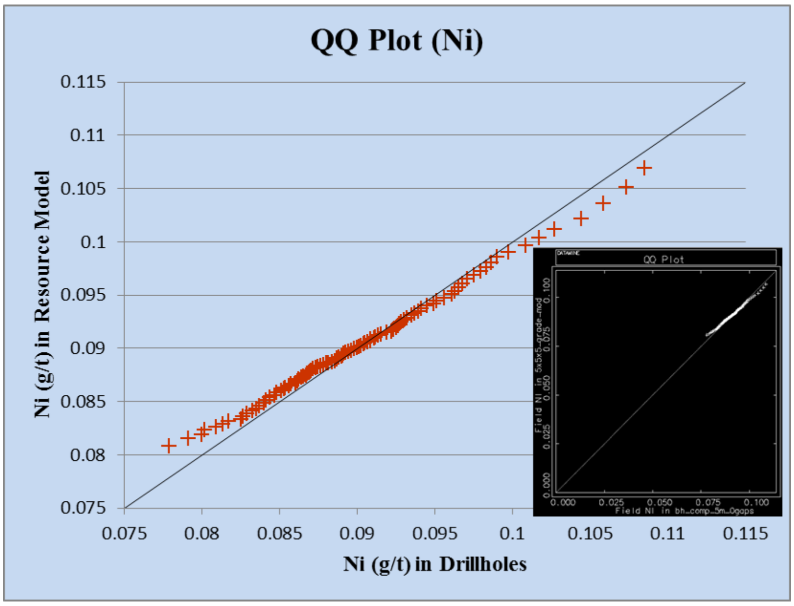

Figure 55: Reproduced Ni QQ Plot (with original Datamine plot insert).

The distributional behaviour of the data sets in Figure 56 is viewed as normal or typical of the Case Study tailings dam. As expected, high dispersion is noted in data with smaller support size, likewise less dispersed distribution is realised in bigger support sizes. Furthermore, estimated grades peak slightly higher than 5 m composited boreholes data, a typical occurrence in other variables.

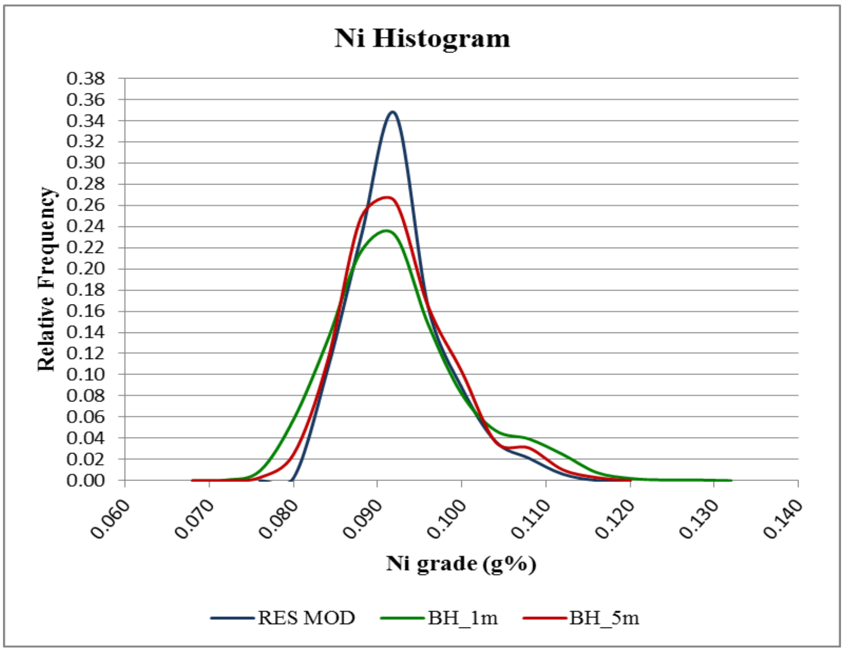

Figure 56: Ni Resource model histogram against boreholes (1 m and 5 m) Composites.

8.2.5. Density

Density is the last variable to be validated. Its QQ plot (Figure 57), is different in that the graph barely touches the reference (45°) line. Although the graph does not necessarily plot exactly on the reference line, the deviation is not severe or a complete turnaround from what has been witnessed in other variables of the Case Study tailings dam. This deviation is largely attributed to the observation that there are possibly more than one density distributions at play in this tailings dam, and exacerbated by the change of support effect between the drill hole and resource model grades.

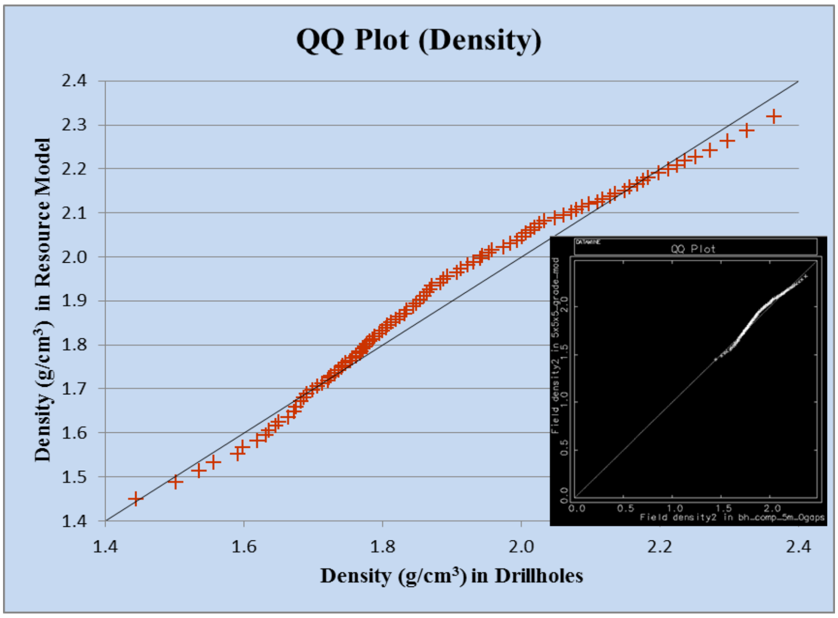

Figure 57: Reproduced Density QQ Plot with (original Datamine plot insert).

The resource model distribution plotted in Figure 58 confirms the existents of at least two populations, as is the case with the boreholes data. Furthermore, the shapes of the distributions of the data sets closely resemble each other. Density QQ plot and the relationship between resource model and 5 m composite data in the histogram are reflective of the quality and reliability of the estimation carried out on density.

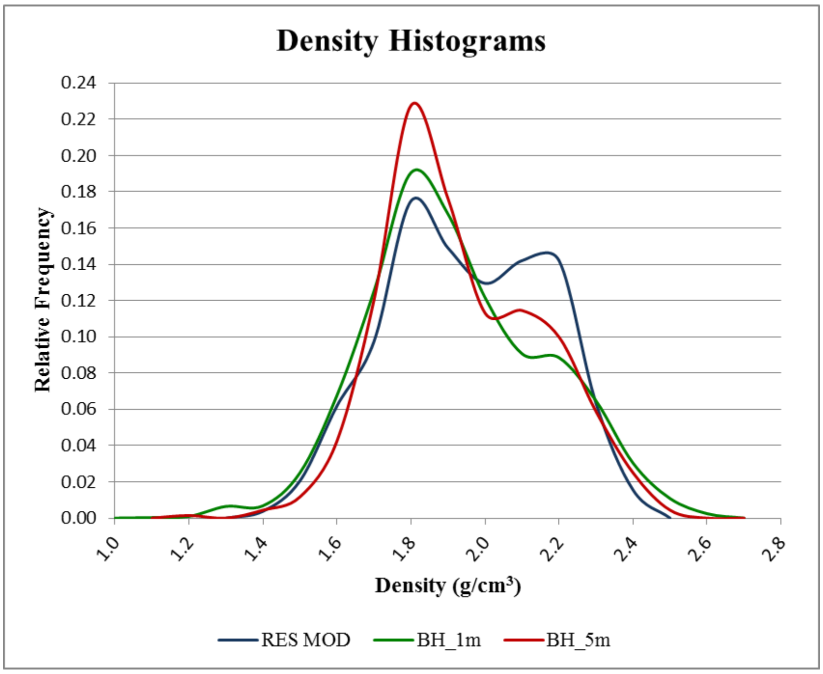

Figure 58: Density Resource model histogram against boreholes (1 m and 5 m) Composites.

8.3. SUMMARY ON VALIDATION

The three-step comprehensive validation process undertaken in this Section is conclusive regarding the case study project. As validated by summary statistics (Table 17 in Section 8.1), differences in the means of the two data sets are minimal on all variables, with the highest difference a reasonable 1.67%.

Furthermore, QQ Plots convincingly confirms that resource model and sample data comes from similar distributions. The histograms of all data sets were further plotted together to confirm the change of support effect, and to assess the closeness of distribution shapes, particularly between resource model and 5 m composited boreholes. In all cases both these requirements were met and allowed for conclusion regarding the kriged estimation method.

Difference between samples and estimated grades, although fairly small and within acceptable ranges as far as the scope of this Research Report is concerned, could possibly be as a result of several factors: kriging neighbourhood analysis that was not undertaken, support size, inherent sampling error, nugget effect, other kriging parameters or even kriging method itself used to conduct estimation.

Validation of the Case Study tailings resources, overall, has revealed a resource model that is acceptable and that reasonably reflects the original sample data. It also gives confidence on Kriging estimation method and its application on estimating tailings resources.

CHAPTER NINE: DISCUSSION

The Case Study tailings resource was successfully evaluated (estimated) using a standard approach to collect samples and analyse data from the laboratory, which involved:

- Drilling
- Sampling
- Assaying
- Statistical analysis
- Geological modelling
- Variography analysis
- Grade estimation, through Ordinary Kriging – a geostatistical technique
- And Resource (grade) model validation

9.1. Drilling, Sampling and Assaying

Sonic drilling, a specialised drilling technique that does not require drilling fluids when drilling loose particles, was successfully employed to extract the full profile of the tailings dam for sampling. Good recovery of tailings was also realised.

Contamination and poor recovery initially encountered on wet samples were the two main challenges that needed to be understood and addressed in the early part of the project. This contamination is generally a consequence of unconsolidated tailings falling off the side walls of borehole being drilled, or material that is being shaved off whilst the core barrel and rods are being retrieved or returned into the hole to drill the next run.

To establish and/or quantify the amount of contamination to be discarded from the sample, several logging factors (parameters) were considered; such as moisture content, colour contrast, tailings material difference (lithology), and compaction. Contamination or the contaminated part was often not consistent with the actual sample being recovered, and these factors were used to separate the two materials.

The short fall in identifying contamination is that the process is not very straight forward and clear cut as expected but it is different from sample to sample or from borehole to borehole. Furthermore, there is an element of subjectivity that often gets introduced, especially in projects being logged by more than one geologist alternating one another. The case study project was sampled by the author throughout, and it is therefore concluded that the effect of subjectivity was greatly reduced. As such, the component of inconsistency in the sampling processes could only have been minimised by such practise, thereby promoting better analysis, particularly geological modelling which is heavily based on logging.

The second challenge was the recovery of tailings in the wet zone of the dam. Tailings saturated with water often slid back into the borehole when core barrel was being pulled up for sample retrieval. This was the biggest problem with the first two or three boreholes where poor recovery to no recovery was being realised in the deepest section of the dam. A device called "Core catcher" was introduced to solve the problem. This device was inserted inside the bit to act as a one-way valve that only allows sample material in and not out (Figure 59).

Figure 59: Core Catcher to improve recovery of wet tailings.

The diameter of the core barrel used for sonic drilling, as pointed out previously, is 9.9 cm. This in turn implies that huge samples were being generated and sent to the lab. Smaller machines of half that diameter, that has the capacity and power to penetrate tailings as deep as 60 m could not be supplied in time for the drilling project. The downside of a big drilling rig is that samples as big as 30 kg were being sent to the lab (Figure 60), the issue is that many standard laboratories are not often built to handle drilling samples that big. Therefore, proper homogenisation of such large samples can be a big challenge. Furthermore, splitting or selecting a representative lot from the sample is undoubtedly open to error. It is a process that is most probably left to input from the lab analysist rather than to a system automated to randomly and independently select a properly homogenised lot.

The problem of large samples was partly addressed by sampling per metre instead of the initial intended 2 m long samples. In addition, the laboratory contracted to do the analysis was required to modify their processes to accommodate huge samples being generated.

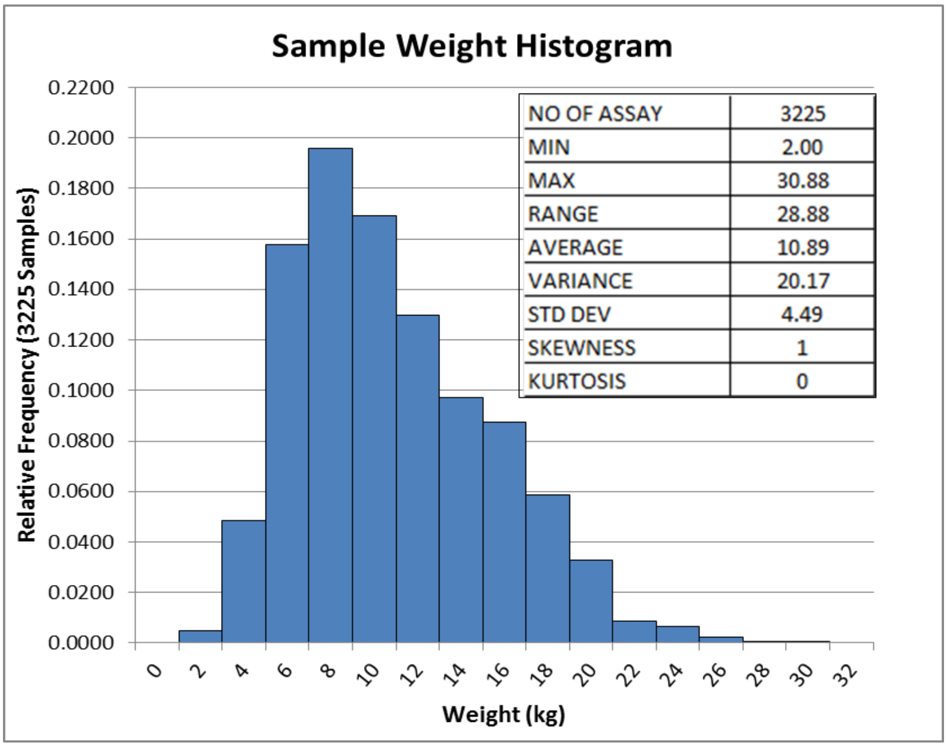

Figure 60: Histogram of sample weights sent to the lab.

Although some improvements can still be made for future projects, sampling conducted is concluded to be the true representation of the Case Study tailings dam. Sonic drilling is therefore one of the best possible ways of drilling and sampling tailings dam.

9.2. Statistical Analysis

Data acquired from the Case Study tailings dam was analysed statistically to reveal the underlying distribution of each grade variable to be estimated. Histograms, cumulative frequency curves and summary statistics tables were generated for a complete overview of the behaviour of each variable (Pt, Pd, 3E, Au, Cu, Ni and Density).

Distributions of most variables have been revealed to be non-symmetrical and slightly positive skewed. The magnitude of the non-symmetry, as deduced by other statistical parameters, was noted to be marginal in most cases.

Critical parameters for all variables were also compiled into one table for a holistic conclusion of the Case Study project (Table 18). Although most of the data is marginally skewed, the difference between the mean and log-normal estimate is negligible. The biggest difference is -0.54% for Au, followed by 0.31% and 0.26% in Pd and Cu respectively. The rest of the variables have almost zero differences. Statistical analyses therefore proved that Ordinary Kriging (OK) raw data would be appropriate for an unbiased estimation.

Table 18: Comparison of some of the parameters from variables summary statistics.

Variable	Skewness	Std Dev	Mean	Log-normal Estimate	Mean vs Log-normal Estimate		Coefficient of Variation
					Diff	% Diff	
3E	2	0.143	0.783	0.783	0.0006	0.08	0.18
Pt	4	0.091	0.478	0.478	0.0006	0.12	0.19
Pd	3	0.068	0.253	0.252	0.0008	0.31	0.27
Au	1	0.021	0.052	0.052	-0.0003	-0.54	0.39
Ni	1	0.0079	0.091	0.091	0.000005	0.01	0.09
Cu	3	0.0019	0.0078	0.0078	0.00002	0.26	0.25
Density	0	0.238	1.873	1.873	-0.0001	0.00	0.13

Nevertheless, out of interest three variables (3E, Cu and Density) were still selected and kriged on both original as well as log-transformed data for comparison purposes only. This confirmed that OK using original data was the more reliable approach.

9.3. Geological modelling

Geological modelling to subdivide the tailings dam into zones of similar characteristics was attempted, but no tangible zones could be established. Three parameters that are relevant to grade continuity were assessed: tailings material (lithology), grade content (4E) and particle size distribution (PSD).

A very broad and general trend was noticed in the dam. The centre of the dam is generally very fine grained, with high clayey content. In general, it is also where the grade is at its highest. It was also discovered that this trend, however, does not extend the full profile of the dam, that is, from top to bottom or across all boreholes in and around the centre. Some portions or part of boreholes in that area were found to contain considerable amount of relatively poor grade or coarser material.

Another trend identified was the bottom-most portions of boreholes throughout the dam are often comprised of very fine grained material with high clayey content, albeit in varying thicknesses, with fairly high grade content. The thickness of the finer and rich bottom part is slightly thin on southern-most portion of the dam, but progressively gets thicker towards the north, with the thickest and richest zone realised on the North-western corner of the dam (Figure 61).

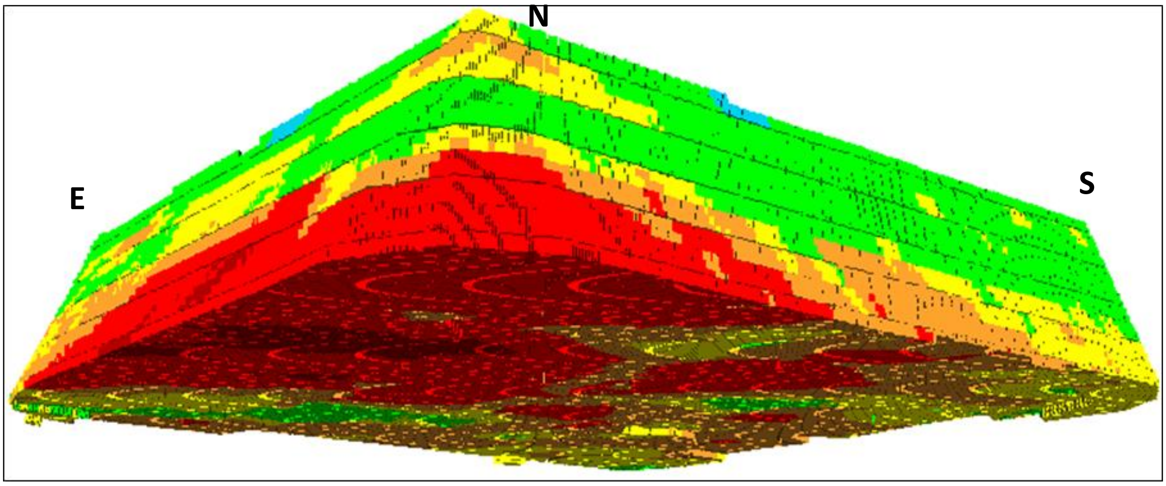

Figure 61: Bottom view of the north-western corner of the model.

Boreholes drilled around the perimeter of the dam were found to be generally coarser in comparison to the centre, with average to low grade content. Furthermore, sporadic lenses with high grade are also encountered in these boreholes.

The trends from any or combination of the variables assessed were found to be inconsistent and very discontinuous to warrant any meaningful subdivision of geological zones, hence only a single zone (envelope) was modelled to cover the entire tailings dam.

Intermittent lithological lenses as well as alternating high and low grade areas are attributed to mechanically deposition process. It is a process which is not consistently continuous, which further depends on the type of ore being milled as well as the capacity or volume being pumped through the system. It often takes years and countless pumping cycles to construct a tailings dam of a considerable size. These pump-stop-pump cycles and tailings settlement processes are what give rise to alternating and inconsistent layering encountered in the dam.

9.4. Mineralisation continuity

The Case Study tailings resource surprisingly revealed sufficient spatial correlation or reasonable mineralisation continuity to be evaluated by Kriging. All seven grade variables analysed have mineralisation continuity with ranges that varies between 330 m and 550 m. A 100m x 100m pattern was followed to conduct drilling on the Case Study tailings project, therefore these ranges (330m – 550m) realised correspond to between 3 and 5 borehole rows; as such search ellipse had enough sample data for proper estimation (Figure 62).

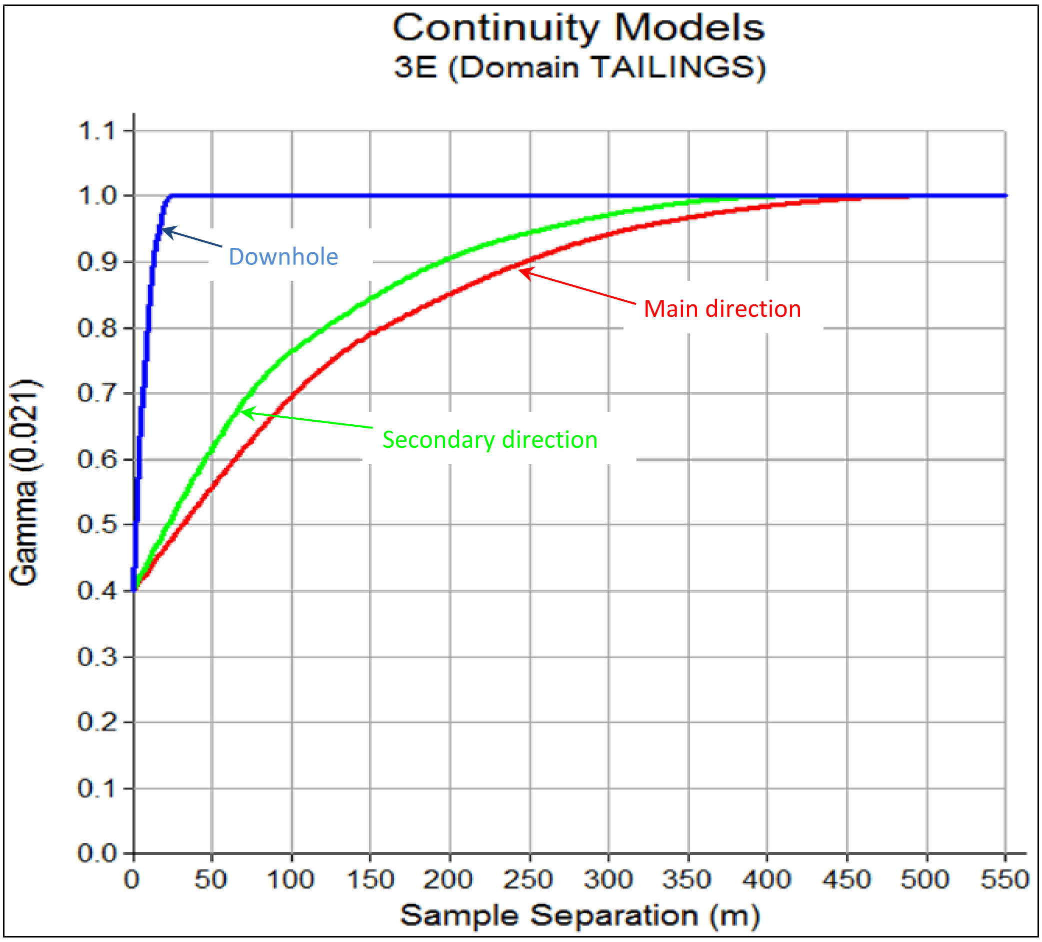

Figure 62: Typical variogram ranges (3E) in all 3 directions (blue, green & red)

All experimental semi-variograms calculated are anisotropic with varying nugget effects. The nugget variance was generated (modelled) from the down-hole or vertical direction, where sample information is over the shortest interval (1 m). Moreover, the nugget variance is concluded to be relatively high. It has been noted to average at about 0.4 of the standardised semi-variogram sill. This implies that it accounts for more than one third (~40%) of the total sill variance in most grade variables. It was further noted that the spatial variance calculated or realised on the first lag of 100 m [γ(100)] averaged 0.7. Therefore, between 0 and 100m, another 30% or a third (0.3) of the total sill is added to the nugget effect (0.4+0.3 = 0.7). As such, the variogram modelled on the actual experimental semi-variogram beyond the first 100m lag [γ(100)] accounts for the last third or 30-40% (0.3) of the total sill variance (1-0.7 = 0.3), see Figure 62.

The effect, on geology, of depositing tailings dam mechanically which rendered subdivision of the dam into geological zones impractical, is understood to be one of the reasons that led to relatively high nugget variances. Although sampling error was not quantified because it is beyond the scope of this research, the nature of sampling tailings as well as sample preparation at the laboratory could possibly have given rise to or contributed to some sampling error, which in turn could have contributed to high nugget variances.

The overall assertion from variography study of the Case Study tailings dam is that variogram models of all grade variables adequately reflect spatial correlations that exist, and are more than sufficient to produce a resource model.

9.5. Resource Model

The resource model of the Case Study tailings dam was subsequently generated based on the mineralisation continuity established. The Geostatistical estimation was performed using Ordinary Kriging in Datamine. The estimation process ran successfully and all seven variables were estimated into each cell of the 5m x 5m x 5m block model.

Parameters used or set to conduct spatial interpolation were tested through cross-validation process, which is an independent process incorporated into Datamine, where each original sample value is removed and estimated (kriged) by the parameters provided together with the remaining neighbouring samples. The difference between key parameters (e.g. mean) calculated from the two data sets (original data and estimated values) were analysed, to anticipate or give a prior indication into the reliability of the estimation of those areas that are not sampled, but would be interpolated. This process confirmed that estimation process input files were sufficiently optimised.

9.6. Resource Model Validation

Spatial interpolation results were accepted as a true reflection of the resources contained in the Case Study tailings dam, after undergoing the final rigorous validation processes. Validation of the resource model was carried out as follows:

Visual Validation: resource model against borehole strings (5 m composites)

The first validation step was a brief overview of the resource model grade profile against the known borehole data, in this case 5 m composited boreholes, on section cut across all drilled boreholes. Estimated grade in the cells adjacent to borehole traces were found to closely resemble grade profile in those boreholes and no difference could be spotted. These sections revealed that, as expected, all cells were within the variogram ranges and therefore estimation was accurately performed into all the cells. As such the model satisfied visual inspection and was passed to the next level of validation.

Statistical Validation

The second validation step was an in depth statistical validation. Means of the estimated resource model were compared against the associated means of the original sample data. The original 1m sample data was composited into 5 m long composites to standardise support size. The reliability of the resource model was analysed by assessing the difference in these averages on a global scale.

All the differences returned a negative value which implies estimation figures to be marginally higher than the original data. The highest difference in the means of the two data sets was a marginal 1.7% realised on 3E. Density and Au has the second and third highest differences, at 1.2% and 1.1% respectively. The rest of the variables have differences between their averages at less than 1%. The smallest difference was realised on Ni which returned a mere 0.2%. The statistical validation process clearly indicates a high confidence in the Ordinary Kriging estimation method of the untransformed raw data.

As pointed out previously, three grade variables were selected for log-transformed kriging. “ESTIMATE” menu in Datamine has the option of linear kriging or lognormal kriging when Ordinary Kriging method is selected. In normal kriging, the weights are applied to the sample grades, whereas for log-normal kriging the weights are applied to the logarithmic values of the grades and are then automatically back transformed by the software. Results of lognormal estimation of the selected variables are presented in Table 19.

Table 19: Lognormal summary statistics against resource model and borehole data.

VARIABLE	TYPE	NSAMPLES	MIN	MAX	RANGE	MEAN	% DIFF OF MEAN	VAR	STD DEV	STD ERR	SKEWNESS	KURTOSIS	LOGVAR	LOGESTMN
3E	Borehole	681	0.523	1.458	0.935	0.786	-1.668	0.01351	0.116	0.00445	2.0	6.55	0.018	0.786
	RES Model	635403	0.556	1.393	0.837	0.802		0.01196	0.109	0.00014	1.7	3.88	0.016	0.802
	RES Mod log	635403	0.703	1.865	1.163	1.088	-35.660	0.02183	0.148	0.00019	1.5	3.41	0.016	1.087
		Log mod vs borehole difference					-38.352							
CU	Borehole	681	0.005	0.018	0.012	0.008	-0.755	0.000003	0.002	0.00006	2.4	8.62	0.032	0.008
	RES Model	634608	0.006	0.016	0.010	0.008		0.000003	0.002	0.000002	2.0	5.28	0.032	0.008
	RES Mod log	634608	0.007	0.021	0.014	0.011	-34.097	0.000004	0.002	0.000003	1.9	5.07	0.029	0.011
		Log mod vs borehole difference					-35.703							
Density	Borehole	681	1.183	2.429	1.246	1.886	-1.141	0.04442	0.211	0.00808	0.3	-0.38	0.012	1.887
	RES Model	635403	1.301	2.377	1.075	1.901		0.04553	0.213	0.00027	-0.1	-0.82	0.013	1.901
	RES Mod log	635403	1.627	3.396	1.769	2.479	-30.418	0.082934	0.288	0.000361	0.1	-0.68	0.014	2.479
		Log mod vs borehole difference					-31.401							

Lognormal kriging significantly over-estimated the three selected variables by quite some margins, 31%, 36% and 38% for Density, Cu and 3E respectively. Therefore, Ordinary kriging of the raw data of the Case Study tailings dam returned the best possible estimates under the current algorithm and variogram models (modelled grade continuity).

QQ Plots and Histograms

The final rigorous validation of the resource model was performed through QQ Plots and histograms.

It is imperative to note that the QQ plots were technically generated from different support sizes, 5 m borehole composites and 5m x 5m x 5m block model. Furthermore, the change of support, as expected was evident through deviations from reference (45-degree) line in the lower and upper extreme values of most variables. However, the graphs were practically sufficient for the scope of the project. Therefore, distributions of estimated variables in the resource model were ultimately comparable with distribution from the 5 m borehole composites, to offer valuable insight into the two data sets, regardless of the technicality.

QQ plots convincingly confirm that the resource model reasonably reflects borehole data from which it was derived from and it is well within acceptable margins. Most of the quantile data points plot very tight to the reference line (Figure 63). Deviations from the reference lines worth mentioning are noted in the middle parts of the Density and Au QQ plots, which are the two variables with the confirmed distribution underlain by multiple populations. The rest of the variables plot approximately on line, with deviations only noticed from marginal points at both extreme ends of PGEs (3E, Pt and Pd) due to the change of support effect.

Deviations being observed from the extreme low and high grade values are manifestation of different (or partly incompatible) support sizes between resource model blocks 5m x 5m x 5m and strings of 5 m borehole composites. Therefore, more emphasis is given to the central part of the QQ plot for this validation purpose. QQ Plot validation confirms that resource model reasonably reflects the original sample data. In turn, this gives an assurance to the Ordinary Kriging method selected to conduct estimation.

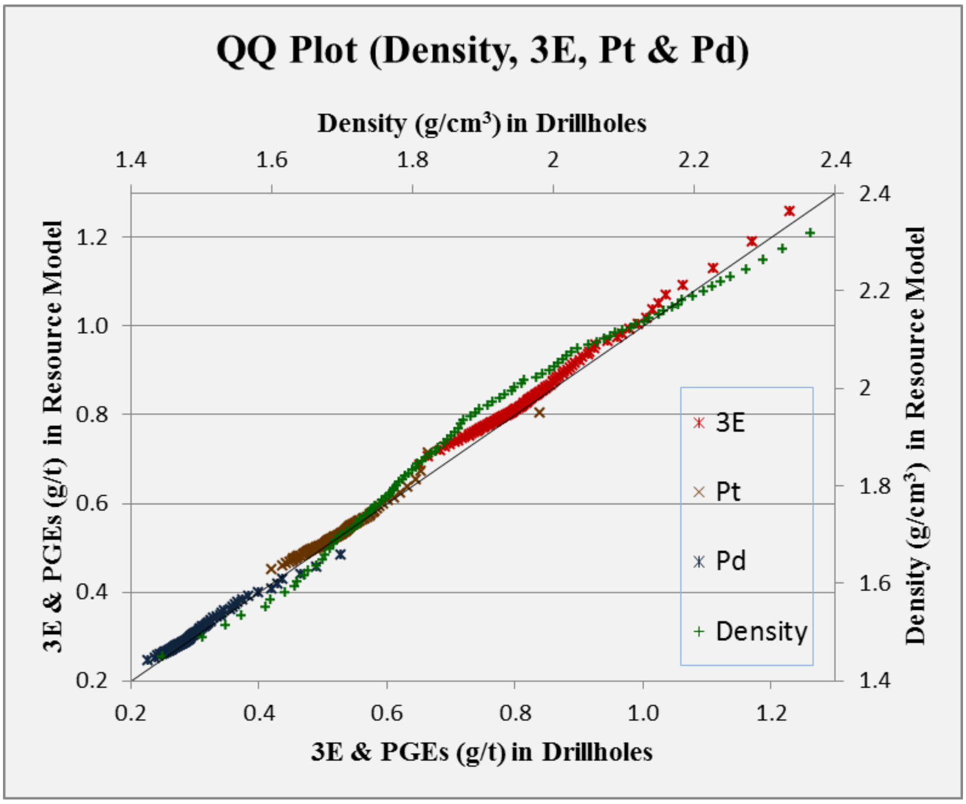

Figure 63(a): PGEs, 3E and Density QQ Plots.

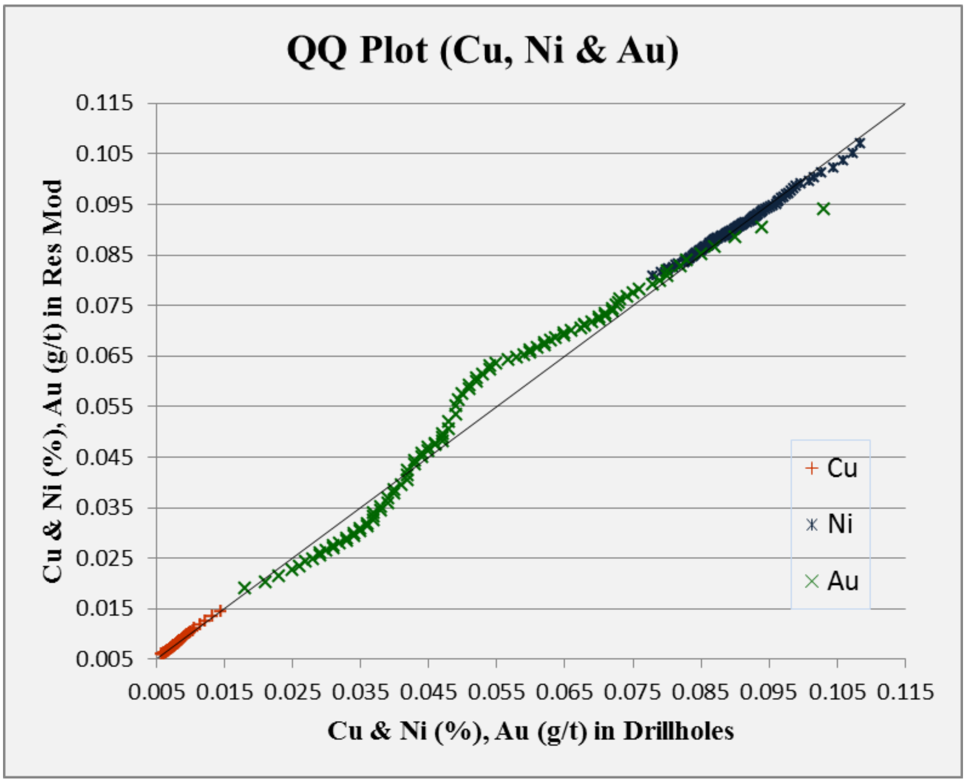

Figure 63(b): Base metals (Cu & Ni) and Gold (Au) QQ Plots.

The last validation point was the histograms to plot the distribution of the two data sets. There is a reasonable similarity in the distribution (spread) and behaviour of resource model and 5 m composited borehole data. Furthermore, the appearance of more than one population in variables such as Au and Density are clearly mapped out and magnified by the resource model data.

The Case Study tailings dam is therefore successfully estimated by Ordinary Kriging, regardless of its formation as a man-made ore body. Validation processes gave a compelling assurance to the quality of the resource model produced. This model, therefore, is considered to be a true representation of sampling data acquired, as well as being an acceptable estimation model and good reflection of mineral resources hosted in the dam, which can confidently be used for economic evaluation or valuation of the dam.

CHAPTER TEN:

10.1. Conclusion and Recommendations

The main aim of the project, which was to prove that geostatistical technique, such as Ordinary Kriging, can be applied in its current form in estimating tailings resources, was achieved. The Case Study tailings dam was successfully estimated through Kriging, a geostatistical technique that has been developed and thoroughly improved to address challenges experienced in estimating *in situ* geological ore bodies.

Sonic drilling was proven to be an appropriate method of drilling and sampling tailings to gather sample information. Some improvements can still be made on the method to address two shortfalls that have the possibility to introduce or increase sampling error, which in turn can have a bearing on nugget effect. The improvements are:

- Reducing the diameter of the bit and core barrel, thereby reducing the sample sizes. This in turn will improve sample handling, homogenisation and simplified splitting of representative lot at the laboratory.
- Drilling can also be carried out in conjunction with casing the entire length of the borehole, to alleviate risk of incorporating contamination into samples or part of samples being discarded as contamination, particularly where *in situ* compaction is highly loosened up during sample recovery.

Furthermore, an opportunity exists to conduct sampling error quantification (exercise). It is an area that can be undertaken to understand sample collection and analysis of tailings resource and the relationship of that sampling and analysis to nugget effect.

The information gathered from the tailings dam and results from the laboratory, were subjected to further data analysis typically undertaken in mineral evaluation.

Statistical analyses of the data collected from the dam revealed the distribution of almost all variables to be non-symmetrical and slightly positive skewed. Moreover, further look at the rest of statistical parameters revealed the margin of the positive skewness to be insignificant. Differences between original data means and log-normal estimates in all variables were negligible, thus no improvement on accuracy of the estimation was expected to be brought about by log-transformation of the data. Statistical analyses, therefore, lead to a conclusion that raw data would be kriged for primary estimation of all grade variables.

Nevertheless, 3 grade variables (3E, Cu and Density) were selected to be kriged twice, firstly linear kriging of raw data and then lognormal kriging. By doing this an opportunity was created to quantify the actual difference between the two procedures, which in turn established magnitude of the effect of the skewness on the estimation of the raw data.

In the end, Ordinary Kriging of raw data produced estimates that were closest to original sample data. Grade variables were only overestimated by values varying between 0.2 and 1.7%. Lognormal estimates in the other hand were not successful. It seriously overestimated the three selected grade variables by margins between 31 and 38%. For example, Density was overestimated from expected and realistic tailings *in situ* 1.9 g/cm^3, to a solid rock equivalent of 2.5 g/cm^3. Further investigation of log-normal behaviour of the Case Study tailings dam is one area of this research that can still be investigated further.

The geological modelling was only attempted from 1m composited boreholes. The trends from any or combination of the variables assessed were found to be inconsistent and very discontinuous to warrant any meaningful subdivision of geological zones, as such, only a single zone (envelope) was modelled to cover the entire tailings dam. However, the estimated grade model is indicating high grade (PGE) zones that are slightly larger and better defined. Therefore, it is recommended that geological modelling should be attempted using 5m composited boreholes.

Sufficient or reasonable mineralisation continuity was found to exist in the dam, and it is characterised by high nugget effect. Also, the mineralisation continuity has got preferred orientation, therefore, it is anisotropic. Variogram models for 3E, Pt, Pd, Ni and Density are nested structures comprised of three spherical models. Au and Cu are also nested structures but are comprised of two spherical models. Breakdown of the standardised (total) sill variance, as modelled by the variogram, is noted as follows:

- Nugget variance contribute about30 to 40%;
- 0-100m contribute a further 30 to 40%;
- and the last 30% exist between 100 to 500 m.

It is evident that 100 m lag distance corresponding to the 100m x 100m drilling pattern produced variogram models that are sufficient for resource estimation for all grade variables. It is still recommended that a drilling grid comprised of one row across the dam and another along should be drilled on 50m x 50m spacing, to reveal the underlying variogram structures that might exist between 0 m and 100 m, which account for 30% to 40% of total sill variance – or 60% to 70% when including the nugget effect.

A block model comprising 5m x 5m x 5m cells was generated and used for Case Study grade estimation, with the initial thought being to standardize the support size as close as possible to the 5 m borehole composites used in the validation process. In hindsight, the drill spacing of 100m x 100m should also have been considered in the selection of the block model size. Therefore, an estimation block size of 50m x 50m x 5m, which is half the horizontal distance between drillholes, would have been more appropriate, as it would have taken cognisance of the missing horizontal spatial correlation information between 0 m and 100 m.

Nevertheless, the rigorous and intensive validation process undertaken confirmed that the resource model produced is acceptable, reliable and a good representation of resources contained in the dam. Therefore, the Case Study tailings dam was successfully estimated with Ordinary Kriging (geostatistical technique), and the resource model can be used for any initial economic decision-making regarding the exploitation of the tailings dam.

One of the key questions (Section 1.5.5) about which estimate is more accurate and reliable between metallurgical accounted grades and geostatistical estimated grades on tailings dams in general, could not be answered for the Case Study tailings dam; as explained below.

The Case Study tailings dam is the oldest in the complex; therefore complete and continuous metallurgical records of its grade and tonnages are unavailable. As such, a figure of 1.1 g/t for the 4E (Pt+Pd+Rh+Au) grade was provided by the metallurgical team prior to tailings drilling commenced. The value is not an official metallurgical accounted number and is only a global figure because there are no auditable (verifiable) records to establish accurate grade and tonnages of the dam from the metallurgical accounting point of view.

Furthermore, comparison between the metallurgical 4E estimate and the Kriged 3E estimate is deemed impossible for the Case Study tailings dam, due to limited Rhodium (Rh) information currently available. Rh was only analysed on samples that have a 3E (Pt+Pd+Au) grade greater than 1.5 g/t. In this case, only 11samples out of 3225 (0.34%) triggered the Rh analysis threshold. Therefore, Rh was excluded from the Case Study resulting in the 3E estimation only. Consequently no sensible comparison between the two estimates is possible; with metallurgical estimate being a 4E and kriged estimate a 3E.

To include Rh in the model, it is recommended that at least 15-20% of the total samples should be analysed for Rh content. Common regression techniques could then be used to estimate Rh grade, based on sufficient and acceptable data quantity from its linear relationship with Pt/Pd grades for all the remaining samples not analysed for Rh. Subsequently, the regressed Rh can then be included in the kriging exercise to produce a 4E g/t kriged estimate, which could be used to establish comparison with metallurgical estimate. In the Case Study tailings dam 15-20% would have been 483 to 645 samples of the 3225 samples, instead of the current 11 samples (0.34%) that were assayed for Rh grade.

Geostatistical techniques, primarily developed to estimate mineral resources in an *in situ* ore body, can be applied to a tailings resource without modification of the standard kriging methodology. It is the best possible estimation technique that can derive practical, auditable and traceable results for economic evaluation of the valuable minerals that still exist in tailings, which were not recovered during ore treatment at the processing plant.

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APPENDIX

Appendix A: Variogram Models

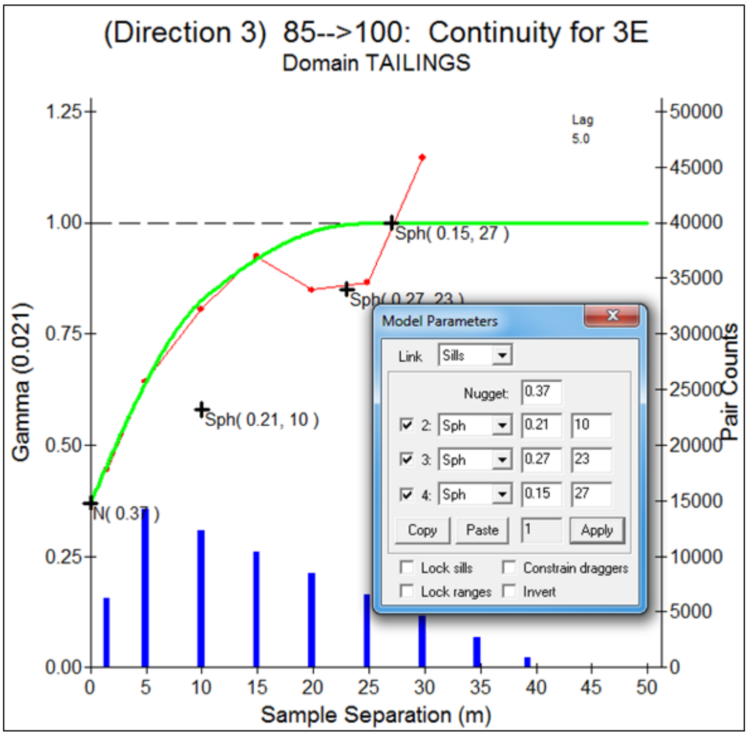

Figure A1: 3E downhole variogram model

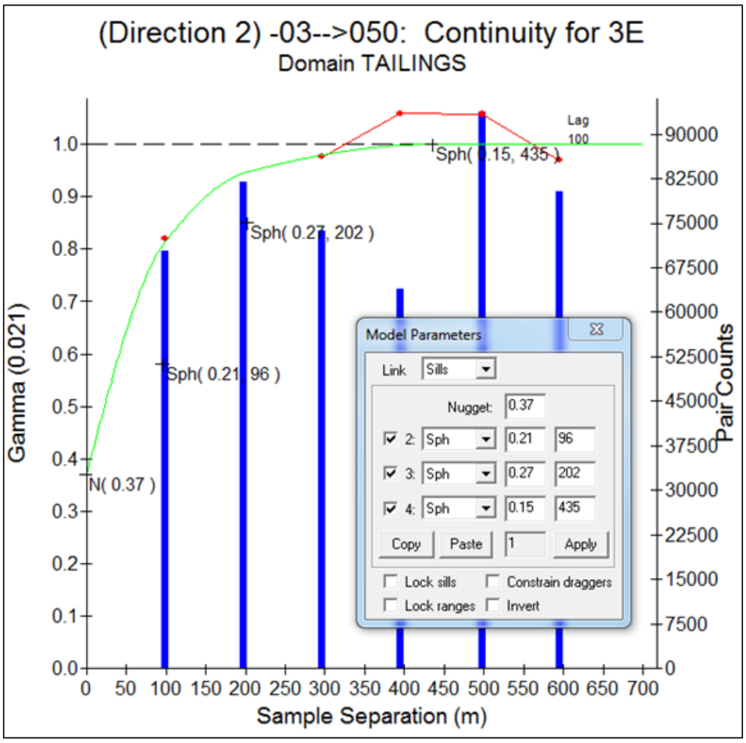

Figure A2: 3E variogram model along secondary direction

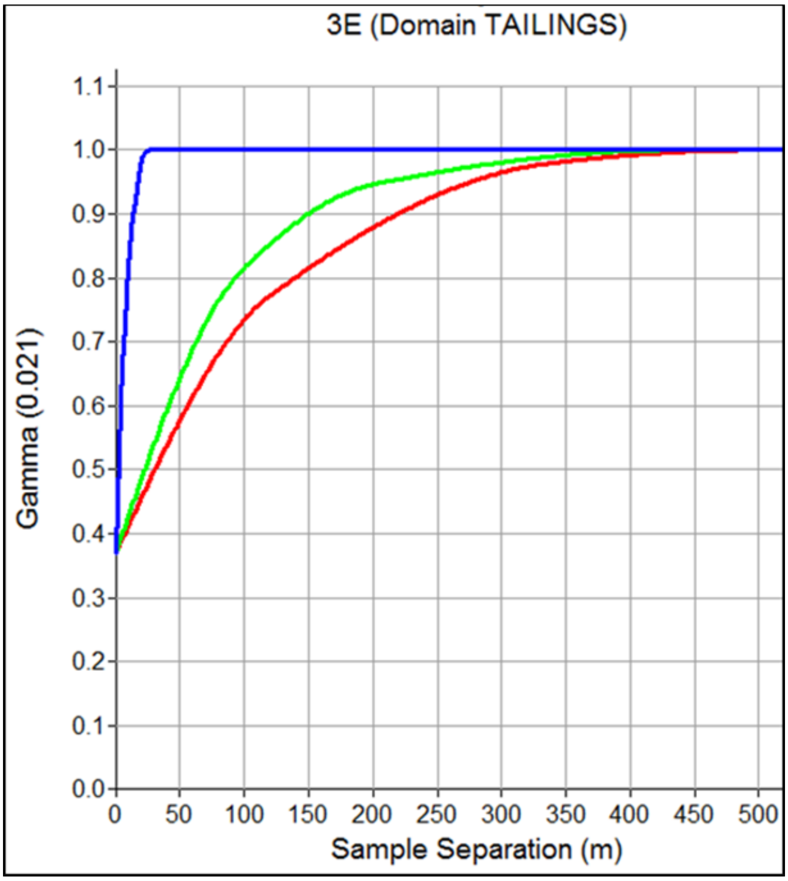

Figure A3: 3E variogram models (all three directions – blue, green & red)

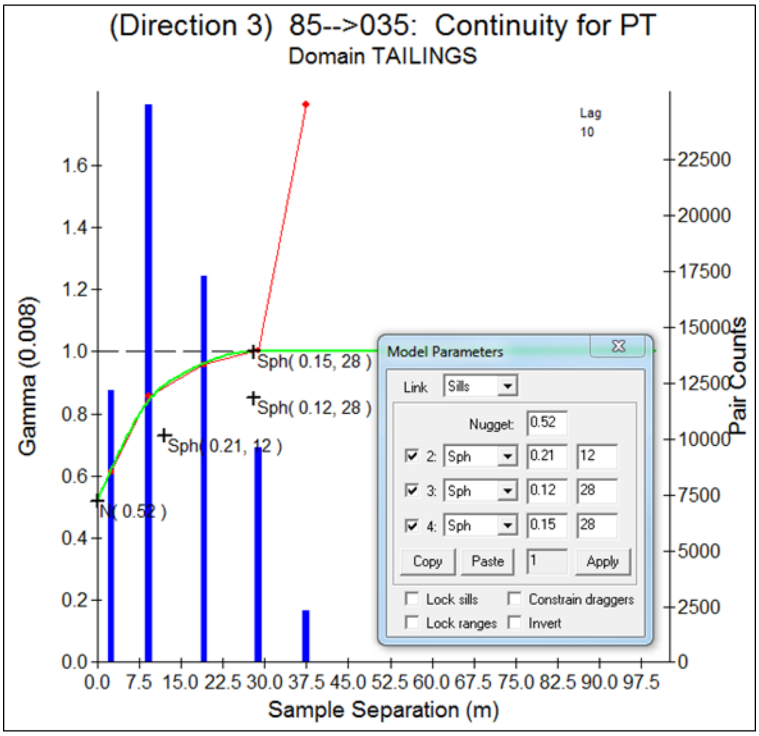

Figure A4: Pt downhole variogram model

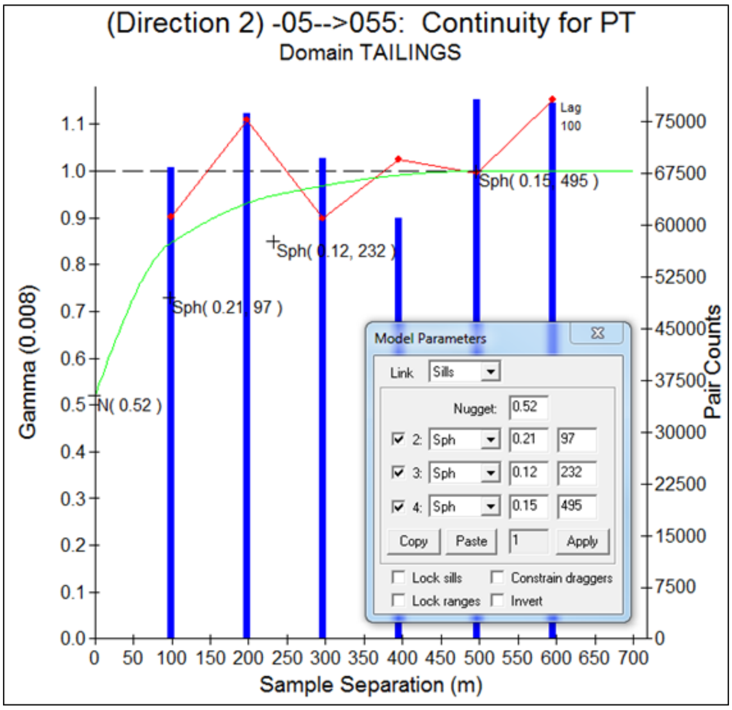

Figure A5: Pt variogram model along secondary direction

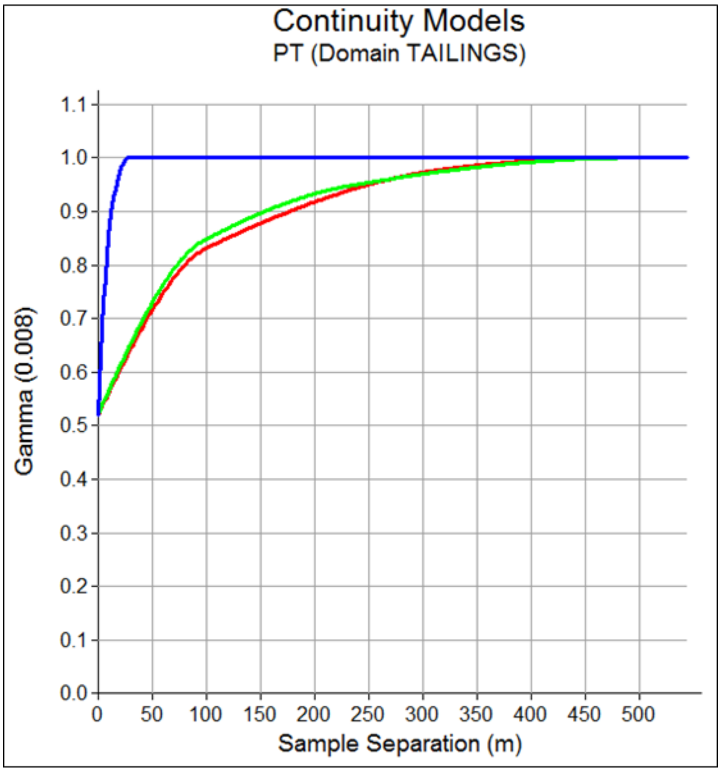

Figure A6: Pt variogram models (all three directions – blue, green & red)

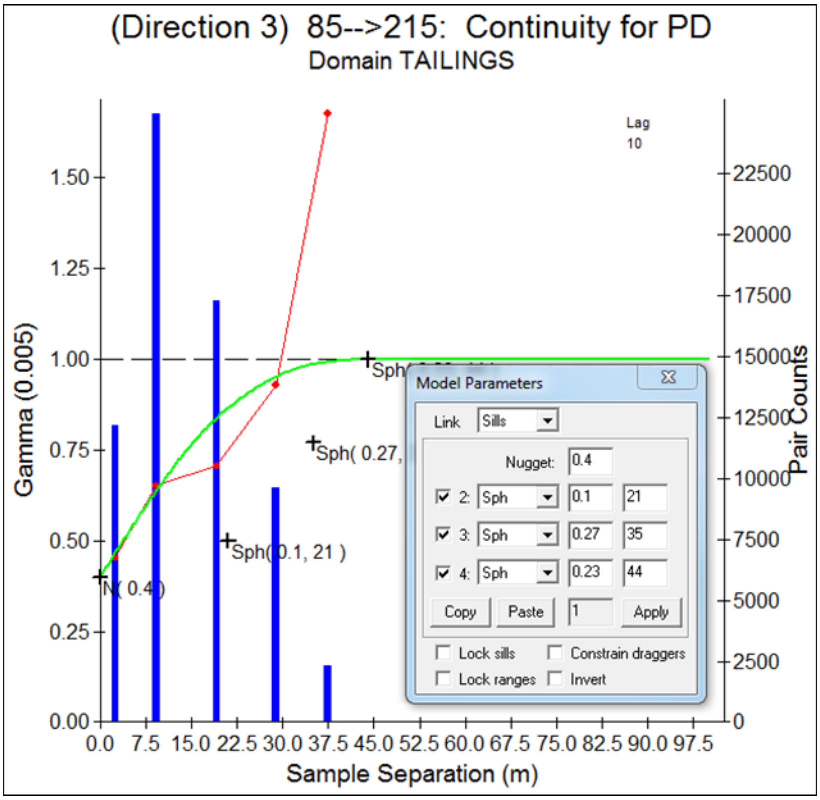

Figure A7: Pd downhole variogram model

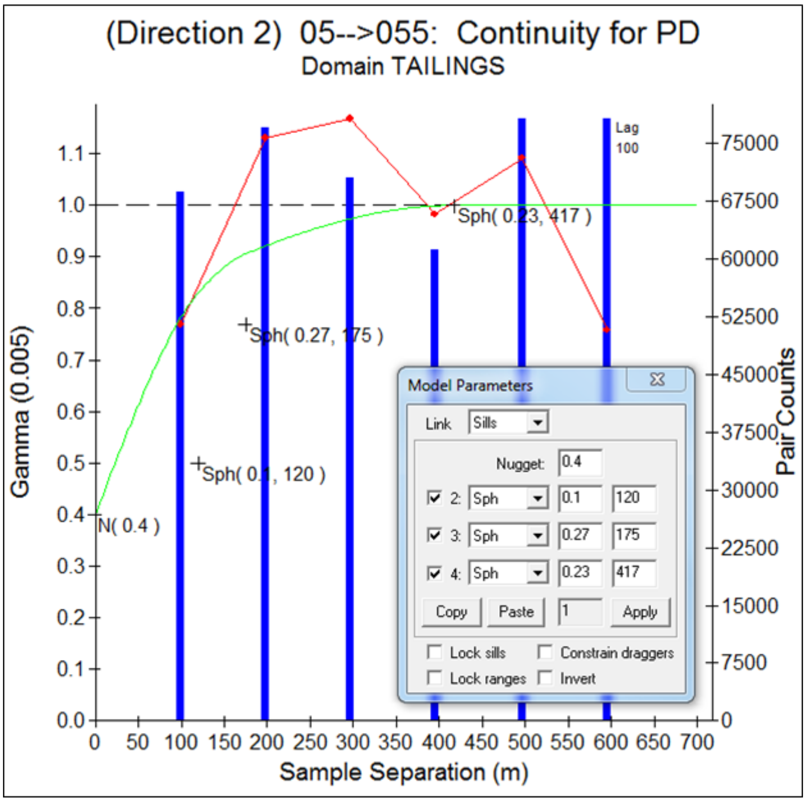

Figure A8: Pd variogram model along secondary direction

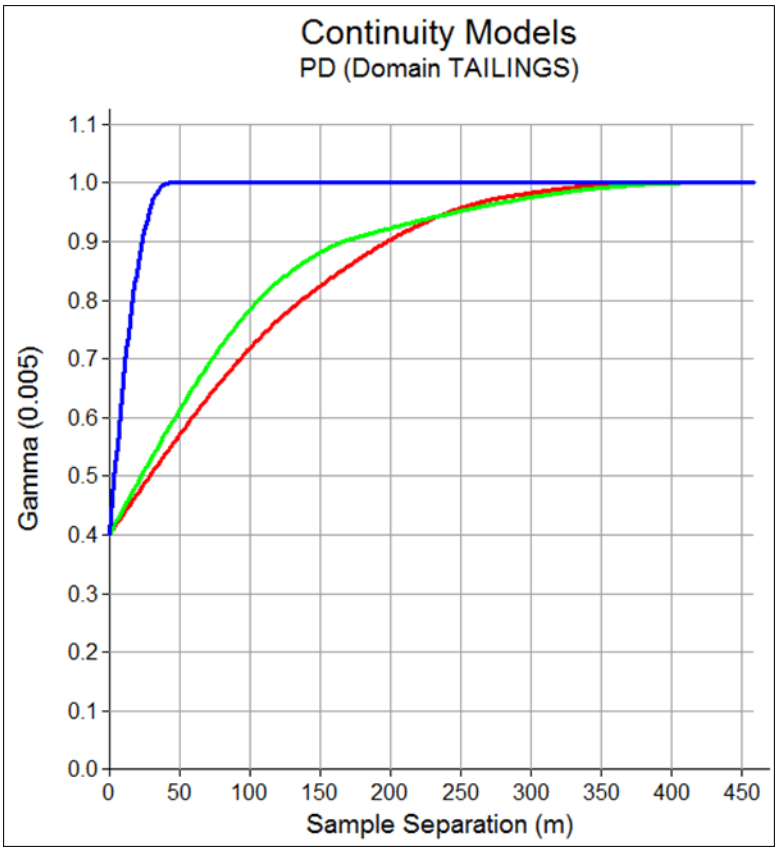

Figure A9: Pd variogram models (all three directions – blue, green & red)

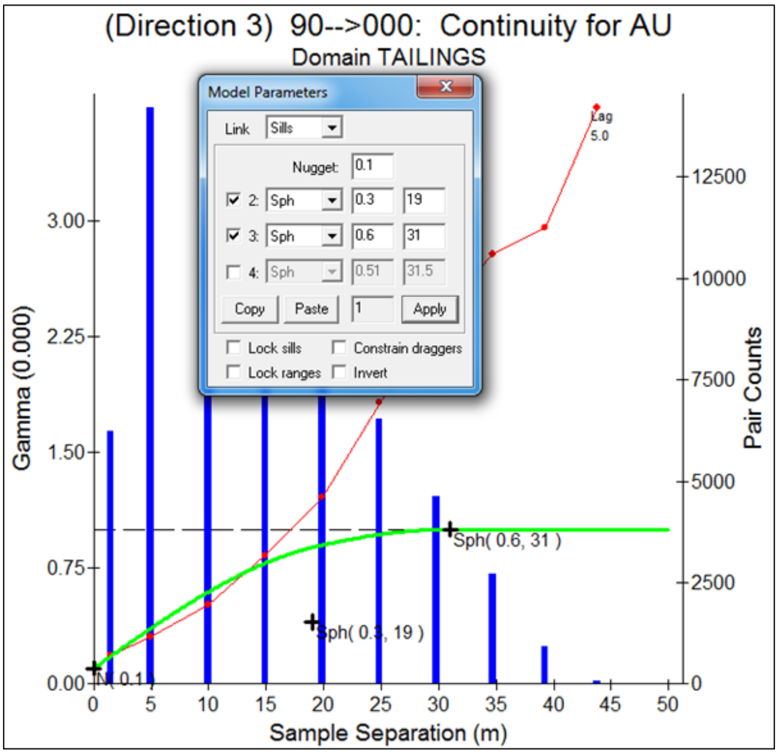

Figure A10: Au downhole variogram model

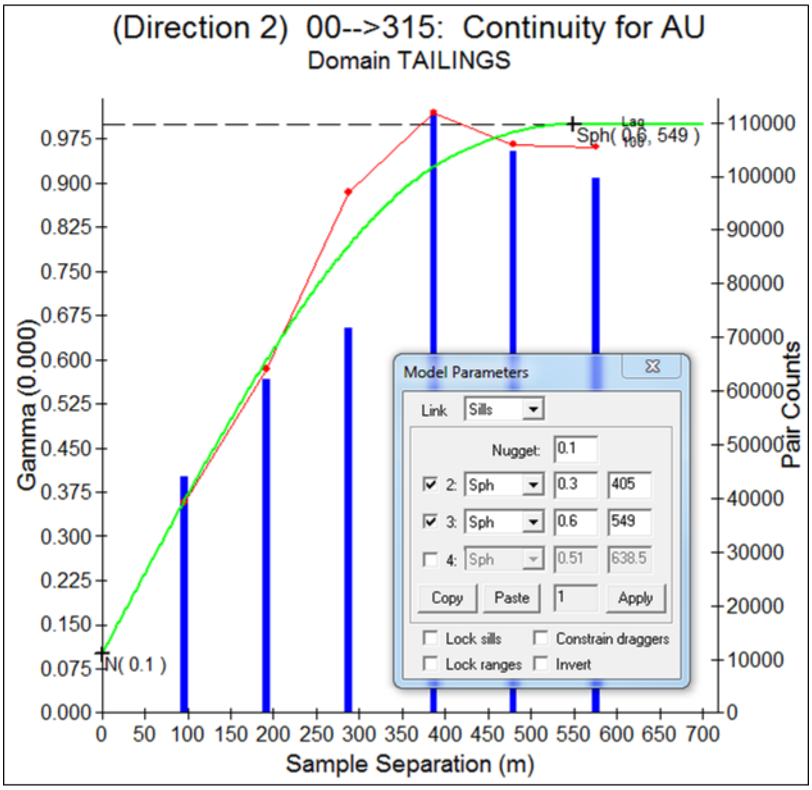

Figure A11: Au variogram model along secondary direction

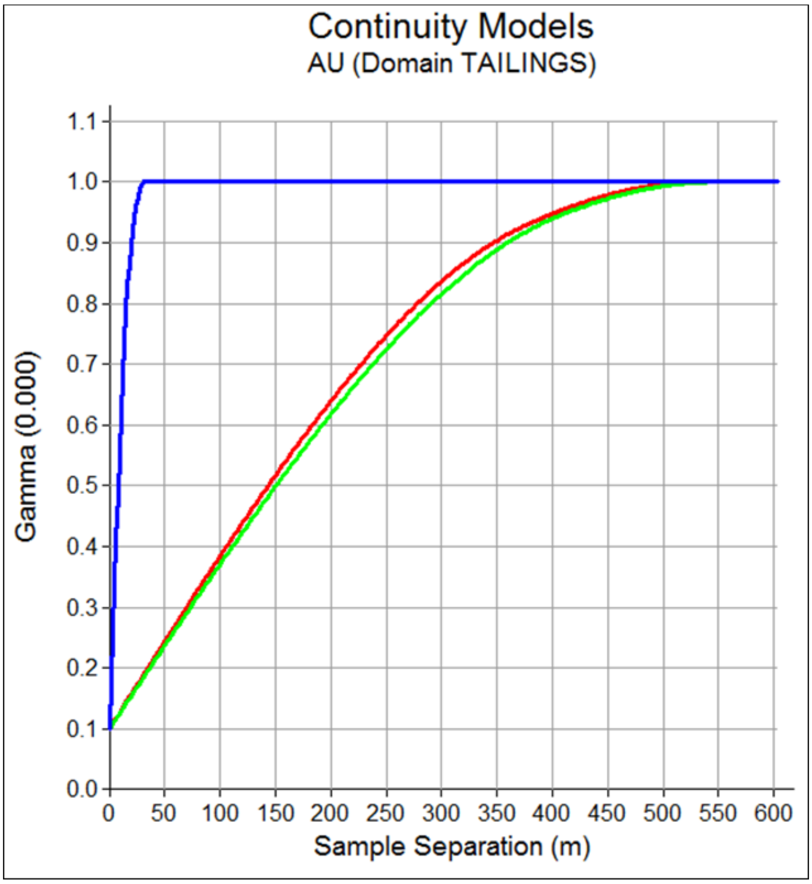

Figure A12: Au variogram models (all three directions – blue, green & red)

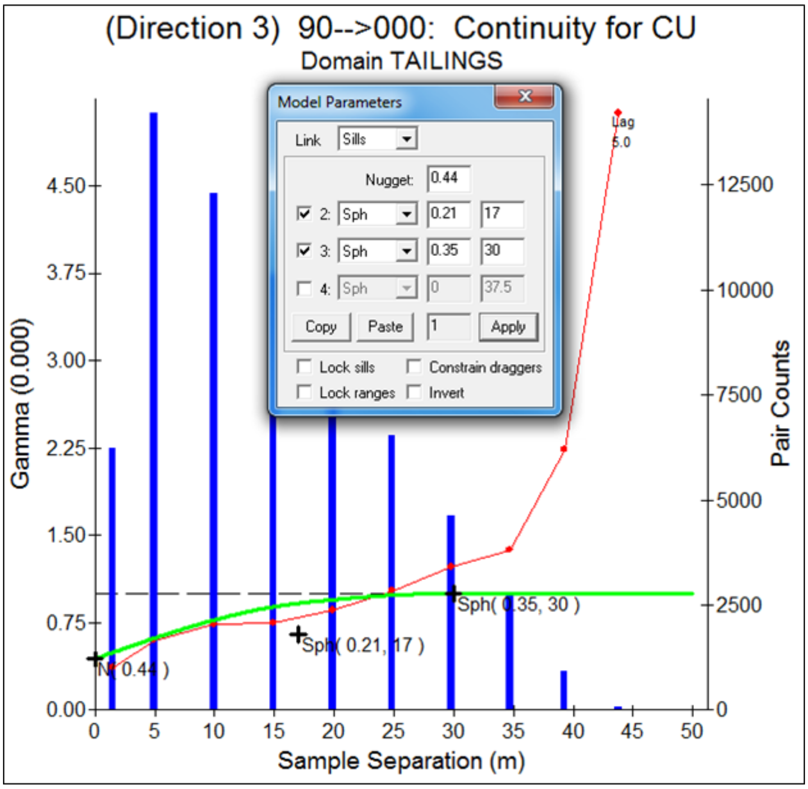

Figure A13: Cu downhole variogram model

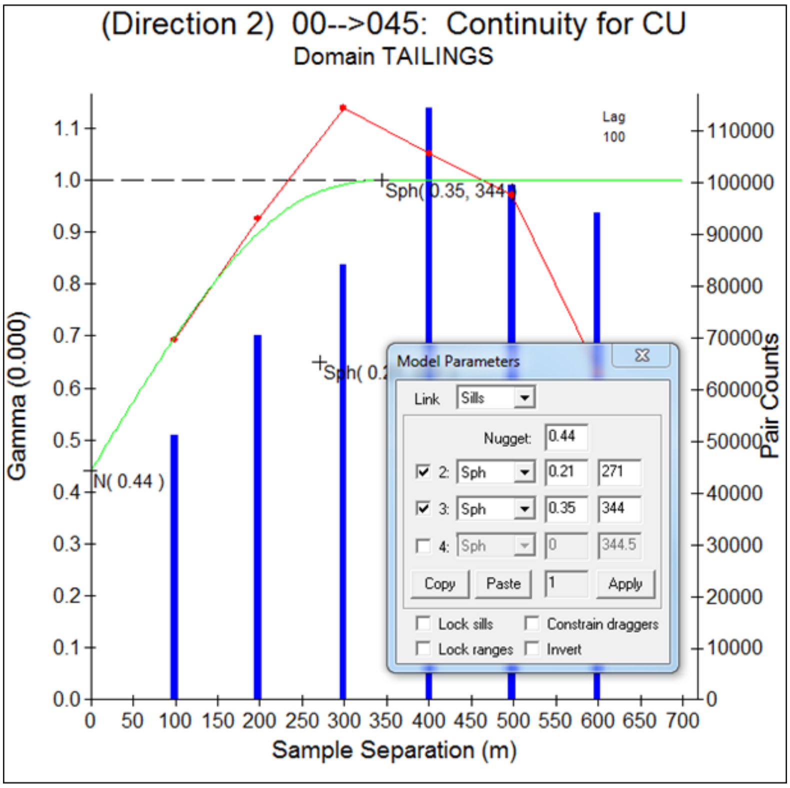

Figure A14: Cu variogram model along secondary direction

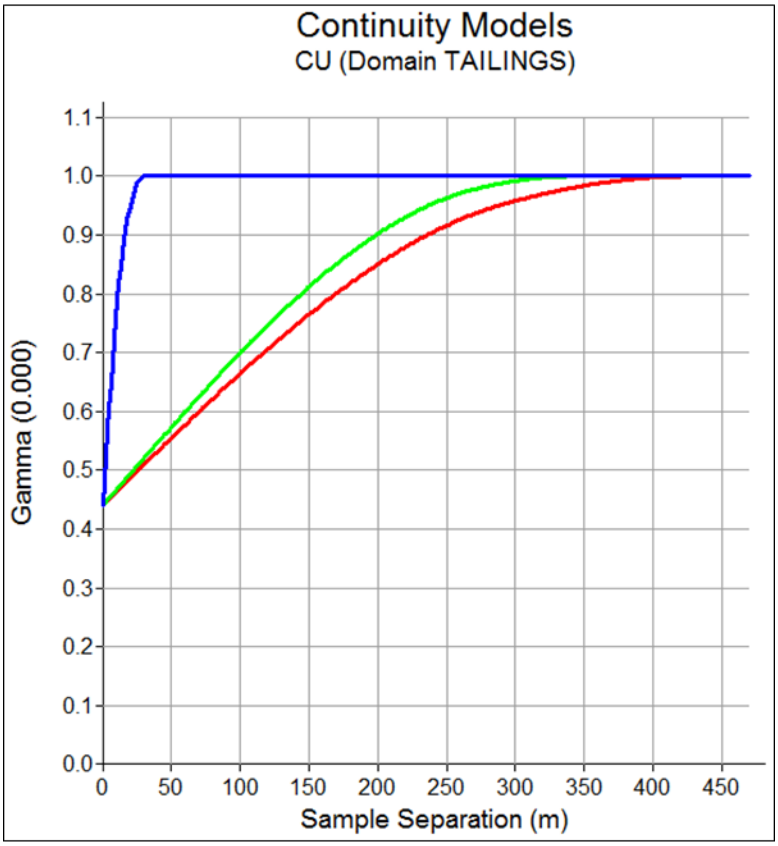

Figure A15: Cu variogram models (all three directions – blue, green & red)

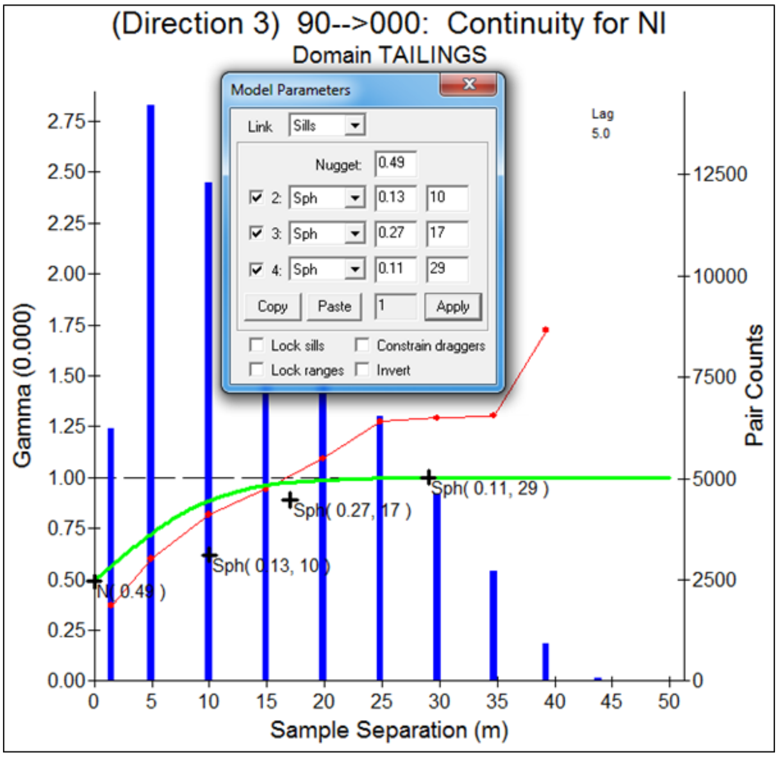

Figure A16: Ni downhole variogram model

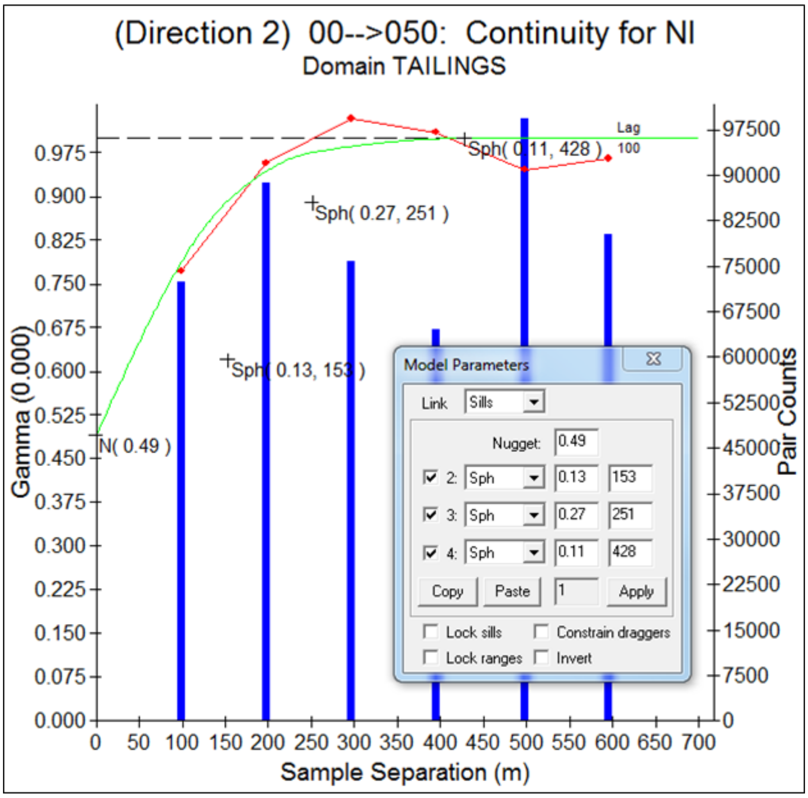

Figure A17: Ni variogram model along secondary direction

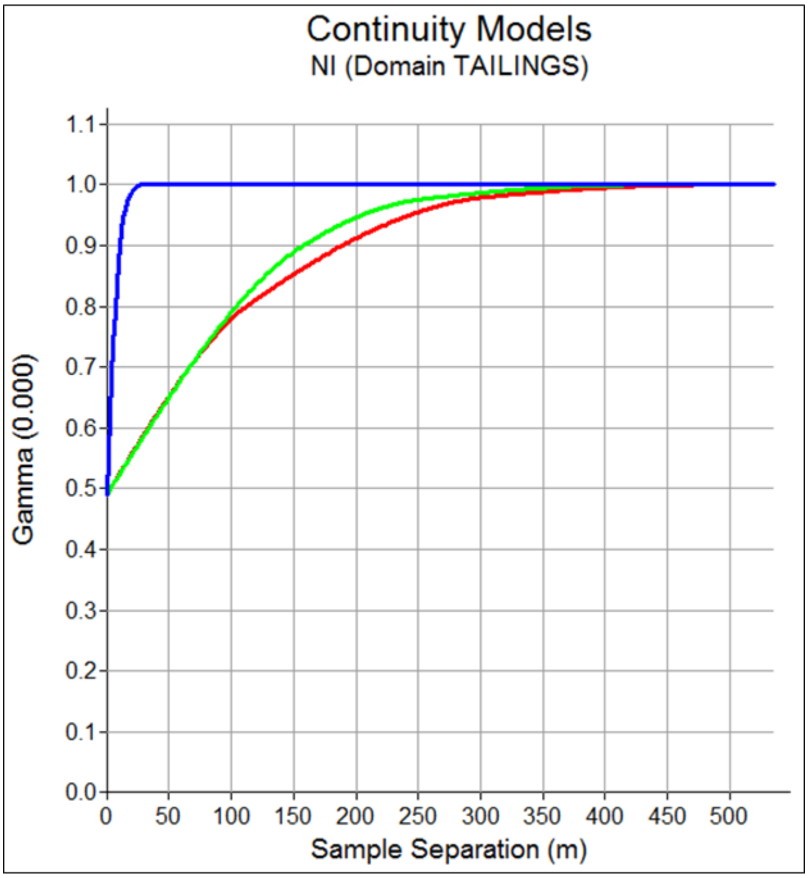

Figure A18: Ni variogram models (all three directions – blue, green & red)

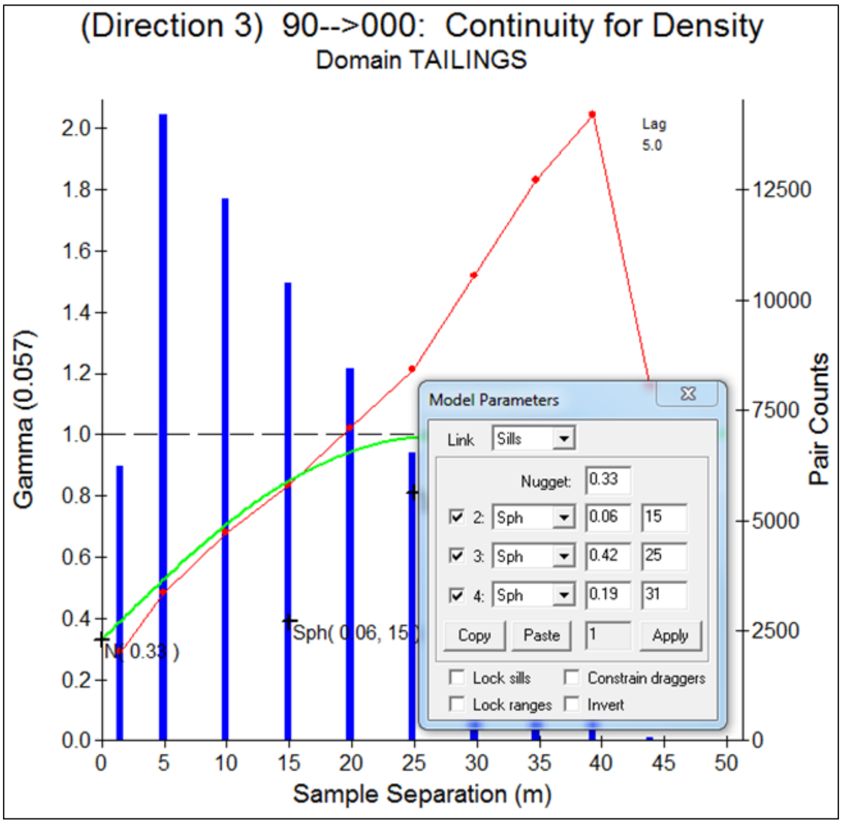

Figure A19: Density downhole variogram model

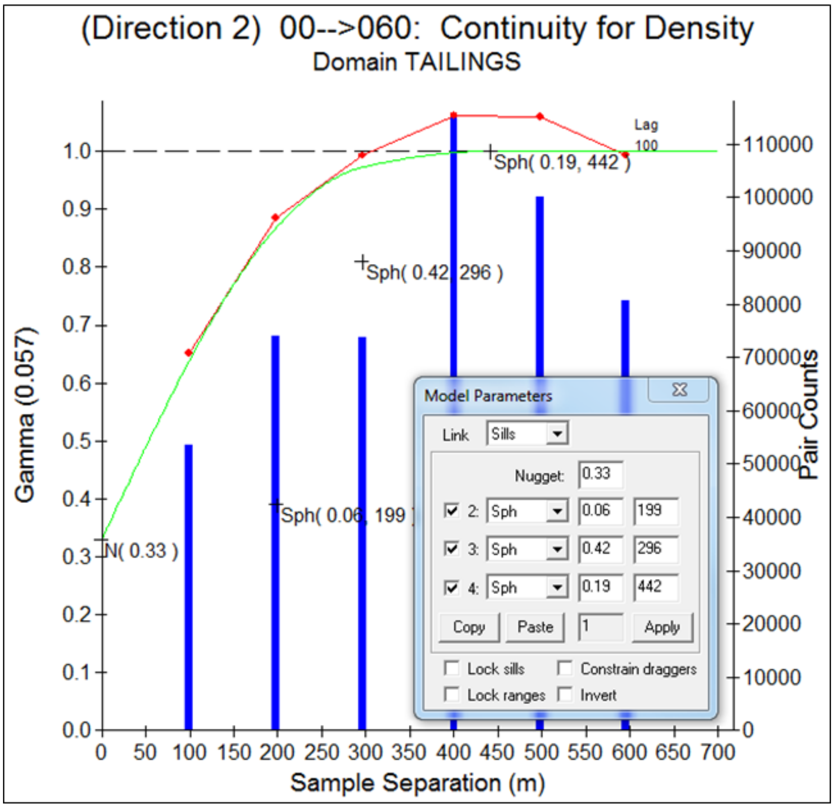

Figure A20: Density variogram model along secondary direction

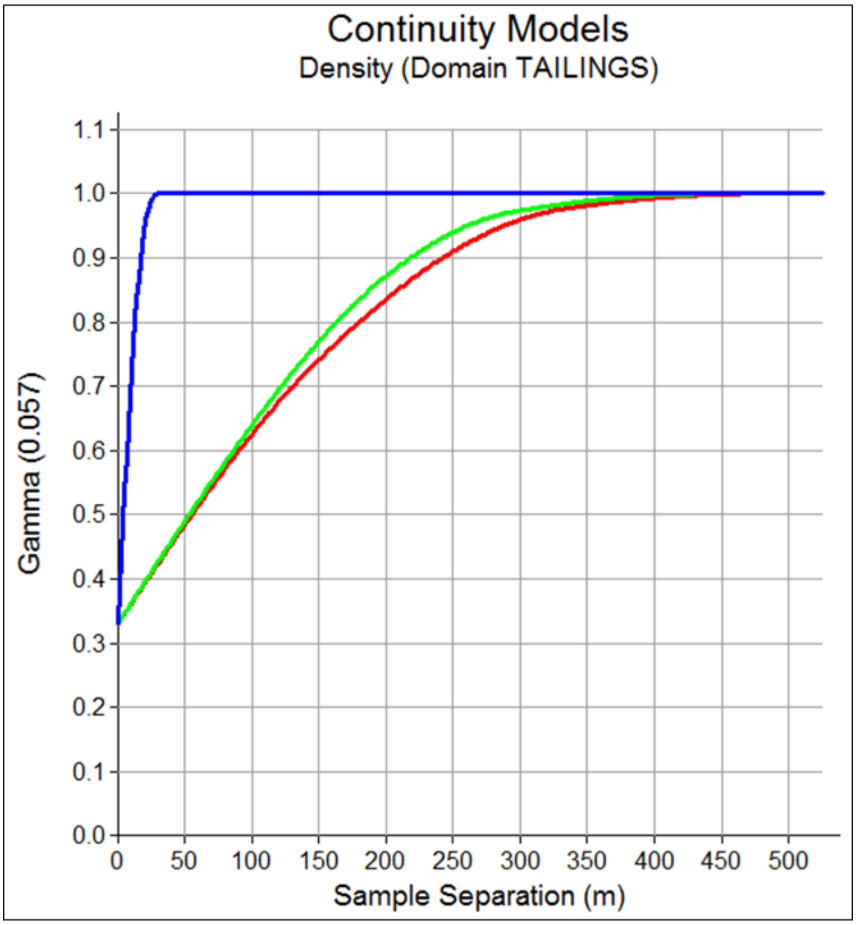

Figure A21: Density variogram models (all three directions – blue, green & red)