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THE INVESTIGATION OF AN OPTIMAL SAMPLING SCHEME FOR STYLDRIFT I MINE BY APPLYING GEOSTATISTICAL, PRACTICAL AND ECONOMIC PARAMETERS

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A research report submitted to the Faculty of Engineering and the Built Environment, University of the Witwatersrand, Johannesburg, in partial fulfilment of the requirements for the degree of Masters of Science in Mining Engineering (MSc 50/50).

2021

DECLARATION

I confirm that this research submitted for assessment is my own unaided work. The research covers work undertaken at Styldrift I Mine of Royal Bafokeng Platinum. It is being submitted in partial fulfilment of the requirements for the Degree of Master of Science in Mining Engineering at the University of Witwatersrand, Johannesburg in the year 2021. It has not been submitted for any degree or examination to any other University. I am aware that using work of another without their permission and/or without acknowledging the original source is wrong and constitutes plagiarism. I have followed the required conventions in referencing the thoughts and ideas of others. I understand that the University of the Witwatersrand may take disciplinary action against for plagiarism



	0201858N	**28/09/2021**
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ABSTRACT

The success of a new mining project is highly dependent on the evaluation of the grade estimate and reconciliation of the planned versus actual achieved grade, which are both realised by means of underground sampling. Styldrift I is a new mine of the Royal Bafokeng Platinum operations that warrants the investigation of an optimal sampling scheme. The Merensky Reef, which is currently being mined, is highly variable by nature and features a transition zone between regional facies types in the study locality. The research study applied an elementary geostatistical method to analyse and identify an optimal sample spacing option that is practical to implement. The estimation error of kriging variance is used as a measure of confidence to test various sampling patterns in a drill hole sampling analysis. Existing sample data was validated and quality checked using statistical methods to model the spatial continuity which is the key component to geostatistics. The regional facies further subdivided into local facies types was an important consideration in applying the geostatistical analysis correctly in terms of understanding the small scale variation. The two local facies types showed unique population characteristics and were therefore evaluated separately. Estimates were conducted and tested for six practical sample spacing options configured for the bord and pillar mining layout. Analysis of the kriging variance results was successful in selecting two optimal sampling options (8 m x 8 m and 16 m x 16 m); however the determinant consideration was an implicit cost benefit comparison that was used to select the optimal sampling scheme of 16 m x 16 m that met the technical, practical and financial requirements for Styldrift I.

ACKNOWLEDGMENTS

Thank you to Royal Bafokeng Platinum for providing financial assistance towards this qualification as well as the opportunity to conduct the research study at Styldrift I Mine. Sincere gratitude goes to Prof. Christina Dohm for all her valuable criticism, suggestions, guidance, insight and patience in the research project. Thank you to my colleagues Karin Greyling, Khangwelo Nemafhohoni and the Styldrift I Mine sampling team for their selfless assistance in gathering information and providing knowledge from their technical expertise. Appreciation goes to Jaco Vermeulen for his valued leadership. Special thanks to my sister, Yusavia Naidoo for her support, encouragement and proof reading. Finally to my husband, Vinesh, whose love and patience is endless.

DEDICATION

To my daughters

Ahalya and Svara

For whom I work to provide, inspire and empower

To my late father in law

Neelan Padiachy

My pillar of strength who has now become my guiding light

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List of Symbols

Billion Years	Ga
Centimetres	cm
Coefficient of Variation	CoV
Correlation Coefficient	r
Distance and direction vector	h
Estimate at location v	Z*(v)
Estimation variance	σ_K^2
Footwall	FW
Gold	Au
Grams	g
Grams per tonne	g/t
Greater than	>
Kilometres	km
Less than	<
Less than or equal to	≤
Maximum	Max
Mean	μ
Measured value of sample	x_i
Metres	m
Millimetres	mm
Minimum	Min
Number of pairs	N(h)
Number of samples	n

Palladium	Pd
Percent	%
Platinum	Pt
Platinum Group Elements	PGE
Rhodium	Rh
Sample	i
Semi-variogram	γ
Square kilometres	km^2
Square metres	m^2
Standard Deviation	σ
Sum of platinum, palladium, rhodium and gold assay values	4E
Regionalised stationary variable values	$Z(x_i)$
Variance	σ^2
Weights	λ_i

1. INTRODUCTION

1.1. Research Background and Setting

1.1.1 The Bushveld Igneous Complex Geological Setting

The Bushveld Igneous Complex (BIC) is known as the world's largest and oldest mafic layered intrusion which covers an area of approximately 65 000 km^2. The Bushveld Igneous Complex dominates the world's production of platinum as it hosts over 80% of the world's deposits of platinum group elements (Cawthorn, 1999). This igneous intrusion consists of three major exposed limbs, as seen in Figure 1, namely the Western, Eastern and Northern limbs (Eales, 2001).

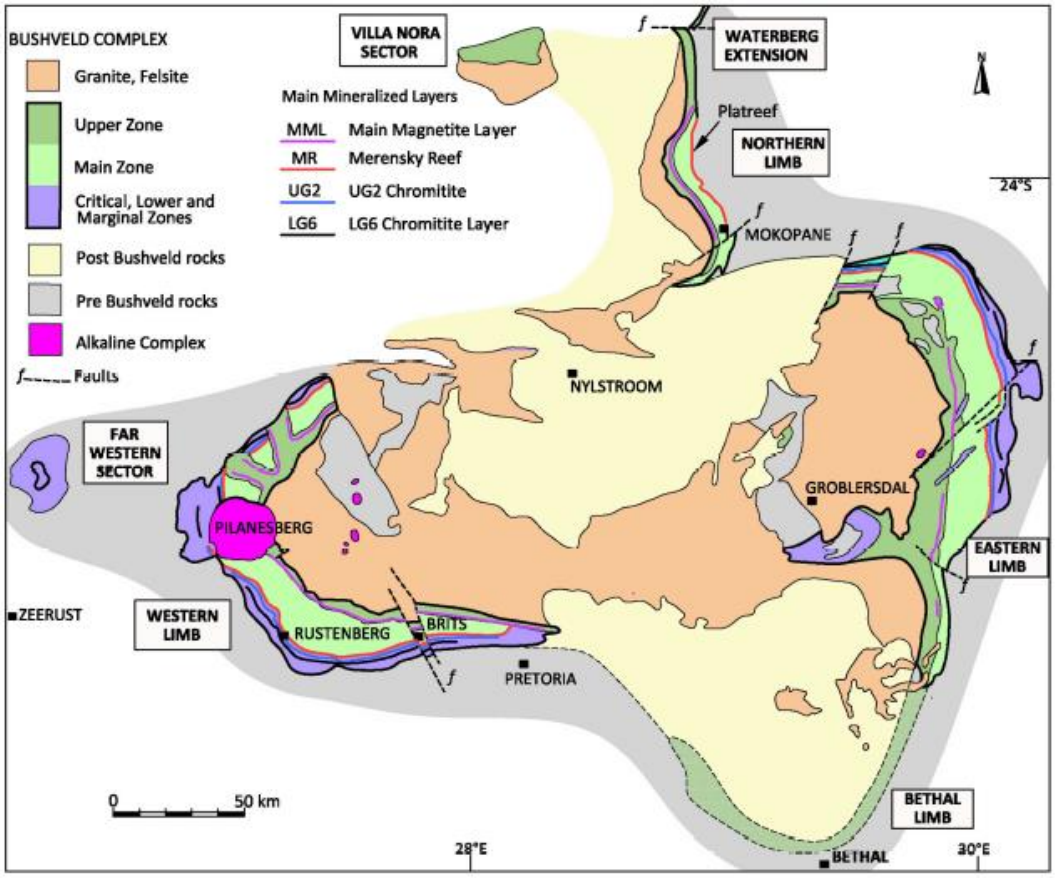

Figure 1. Three-dimensional illustration of the Bushveld Igneous Complex (Viljoen., 2016).

The Rustenburg Layered suite which is the mafic layered section of the BIC is subdivided into five main stratigraphic units which are the Marginal Zone, Lower Zone, The Critical Zone, the Main Zone and the Upper Zone (Viljoen,

2016). The Critical Zone, which is of major economic importance is subdivided into the Lower Critical Zone and Upper Critical Zone and hosts the Platinum Group Elements (PGE) bearing Merensky and UG Reefs as illustrated in Figure 1.

1.1.2 **Locality**

This research study was conducted at the Styldrift I Mine which is part of the Royal Bafokeng Platinum (RBPlat) mining operations. It is located 30 km north-west of the town of Rustenburg in the North West Province of the Republic of South Africa as seen in Figure 2.

RBPlat operations are situated on the western limb of the Bushveld Igneous Complex which is featured in the image of Figure 2. Styldrift abuts against the southern limit of the Pilanesberg Complex, which is the remnant of an alkaline volcanic plug, that intruded into the Bushveld Igneous Complex about 1,250 million years ago (Eales, 2001). Immediately west is a range of low-lying quartzite hills of the underlying Transvaal sequence. The Rustenburg Fault can be mapped through these hills, approximately 5 km from the mine (Lionnet and Lomberg, 2006).



Figure 2. Locality map of the RBPlat Operations, (Padiachy et al., 2021).

RBPlat operation includes the Bafokeng Rasimone Platinum Mine's (BRPM) North Shaft and South Shaft, Styldrift Mine, which include a currently operating Styldrift I shaft and proposed Styldrift II shaft, and the recently acquired Maseve Mine (Figure 3). The Upper Critical Zone rocks containing the Merensky and UG2 Reefs strike northwest for some 6.5 km and 6.8 km across BRPM and Styldrift, respectively. The Merensky and

UG2 Reefs subcrop within the property and occur to a depth of some 1400 m on the most north eastern boundary of Styldrift. Adjacent properties where mining of PGE's is currently taking place include Impala shafts down dip of BRPM North and South Shaft as plotted in Figure 3. North of Styldrift I is the Bakubung Platinum Mine (Wesizwe) whilst to the southwest of Styldrift I is the Maseve mine, which is under care and maintenance. The Merensky Reef is currently being extracted from BRPM's North and South Shafts and Styldrift I. Mining of the UG2 is limited to BRPM North and South Shaft. The Styldrift I shaft is the newest mine added to the RBPlat mining operations which has just completed ramp up to a steady state production.

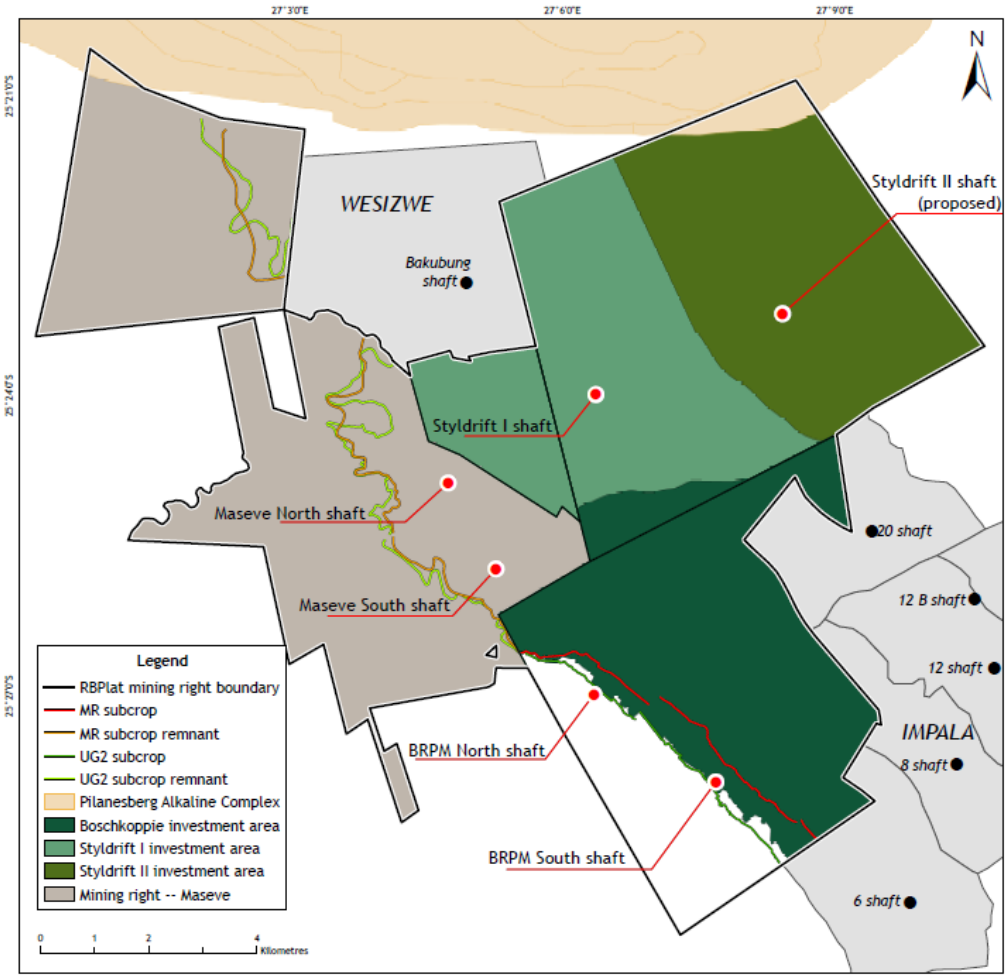

Figure 3. Spatial distribution of the BRPM JV shafts and neighbouring shafts, (Padiachy et al., 2021).

1.1.3 **Background and Mine History**

The Royal Bafokeng Nation (RBN) commenced purchasing land in the 1860's. Thereafter, in 1925 platinum was discovered in the BIC, known as the world's largest platinum group metal deposit. The Merensky Reef was discovered on the farm Maandagshoek near Steelpoort by Dr Hans Merensky who was alerted to the find by a farmer Andries Lombaard in 1924 (Cawthorn, 2007).

The first platinum exploration in the RBPlat lease area was undertaken in 1950 but was soon abandoned by 1956 as the area was considered too complex for mining operations at the time. Exploration resumed in 1993 by Anglo American Platinum (Amplats) resulting in a successful exploration campaign (Padiachy et al., 2021). Discussions on forming a 50/50 joint venture ownership between RBN and Amplats started in the 1990's. Site establishment of the Bafokeng Rasimone Platinum Mine Joint Venture (BRPM JV) commenced in 1997 on the farm Boschkoppie 104JQ (Figure 3). Initially the BRPM JV operated as an opencast Merensky mine, followed by development into underground mining which started in December 1998. BRPM produced its first platinum concentrate, which was sold to Amplats for beneficiation in its refineries in December 1999. A 50:50 joint venture agreement between Amplats and the RBN formed the BRPM JV in 2002. In 2009 RBN created the company Royal Bafokeng Platinum and restructuring resulted in RBPlat obtaining 67 % interest in the BRPM JV. In 2010 RBPlat assumed operational control and became the first community owned business to list on the Johannesburg Stock Exchange. Successful acquisition of Amplats' 33 % interest in the BRPM JV took place in 2018 with the transfer of the undivided interest in the Mining Right to RBPlat (www.bafokengplatinum.co.za).

Current operations at BRPM's North Shaft and South Shaft are accessed via two twin decline systems extracting both the Merensky Reef and the UG2 Reef horizons by means of a conventional mining method. Styldrift I Shaft (study area) is a newly sunk vertical shaft, with the ramp up to mining the Merensky Reef now completed, having reached steady state production

in 2021. There is a regional facies difference between BRPM's North and South Shaft to that of Styldrift. A bord and pillar mining method is planned for large portions of Styldrift I, due to the thicker Merensky Reef mineralisation, stable ore-body planarity and the low dipping orebody which supports a mechanised mining approach in this area.

The current mining extraction of Merensky and UG2 reefs on the RBPlat operations are depicted in Figure 4.

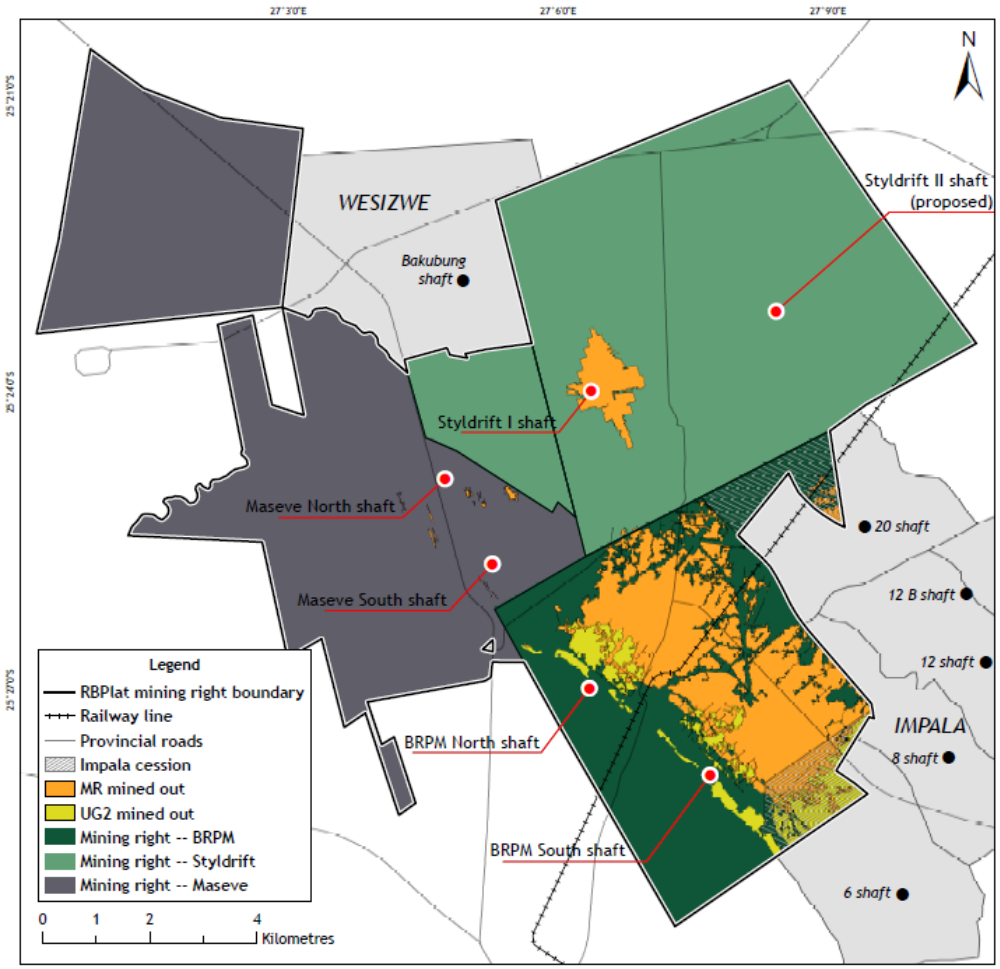

Figure 4. Current mining operations and mined out areas at the BRPM JV, (Padiachy et al., 2021).

1.2. Purpose of the Research Study

The main purpose of the research study was to find an optimal sampling scheme for the new Styldrift I mine to result in an estimate at an acceptable level of accuracy that is practical in its application and to understand the

associated cost. The current sampling standard applied had been adopted from the older existing operations at BRPM. As a result, there is a need for an appropriate underground sampling strategy for the Merensky Reef within the Styldrift I facies sub-domain.

1.3. Research Aim and Objectives

As mentioned, the sampling standard that is applied on this new mining operation has been inferred from the neighbouring mining shafts. Styldrift I facies of the Merensky Reef is located in a transition area of a regional facies change and therefore also varies locally. The aim of this research study was to establish an appropriate underground sampling strategy for the Merensky Reef within the Styldrift I that focussed on the facies sub-domains where current mining is taking place in this new shaft. An understanding of the local geological domains will assist in defining an optimum sampling strategy. Whilst defining the optimal sampling strategy from a technical viewpoint, an additional key factor in the research is to understand the cost benefit of implementing the recommended sampling scheme at Styldrift I.

The objectives addressed in this research study are the following:

- Development of an optimal sampling grid required to obtain a grade estimate of acceptable accuracy by applying geostatistical techniques,
- Selection of a sampling pattern that would be practical in its implementation and appropriate for the new mechanised mining layout for Styldrift I,
- Cost analysis of the sampling schemes and recommend the desired grid which balances confidence in the estimate and the associated cost understanding the loss of opportunity.

1.4. Research Questions

The key questions to be addressed by this research study are the following:

- Can an improved sampling grid be determined using all relevant information by analysing sample grid options using geostatistical techniques?
- Can a cost effective solution be found from the evaluation?
- Can the geology be subdivided into a local characterisation for an optimal design that takes local spatial variation into account?
- Is the data used in the study representative for use in understanding this local spatial variation for the geostatistical evaluation?

1.5. Research Motivation

The success of a new mining project is highly dependent on the evaluation of the grade estimate and reconciliation of the planned versus actual achieved grade, which are both achieved by means of underground sampling. Minnitt (2007) explains how a correct sampling protocol can improve recovery with close reconciliations between estimates and actuals thereby impacting cash flow and profit of the business. Therefore, it is the potential of negative financial impact of employing the incorrect sampling density that is the motivation to investigate an optimal sample spacing for Styldrift I.

1.6. Research Methodology

The methodology required to carry out the investigation involved applying geostatistics to analyse and identify the optimal sampling design. The mechanised mining layout for Styldrift I was studied in order to gain an understanding into the mining production method which included an underground visit and observation of the sampling process. Sampling grid patterns were then designed with this mining layout in mind to ensure practicality. Data was validated and an analysis of the quality and representativeness for its use in this study was assessed. Classical statistical methods were employed to confirm the definition of the local facies types which were used as domains for understanding the variability and modelling this spatial relationship by means of a variogram for use in the geostatistical evaluation. The kriging variances were calculated as a measure of estimation confidence for the sampling options and a number of

iterations were tested, plotted, and examined to identify the proposed sampling grid. An implicit cost comparison of the various sampling strategies was conducted to understand the cost benefit of each sampling grid. Followed by the final recommendation on the implementation of the optimal sampling strategy that met technical, practical and financial requirements for Styldrift I.

1.7. Research Assumptions

The following research assumes the same assumptions of mineralisation continuity as that of any geostatistical evaluation. Ordinary kriging relies on the assumption of stationarity whereby it is assumed that the underlying grade distribution is the same throughout the domain. In this research study stationarity is assumed; meaning the known sample data is representative of the domain and reflects the underlying grade distribution characteristics and features at the unsampled locations within the domain. Under this assumption Ordinary Kriging can be applied to produce estimates at all locations within the domain.

1.8. Introduction Summary

This research study was conducted at the locality of the Styldrift I mining operation of RBPlat. RBPlat operations are situated on the western limb of the Bushveld Igneous Complex with the Styldrift I project as the newest shaft currently extracting the Merensky Reef by means of a mechanised bord and pillar method. The current sampling strategy is adopted from the neighbouring RBPlat mines but there is a regional facies change between the older BRPM shafts and Styldrift I. The research study addresses the requirement of designing and evaluating an optimal option for an underground sample strategy that is both practical and economical. It employed the methodology of applying a geostatistical evaluation of the kriging variance estimation error to test the various sampling grid options.

2. LITERATURE REVIEW

The scope of the literature review for the research study covers the geological context of the study area. A detailed understanding of the geology provided foundation for the work, therefore the literature documents both the regional and local geology. The justification for the research study is also provided as part of the literary review highlighting the importance of sampling in a mining operation. The literature review was used to develop insight into the research objectives, assisted with the approach for the research project and provided key concepts that were employed.

2.1. Geological Setting Bushveld Regional Geology

The Bushveld Igneous Complex found in the northern part of South Africa, is the world's largest layered intrusion. The complex is also a rich source of minerals and renowned for its economic interest hosting over half of the world's platinum, chromium, vanadium and refractory minerals (Eales, 2001). The emplacement of the Bushveld Igneous Complex is dated at 2.06 Ga (Eales 2001; Kinnaird, 2005) with an area of 65 000 km^2, covering a distance of 450 km east to west and 250 km north to south, spanning parts of the Limpopo, North West, Gauteng and Mpumalanga Provinces. The BIC consist of 3 major lobes namely the Western, Eastern and Northern limbs as illustrated in Figure 5 (Kinnaird, 2005). The four main suites of rocks of the BIC are the Rooiberg Group volcanics, the mafics of the Rustenburg Layered Suite, the felsics of the Lebowa Granite suite and the Rashoop granophyre suite (Kinnaird, 2005).

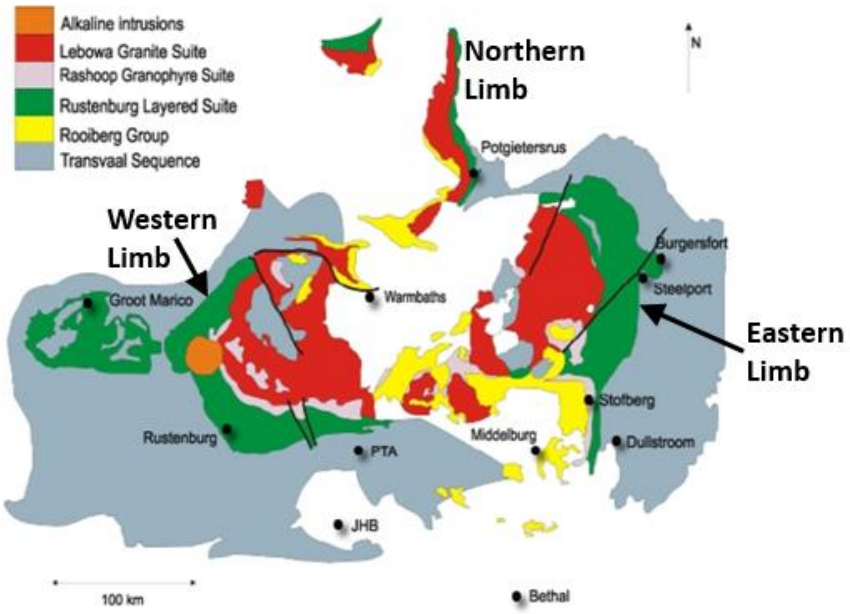

Figure 5. The Bushveld Igneous Complex
(Source: http://www.largeigneousprovinces.org)

The Rustenburg Layered Suite with a maximum thickness of up to 9 km (Kinnaird, 2005) is subdivided into 5 units namely the Marginal Zone, Lower Zone, Critical Zone, Main Zone and Upper Zone. These zones with their respective rock types and reef types are displayed in the stratigraphic column of Figure 6. The Marginal Zone is not always present but is considered to be the basal contact of the BIC. The most common rocks are norite with some occurrence of pyroxenite and harzburgite. The Lower Zone contains layering of harzburgite and pyroxenite. The Critical Zone layering is geologically impressive hosting the largest stratiform chromitite deposit and a world class PGE deposit (Kinnaird, 2005). The Critical Zone is divided into a lower and an upper zone. The lower zone is ultramafic with seven extensive chromitite layers referred to as the Lower Group (LG1-7). The Upper Critical Zone comprises of cyclic packages of chromitite, harzburgite and pyroxenite through norite to anorthosite (Eales, 2001; Kinnaird, 2005). Four chromitite layers occur across the Lower and Upper Critical Zone termed the Middle Group (MG1-4) whilst there are two Upper Group (UG1-2) chromitite layers are found in the Upper Critical Zone. The well-known platinum bearing layers, the Merensky Reef and UG2 Reef occur in the Upper Critical Zone as highlighted in Figure 6. The Main Zone which is

greater than 3000 m thick comprising of a succession of norite and gabbronorites. The Upper Zone's dominant rock type is gabbronorite but it is also host to a rich source of vanadium from 25 titaniferous-magnetite layers (Kinnaird, 2005; Viljoen, 2016).

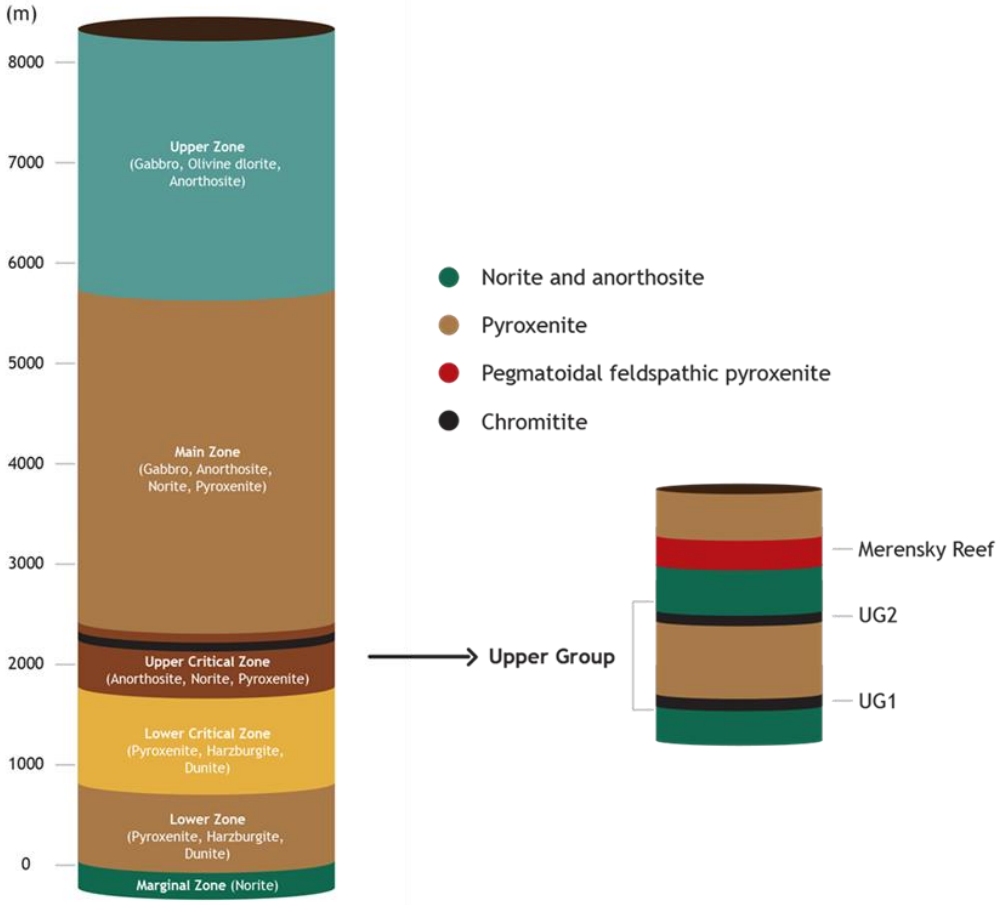

Figure 6. Stratigraphic column of The Rustenburg Layered Suite, (Padiachy et al., 2021).

2.2. The Merensky Reef

The focus of the sampling strategy investigation is on the Merensky Reef which is currently being mined at Styldrift I. The Merensky Reef is defined as the mineralised portion of the Merensky Pyroxenite comprising of a heterogenous pegmatoidal feldspathic pyroxenite bounded by two chromitite stringers above and below. PGE mineralisation is concentrated at or tends to peak at these stringers. The Merensky Reef may also consist of a single chromitite stringer only, with an overlying hangingwall pyroxenite.

The immediate footwall to the reef is typically norite or anorthosite which may also contain mineralisation (Viljoen, 2016; Smith, 2013). The dip of the Merensky Reef within this area is in a north easterly direction between 5º and 12º, steepening on the north eastern side of Styldrift. The Merensky Reef hosts the world's largest known resource of platinum (Viljoen, 2016; Cawthorn 1999).

2.3. Merensky Reef Facies

2.3.1 Regional Facies (Rustenburg vs Swartklip)

The study area is located in a Regional Facies transition zone which adds to the complexity and variability of the reef. This section discusses the facies classification and provides background to defining the facies types that informs the domaining for the sampling evaluation in the study area.

Reef facies refers to a unique geological domain with regards to a specific reef type with consistent similar geological characteristics such as lithology, mineralogy, stratigraphy and geochemistry. Facies is a sedimentary concept defined as "the appearance and peculiarities that distinguish a rock unit from an associated or adjacent unit" (Lapidus, 1990). This principle is applied in the igneous environment in order to understand the vast lithological and stratigraphic variations of a single rock unit and ultimately the characterisation of the associated mineralisation. The Merensky Facies classification is largely a function of the reef characteristics, the stratigraphic position of the reef and the footwall layering (Moodley, 2008).

The regional facies delineation is related to the thickness between the Merensky Reef and UG2 Reef. The Upper Critical Zone of the Rustenburg Layered Suite thins substantially in a north westerly direction on the Western Limb of the BIC. The vertical separation between the Merensky and UG2 reefs decrease from ~150 m southeast of Rustenburg to ~30 m north of the Pilanesberg (Vermeulen, 2010).

Regional variations in the geological characteristics of the Merensky Reef on the Western Limb was first noted by Wagner (1929) and subdivided the Merensky Reef into two main subtypes or facies. He termed the occurrence

of the Merensky Reef south of the Pilanesberg, the Rustenburg Facies and north of the Pilanesberg, the Swartklip Facies.

The Rustenburg facies of the Merensky Reef, as Wagner (1929) defined, is developed from just south of the Pilanesberg in the west, to beyond Brits in the east (Viljoen, 2016). The Swartklip facies occurs to the north of the Pilanesberg and is well known at Union, Tumela, Dishaba and Zondereinde Platinum mines (Wagner, 1929; Smith et.al., 2003).

The transition between the Rustenburg Facies and Swartklip facies occurs in the area of BRPM and Styldrift. There are significant stratigraphic changes from the Rustenburg Facies at BRPM to the Swartklip Facies at Styldrift. It should be noted that the stratigraphic footwall unit nomenclature applied at RBPlat operations is adapted from Leeb du Toit (1986). The footwall number begins from Footwall 1, which is the first unit below the Merensky Reef and increases in numbering moving down deeper into the stratigraphy. Footwall is abbreviated to FW in the naming convention, where norite footwalls are assigned odd numbers and poikilitic anorthosite footwalls are assigned even numbers (Moodley, 2008). Illustrations of the stratigraphic columns are depicted in Figure 7.

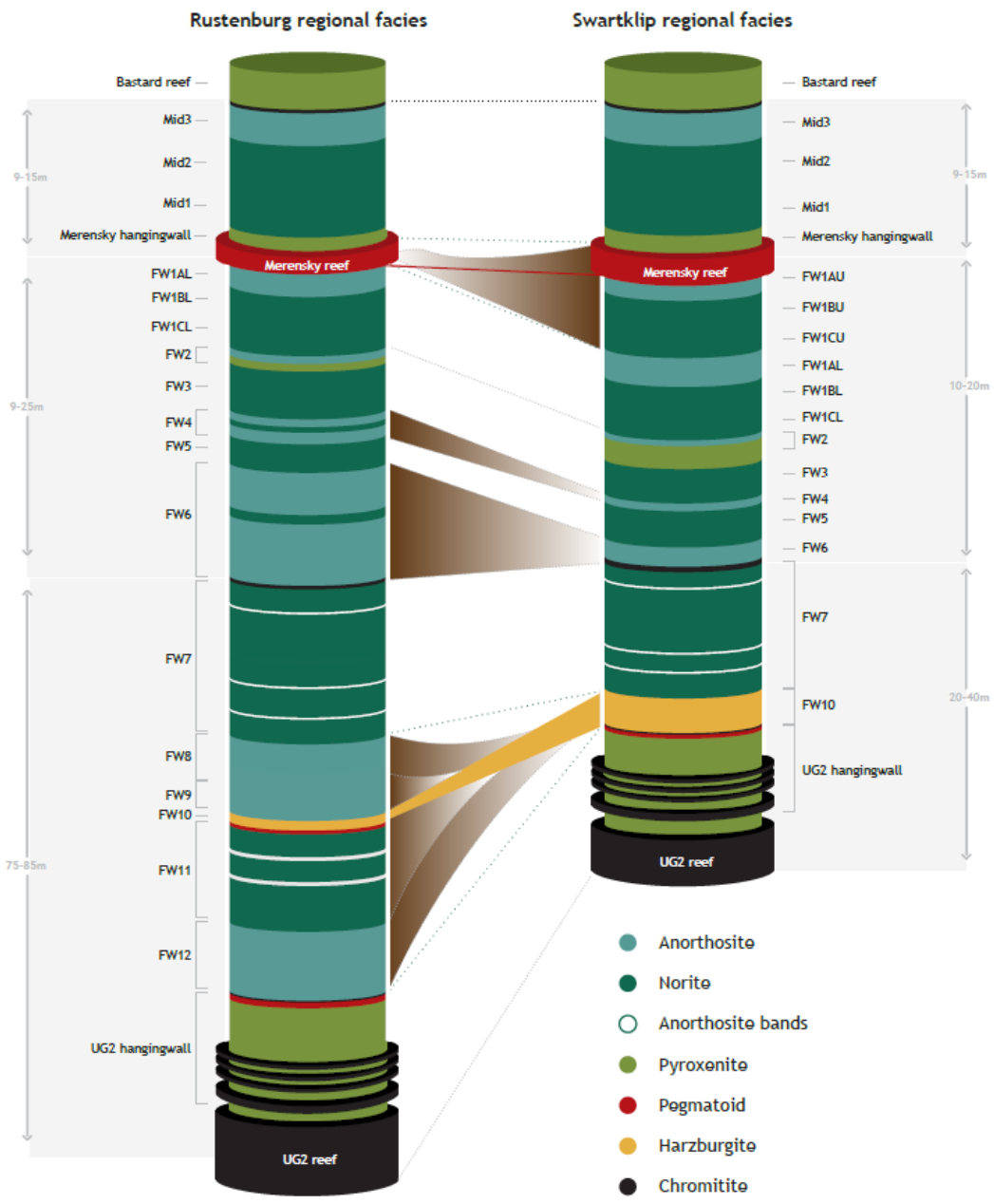

Figure 7. Typical Stratigraphy of the Rustenburg and Swartklip Regional Facies Types, (Padiachy et al., 2021).

The first noticeable difference is the duplication of the FW1 stratigraphic layer towards the northeast of Styldrift where the Merensky is now situated on a higher stratigraphic elevation of the FW1 cyclic unit as illustrated by the thickening isopach in Figure 8 (green to yellow shade of colours). The second prominent change moving north east from BRPM into Styldrift is the thinning of the footwall stratigraphy with the middling between the Merensky and UG2 decreasing from an average of 75 m to 15 m. Figure 9 isopach

depicts the pinching out of FW8, FW9, FW11 and FW12 (Vermeulen, 2010). FW7 overlies the feldspathic harzburgite of FW10 which is thicker and is suggested to correlate with the Pseudo Reef commonly known as the Tarentaal reef at Union Section (Viljoen, 2016). Refer to Figure 7 for a stratigraphic column comparison between the two regional facies types.

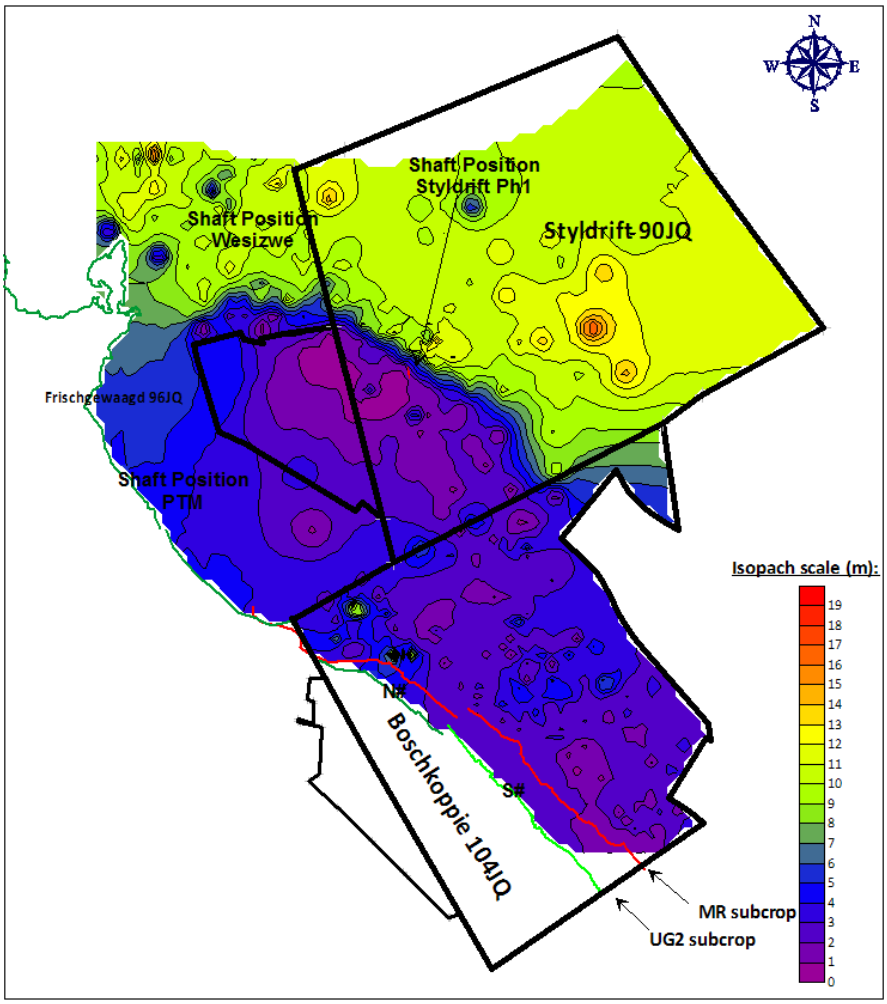

Figure 8. Vertical thickness isopach between Merensky Reef and FW1 bottom, (Vermeulen, 2010).

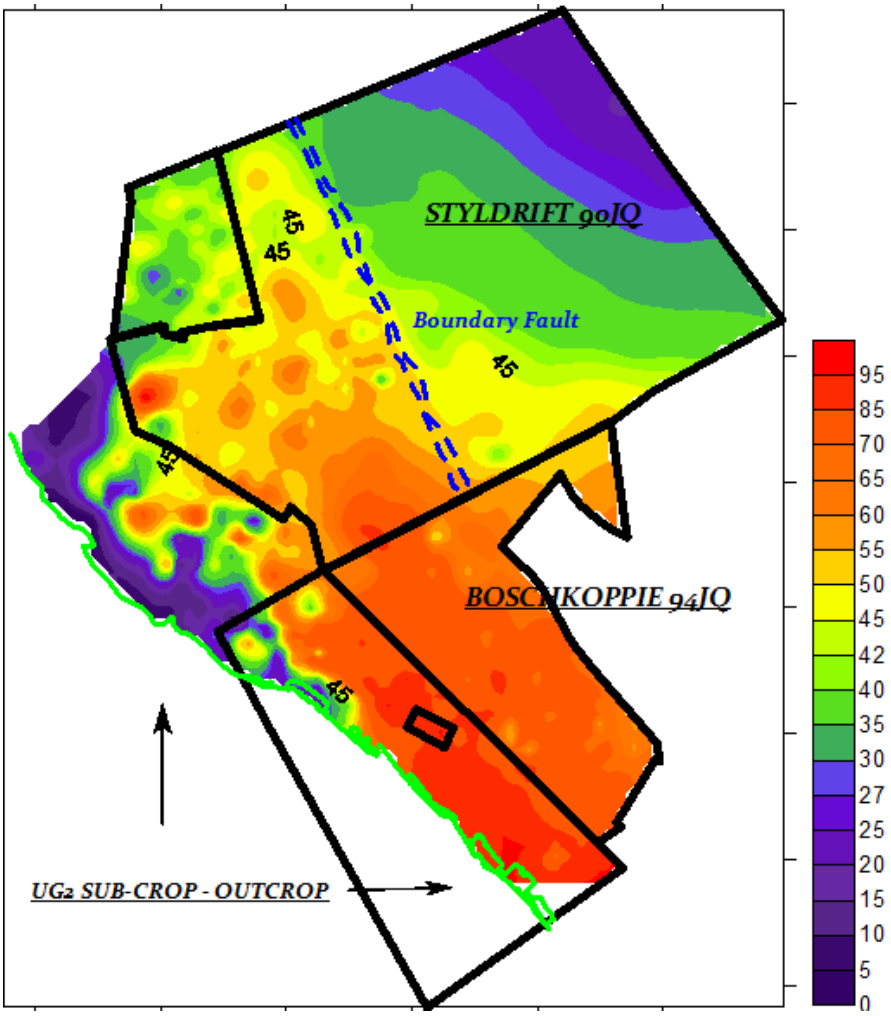

Figure 9. Isopach of the middling between Merensky and UG2, (Vermeulen, 2010).

2.3.2 Local Facies Variation within the Merensky Reef

Mining of the Merensky Reef orebody at BRPM proved to be geologically complex due to its highly variable nature. This variability necessitated a classification of the local facies types (Moodley, 2008), which has become key in the mining history of BRPM evaluation model. It highlights the importance in fully understanding the geology at the newer Styldrift mine. It is common practice for mines to subdivide regional facies types into smaller local facies types that are mine specific (Lionnet and Lomberg, 2007; Moodley, 2008).

These unique geological facies types at BRPM and Styldrift have been well defined and interpreted from studying information of a large dataset of exploration drill holes, underground prospect drilling and 20 years of

underground mining. Local facies types have been identified based on the reef and footwall lithology type, the associated stratigraphic sequence, widths and mineralisation distribution or grade profiles. The local Merensky facies constitute from west to east are Abutment Facies, Terrace Facies also referred to as Pothole Facies and Central Reef Facies which fall under Rustenburg regional facies type and from the Swartklip Facies type is the Transition Facies, Normal Reef Facies, Normal Top Loaded Facies, Main Reef Facies. The cross-section and Merensky facies plan are demonstrated in Figure 10 and Figure 11 below.

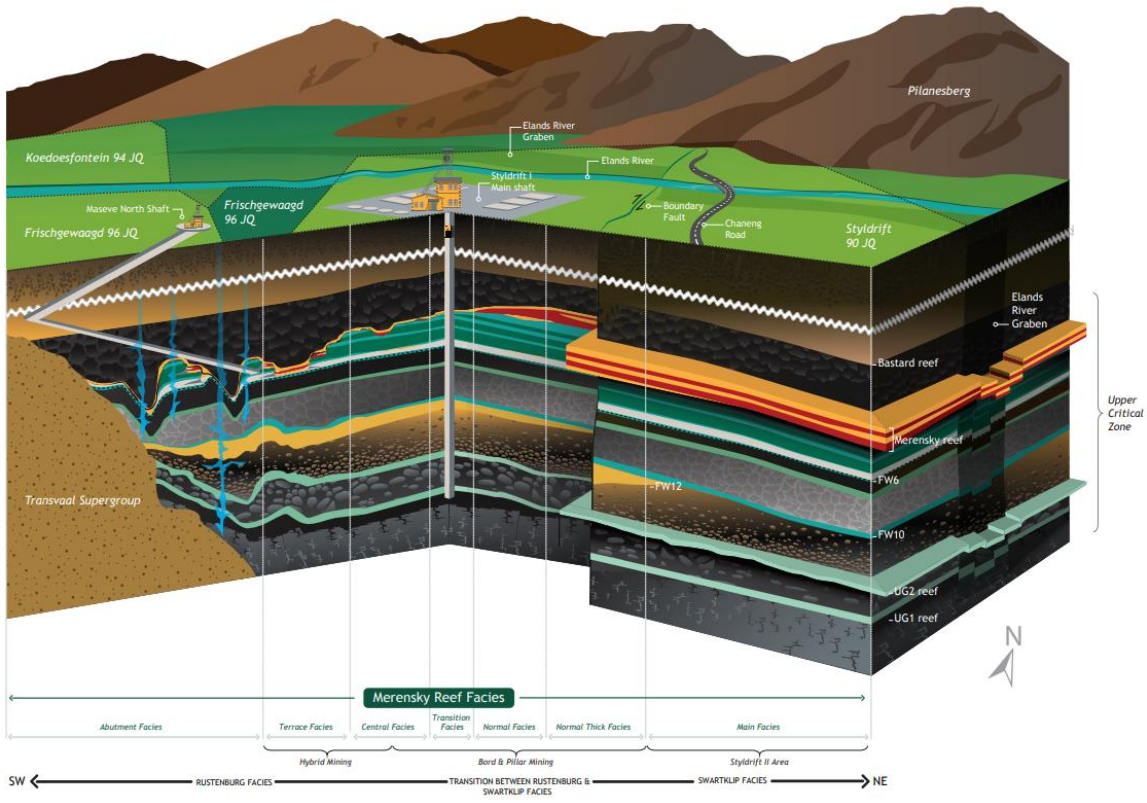

Figure 10. Cross-section view of the Merensky Reef Facies, (Padiachy et al., 2021).

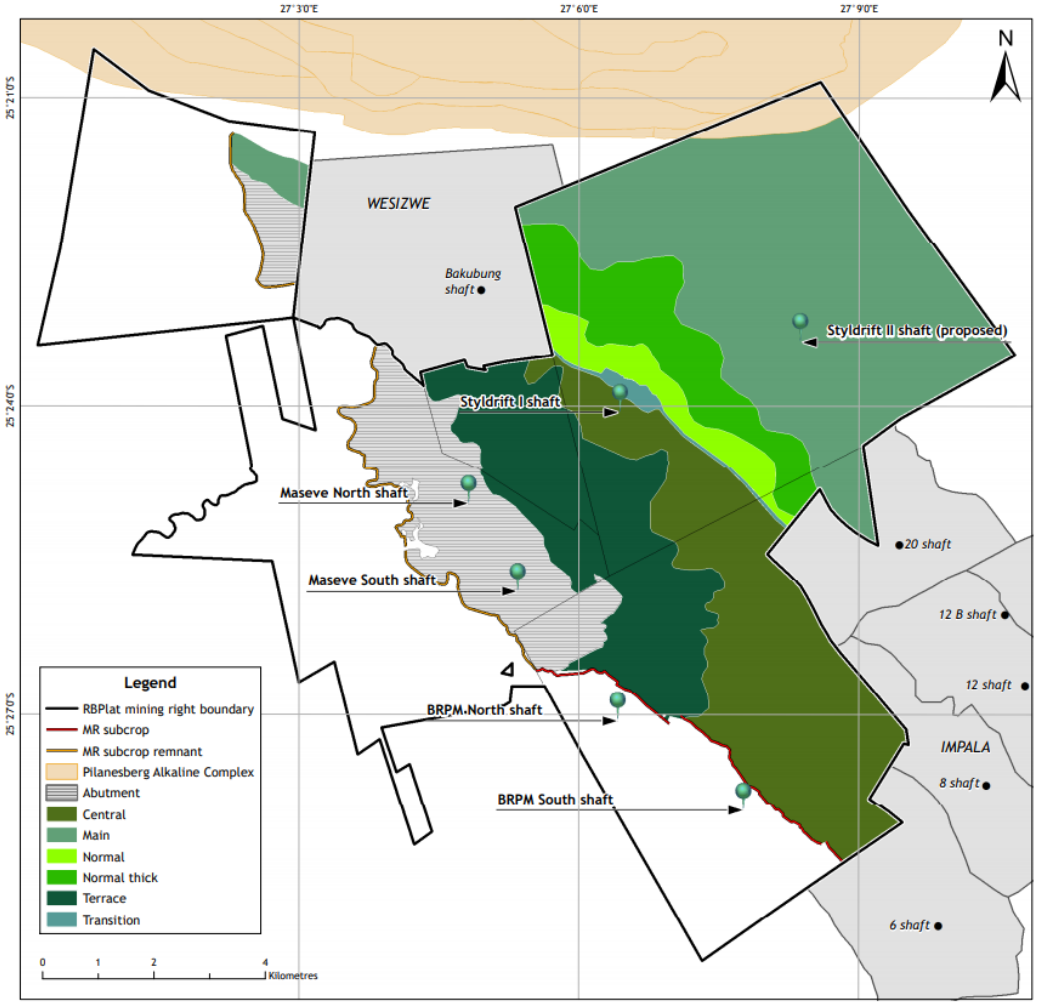

Figure 11. Merensky Facies plan of RBPlat, (Padiachy et al.,2021).

The next paragraph describes the two relevant local facies types over the study area which informs the domains for the sampling evaluation. The Central Reef Facies occurs on a large area of the RBPlat operation from the entire South Shaft extending to down dip eastern part of North Shaft and into the current Styldrift mining area. Refer to Figure 11. The Merensky Reef of the Central Reef Facies occurs as a single chromitite stringer between the hangingwall pyroxenite and the FW1A poikilitic anorthosite. The typical pegmatoidal feldspathic pyroxenite normally associated with the Merensky Reef is absent (Moodley, 2008). The chromitite stringer ranges from 2 mm to 2 cm in width. The chromitite stringer becomes more disseminated when it gets thicker. The mineralisation is concentrated at the chromitite stringer and extends into the footwall as displayed in the mineralisation profile below (Lionnet and Lomberg, 2007; Cawthorn, 2010). The Central Reef Facies is

similar to the Contact Reef sub-facies identified in the Rustenburg Facies (Smith, 2013; Cawthorn, 2010).

The Normal Reef Facies is a pegmatoidal felspathic pyroxenite bounded by a bottom reef chromitite and a top reef chromitite, either of which may not always be present. This reef is situated on a poikilitic anorthosite of the FW1A upper cyclic unit which is where the Swartklip Regional Facies starts to feature. The mineralisation associated with the Normal Reef Facies is confined to the pegmatoidal feldspathic pyroxenite and kicks at the chromitite stringers when present (Figure 12). There is no footwall mineralisation with this reef type and often termed as top loaded (Smith et al., 2004), which is clearly depicted in the grade profile below.

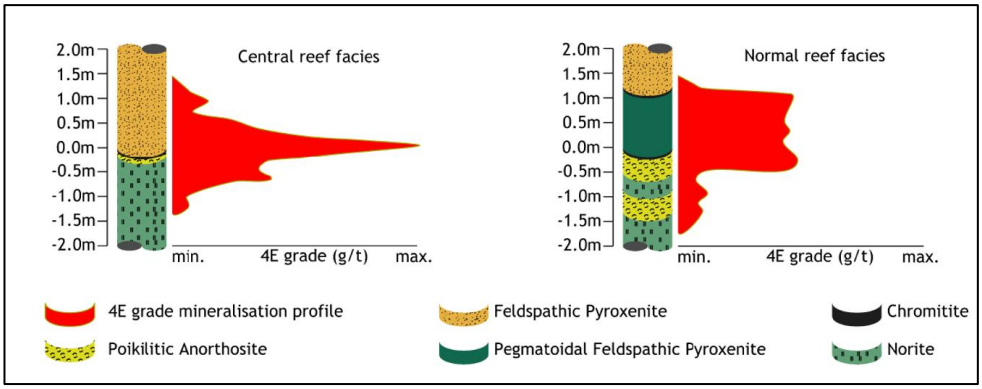

Figure 12. Merensky grade distributions for the Central and Normal Reef facies, (Padiachy et al., 2021).

2.4. Underground Mine Sampling and its Importance

Dominy (2015) explains sampling is a process of sample collection, preparation and assaying with the aim of gaining a representative sample that is an accurate description of the unknown content (Dominy, 2015). The sampling process entails two steps which is the physical in-situ sample collection and secondly the chemical analysis of those samples. Lomberg (2014) documents the best practice sampling methods, assay techniques and quality control employed across the PGE industry which have been developed over several decades. He emphasizes the need for a better understanding of the challenging PGE orebodies and the selection of the

different sampling techniques which is the foundation of developing a geological model and Mineral Resource estimate.

The mining department, on operating mines extracting the Merensky Reef, require a mining cut instruction as the mineralisation is not bound by lithology and cannot visibly be seen. Determination of the cut-off grade and the optimal mining cut for economic extraction is achieved by means of sampling of the underground excavations. Similarly, at BRPM, underground sampling is used for the daily grade control evaluation and provision of an optimal cut, more commonly known as a best cut. The data is also used for comparisons of planned versus actual grades and reconciliation with the processing plant which is used for evaluation and long term planning (Minnitt, 2014). The implementation of a correct sampling protocol will improve metal recovery and assist in attaining a closer reconciliation between the grade model estimate and the metal output, which ultimately leads to increased profits and value add (Minnitt, 2007).

The importance of underground sampling is discussed in the three points below:

2.4.1 Small Scale Variability

Precious metal orebodies are known to have small-scale variability and the understanding of this variability is an essential part of the mining operation (Minnitt, 2014; Lomberg, 2014). Underground sampling assists in providing the information to gain knowledge on the reef variability. In-situ variability can be enhanced due to poor sampling practices (Dominy, 2015). A poorly designed sampling protocol introduces errors and bias. Dominy states that when sampling is optimised to suit the orebody type, sampling error can be reduced and therefore decreases the risk of financial and intangible losses (Dominy, 2015). There tends to be an under appreciation towards the gains and value add of an effective and optimal sampling strategy (Minnitt, 2014; Dominy, 2015). Sampling informs many studies on which major financial decisions are made and can influence valuations with implications of losses or missed opportunities (Minnitt, 2007). The importance of sampling not only in reference to underground sampling but across the entire mine value chain

has been discussed and documented at length by many (Carrasco et al., 2004); (Pitard, 2009); (Minnitt, 2007); (Dominy,2015). Carrasco et al. (2004) demonstrates the value of sampling density to mining and the negative economic effect of an incorrect sampling pattern which drives the understanding of the grade variability. Incorrect or sub-optimal sampling protocols compromise decision making and result in the misclassification of ore and waste. The financial implications can be significant due to ill-informed operational decisions, poor Mineral Resource and Mineral Reserve estimates and poor reconciliation (Dominy et al., 2018).

2.4.2 Front end loading

Styldrift I project is currently at a critical stage with completion of ramp up and at the beginning stages of full production. Full underground access and exposure to the orebody is available to evaluate the long term sampling estimations and conduct reconciliations of the actual value as mining progresses. Early development is the most vulnerable stage in the life of a mine (Minnitt, 2007). Minnitt describes the mineral process development as a stacked pyramid with exploration at the base, followed by technical analysis and financial assessment at the top, refer to Figure 13. The application of sampling expenditure at the early stages of the mine, front end loading, increases the probability of a successful project with large returns. Lack of sampling information at the prospecting level can result in biased Mineral Resource and Mineral Reserve estimates, project delays and wasted capital expenditure. At a mine operation level, poor sampling could relate to ore and waste misclassification and issues of reconciliation (Dominy, 2015).

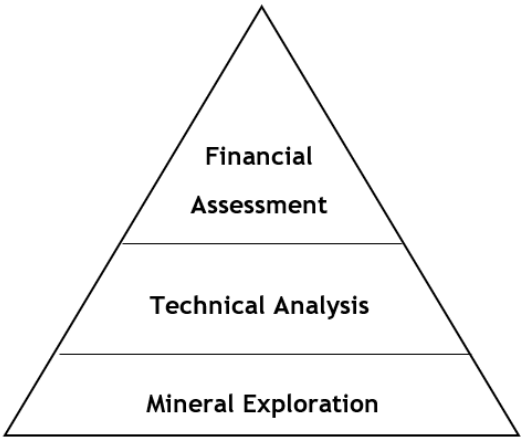

Figure 13. Mineral Process Development hierarchy, (Minnitt, 2007)

2.4.3 Regulatory requirements

Discussion on changes to the political and economic environment marked an interest into sampling in the minerals industry in reference to a paper on 'Sampling in the South African Minerals Industry' by Minnitt (2014). Growing foreign investment and The Mineral Resources and Petroleum Resources Development Act (Act 25 of 2002) providing opportunities of investment for those who were historically disadvantaged. The importance and interest in sampling was stimulated due to mergers and acquisitions for due diligence studies and valuations. The infamous Bre-X scandal in the late 1990's, created a worldwide focus on sampling and resulted in the creation of guidelines and codes to control the reporting of Mineral Resources and Mineral Reserves (Minnitt, 2014).

Regulatory bodies have introduced a greater appreciation for the relevance and importance of sampling in the developing a Mineral Resource. The South African Code for Reporting of Exploration Results, Mineral Resources and Mineral Reserves (SAMREC) provides a checklist referred to as Table 1 which requires the disclosure of relevant sampling information such as sampling method, collection, capture, storage, sample preparation and analysis and sampling governance. The Johannesburg Stock Exchange listing requirements in their Section 12 applies the SAMREC Code for listed companies disclosure rules. Minnitt (2014) also highlights the vast wealth of mineral resources which contribute greatly to South Africa's economy and therefore correct sampling is of key importance to the mining industry.

2.5. The Application of Geostatistical techniques to design an Optimal Sampling Scheme

Englund and Heravi (1993) and Armstrong (1998) state how the application of geostatistics is commonly used to solve the issue of designing sample schemes which was introduced by Journel and Huijbregts in 1978. Particularly, the use of the standard error of a kriged estimate or the kriging variance as the measure used to determine ideal sampling patterns (layouts) and the minimum number of samples required to attain a certain degree of accuracy is discussed by authors such as McBratney et al. (1981) and Rendu (1976).

The sampling configuration and sample density determines the quality of kriging predictions therefore, the kriging variance is used as a tool to evaluate the quality of the sampling scheme (Van Groenigen, 2000; McBratney et al., 1981). Kriging is a process that estimates values for unsampled locations using a weighted average of neighbouring samples. The weights are a function of the semi-variogram and therefore can be optimised based on our understanding of how samples relate to each other in space. The semi-variogram models the grade continuity according to the distance and orientation between samples (Clark and Harper, 2000; Coombes, 2008). Atkinson and Lloyd (2007) confirm that kriging can be used as a tool to design optimal sampling strategies by considering the kriging error with a known variogram for various strategies.

The results from the drill hole sample analysis can be used to prioritise and identify areas that require sampling or as in this present research suggest a grade control density required to capture the short term variability for optimal extraction. Publications by Bertoli et al. (2013) and Williams et al. (2015) describe the application of drill hole spacing analysis based on estimation variance as a measure to assess confidence in a coal operation in the Bowen Basin. A geostatistical analysis of quantitative measure of the global estimation precision was used to characterise the spatial continuity of seam thickness and raw ash resulting in a variation of proposed spacings for the different coal seams. The proposed drill hole spacings were compared to

coal reporting guidelines who is often used sample spacing in coal studies is the key determinant parameter for the Coal Resource confidence. It was recommended that the use of the geostatistical method is considered as best practice as it is drive by the actual in situ variability or spatial continuity.

2.6. Domaining and the application of a local variogram

Resource estimation and geostatistics is based on the concept of stationarity, whereby data of similar geological, mineralisation and statistical properties are used to estimate a zone (Ronald, 2016). Domaining is the boundaries within which areas of similarity are grouped into a single grade population and the geology is known to be consistent (Coombes, 2008). Domain interpretation and understanding the variability or spatial relationship within a domain is very important when modelling variograms for a kriged estimation. A variogram is calculated from the samples within the same domain as the statistical and geological properties are the same (Ronald, 2016).

Variograms form a central part of the geostatistical model and as a result it is discussed how the application of a global variogram or also referred to as stationary variogram may be inappropriate (Atkinson and Lloyd, 2007). Atkinson and Lloyd compared the kriging errors predicted using the global variograms compared to the local variograms and found large differences whereby the local segment based variogram estimation was more accurate which will assist in a more efficient sample design. They further recommend being cautious when applying a stationary variogram or global sampling design as a local application of sampling designs prove to be more optimal.

2.7. Literature Review Summary

In summary, the literature review emphasises the vital contribution sampling makes in the evaluation of a Mineral Resource and the severe economic impact if done incorrectly or not done at all. Geostatistics is commonly used to design optimal sampling schemes which is the tool employed in this research study. The study area is located in a transition zone of a regional geological facies change from the Rustenburg Facies to the Swartklip Facies which adds to the variability of the reef which is well documented in

previous geological work. The regional facies which is further subdivided into the local facies types was an important consideration in applying the geostatistical analysis correctly in terms of understanding the small scale variation. The concept of the study follows a similar use of comparing kriging variances from a drillhole sample spacing analysis to published studies employed by Williams et al. (2015) and Bertolli et al. (2013) to investigate resource confidences and drill hole spacing determinations from differing sample supports.

3. METHODOLOGY

The first part of the research followed a methodology of reviewing the underground sampling procedure, validating the data used in the research, quality assurance and quality check analysis. This is the foundation and a fundamental step that confirms the data representation and its robustness. The second step is accurate interpretation of the geological facies using statistical analysis and knowledge of the geology. Lastly, the geostatistical evaluation of the kriging variances for the individual sample spacing options was conducted.

3.1. RBPlat Sampling Procedure Overview

This section summarises the physical sampling procedure that was observed during an underground visit at Styldrift I Mine. This assisted in gaining an understanding of how the activity is executed, the quality control employed and the limitations in taking samples in a bord and pillar mining layout which is key in the practicality of designing the sample spacing

3.1.1 Safety Requirements

Safety whilst working underground is of paramount importance. All workers must be educated in basic hazard identification. Personal protective equipment is worn at all times whilst conducting the sampling. Before the sampling team proceed into the workplace, a safety meeting is conducted at the waiting place to discuss any risks and permission must be obtained to enter the workplace from the shift supervisor or miner. Once at the workplace, all hazards are identified prior to the sampling task. Sampling

will only begin when the workplace is declared safe for commencement of work .

3.1.2 Sampling Team requirements

The personnel required for the sampling team include a qualified sampler with a minimum basic sampling certificate, a diamond saw operator certified and competent in the use of a diamond saw, and at least three sampling assistants. The equipment for sampling includes a standard diamond saw, an air hose and one water hose, a step ladder, pre-printed bar coded sample tickets, a clino-rule, a sampling dish, chalk or crayons, sharp chisels, a brush to clean the face and sample bags.

3.1.3 Sampling Procedure

Sample sections are taken according to the mine's sample spacing standard, which is under review in this study. A sample section may only be moved due to dangerous conditions, obstructions or major geological features. The face for the final sample position must be cleaned and trimmed of loose rocks that may cause contamination or incidents. Two section lines demarcating the channel groove are drawn perpendicular to the dip of the reef extending into the hangingwall and footwall. Samples are demarcated at right angles to the section lines. The first sample reference is taken 2 cm below the bottom reef contact to ensure no loss or dilution of the high grade reef sample. Channel section lines are 8 cm apart and samples lengths vary from 10 cm to 15 cm and may not exceed 20 cm. A mark-up of the samples is outlined in Figure 14. These sample dimensions ensure a minimum 300 g sample mass requirement. Once samples are marked, lithological rock types were recorded with a sample number by means of a sketch for reference. The diamond saw operator cuts the samples to a minimum depth of approximately 4 cm to ensure the required sample mass is achieved. The face is washed thoroughly to remove sludge from cutting and samples are chipped with the chisel from bottom to top in order to limit contamination. The sample dish is held below the sample being chipped to ensure all the rock sample is caught and not lost. Contaminated

samples are discarded and re-cutting of the sample is taken adjacent to the lost sample. Samples are bagged with a unique sample barcode.

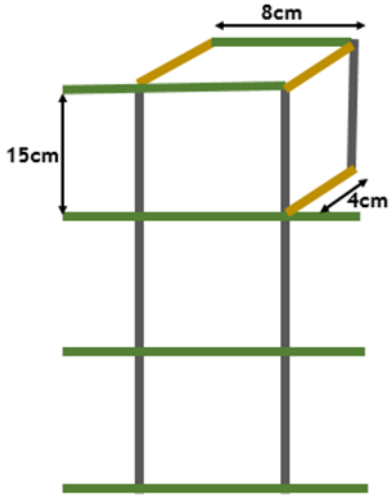

Figure 14. View of sample section

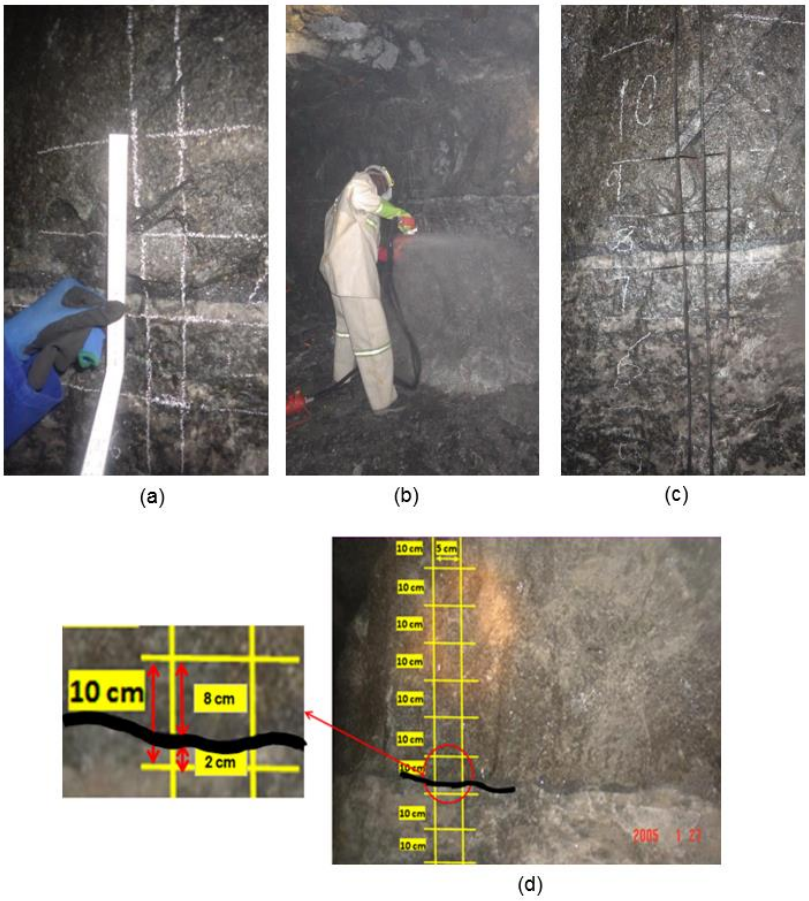

Figure 15. (a) Marking up of samples using a clino-rule for measuring; (b) Diamond saw cutting of the demarcated samples; (c) Samples taken at 10 cm intervals with the first sample marked at 2 cm below the bottom reef contact.



Figure 16. (a) Chipping of samples with a chisel from bottom to top of the sample channel; (b) Washing of sample instruments and bowl after every sample to prevent contamination; (c) Samples put into clean bags which are properly labelled.

3.2. Exploratory Data Analysis or Classical Statistics Investigation

Exploratory data analysis (EDA) includes a classical statistical analysis which is the first step that lays the foundation in any geostatistical analysis. A statistical analysis characterises and interprets the behaviour of the existing sample data in order to produce the best estimate for unknown sample values thereby guiding us in search for optimal locations for future sampling (Clark and Harper, 2000). Clark and Harper argue that before the application of geostatistics there are some essential assumptions made of the sampling dataset. Exploratory data analysis is used to validate the data against the principles of the required conditions, which are listed below:

1. Sample values are precise and reproducible.
2. Sample values are accurate and representative of the true value that location.
3. The samples taken constitute a very small portion of the population of potential samples.
4. The samples were taken randomly and independently from the population and are not biased.

3.2.1 Data Validation and QAQC

All available sample data from locations that are not structurally disturbed and located within the two facies types (Central Reef Facies and Normal Reef Facies) that cover the selected study area, was extracted from the RBPlat sampling database for analysis. These sampling methods of collection include geological core drilling from exploration surface, underground geological prospect core drilling and underground channel sampling. To ensure integrity of data being used, the selected sample dataset was rigorously checked and validated as part of quality control and quality assurance (QAQC). Data quality is more critical than the adoption of the correct estimation technique therefore considerable time and effort has been spent on ensuring all checks are performed.

Checks and validations conducted include, but were not limited, to the following:

- Verification of the lithological and stratigraphic sequence.
- Validation of collar co-ordinates.
- Missing 4E grade values.
- Sample 4E prill grades outside the minimum and maximum thresholds.
- Sampled lengths less than the minimum or greater than the maximum width tolerance.
- Sample sections where there is no full exposure of the reef present.
- Density values out of expected range.
- Downhole survey dip angles outside of the tolerance limits set.
- The Pt:Pd ratio inside the average range.

In order to test the first two conditions mentioned in Section 3.2, the QAQC of chemical analyses was investigated by means of testing the accuracy and precision of the samples analysed. Precision of a sample measurement can be described as the ability to be able to reproduce the same measured result or how close the repeated result is to the original. It is important to note that even if the repeated result is identical to the original result indicating excellent precision, it does not mean that the results are the true

value (Abzalov, 2011; Pitard, 2009). Accuracy of a sample measurement is a result that is closest to the correct true value (Abzalov, 2011; Pitard, 2009). An illustration wherein the difference between accuracy and precision is provided in Figure 17. If one were to play a game of darts targeting the bullseye which represents the true value and all your darts symbolising the sampled values hit the centre then it would be both accurate and precise. If all the darts are clustered in one area but far from the centre bullseye, then it would be precise but not accurate. If the darts are spread around a wider area but close to the centre, it is accurate but the precision is poor. Darts spread are all over the board is poor precision and poor accuracy.

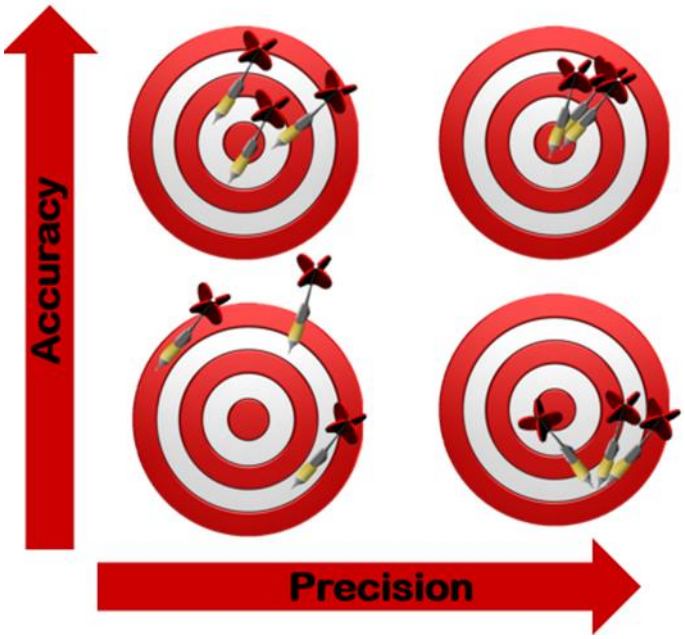

Figure 17. Accuracy versus Precision

The analytical QAQC method applied is according to the RBPlat standard procedure, which is compliant with industry best practice. Precision is tested by means of a twin stream analysis (Moodley, 2015). 10 % of the sampled data is crushed and split into two legs that are assayed separately. The two values are compared by means of the average percentage relative difference (%AVRD) = 100 X (A – B)/((A + B)/2)), and the correlation coefficient. Twin stream outliers are acceptable within 3 to 5% and suspect if greater than 5%. The acceptable precision range requires 80-90% of the data falling within a 10% difference in the reported value. Accuracy testing the reliability

of the sample value analysed is conducted by inserting certified reference materials (CRMs) known as standards and blanks. Standards are samples with known PGE content values which are submitted with the dispatched sample batch to the laboratory as anonymous samples. The standards are prepared from material that is of the same reef type, which is prepared by commercial laboratories and statistically certified (Abzalov, 2011). Contamination of samples during analysis is detected by the insertion of blanks which contain no or negligibly low metal content (Moodley, 2015). Blanks are generally made from barren quartz (Abzalov, 2011). For the purpose of quality control, 20 % of the submitted samples per batch are certified reference materials as per the minimum required industry standard of good practice. These are equally divided into standards and blanks. For the purpose of this evaluation JMP Version 14 (Statistical software package) has been used, with custom scripts that were originally written by Anglo American Platinum and adopted by RBPlat. Platinum group elements and gold were analysed by lead collection fire assay with ICP finish. Rhodium is analysed when the platinum, palladium and gold prill is greater than 1.5g/t.

3.2.2 Compositing and Statistical Data Analysis

Each reef intersection is coded according to an economic 4E (Pt, Pd, Rh, and Au) grade value and it's stratigraphic unit, i.e. hangingwall, reef and footwall. The samples from each coded unit is extracted and composited into a single 4E grade value, see Figure 18. The composites are length and density weighted as samples are collected over variable lengths which may introduce a risk of bias into the data analysis. The drill hole composites are also corrected for dip to ensure perpendicular true widths are calculated.

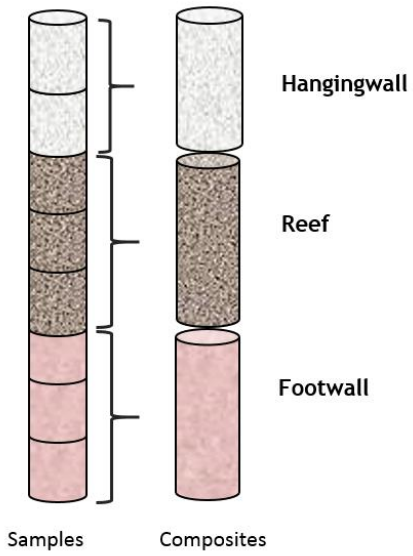

Figure 18. Grade values composited into single units from length and density weighted sample grades

Descriptive statistical analysis was conducted on each data variable, 4E PGE and each individual prill for the 4E grade (Pt, Pd, Rh, Au). This was examined separately for each domain in the study area. Statistics provide a quick and easy methodology to assess the behaviour of the sample data representative of the population (Clark and Harper, 2000). It will be used as means to validate the domains identified, provide an indication of any mixed populations and whether further domaining is required. It also allows for the opportunity to compare the data between the different domains. The statistical tools (Clark and Harper, 2000) discussed below will assist in identifying characteristic trends of each domain and how variable it is

Mean

The mean (μ) is the average value for the variable distribution calculated by dividing the sum of the variable values by the number of samples as per the formula

$$\mu = \sum_{i=1}^{n} \frac{x_i}{n} \qquad \text{Equation 1}$$

Where x_i is the measured value of the sample i, n is the number of samples.

Range

The range is the difference between maximum and minimum sample value.

$$\text{Range} = \text{Max}(\mathcal{X}i) - \text{Min}(\mathcal{X}i) \quad \text{Equation 2}$$

Variance and Standard Deviation

Variance (σ^2) is the measure of the squared deviation of sample values from their mean. Variance is calculated by measuring the difference of the mean from each sample grade. The differences are squared to ensure the positive and negative deviations do not cancel out. The summation of the squared differences are divided by the number of samples less one to account for the loss of a degree of freedom, when subtracting the mean from each sample.

The formula is expressed as :

$$\sigma^2 = \sum_{i=1}^{n} \frac{(x_i - \mu)^2}{n-1}. \quad \text{Equation 3}$$

The square root of the variance is the standard deviation (σ) which is a more meaningful statistic relating to the amount of dispersion in the original sample unit of measurement.

$$\sigma = \sqrt{\sum_{i=1}^{n} \frac{(x_i - \mu)^2}{n-1}} \quad \text{Equation 4}$$

Coefficient of Variation (CoV)

Coefficient of variation is a statistical measure of the variability relative to the mean. It assists when comparing two datasets and gives an indication of the severity of the skewness or shape of the distribution.

$$CoV = \frac{\sigma}{\mu} \quad \text{Equation 5}$$

Skewness and Kurtosis

Skewness is a measure of the symmetry of the distribution. If the skewness is > 0, it is indicative of a positively skewed distribution which has a long tail towards the high values. If the skewness is < 0, it is reflecting a negatively

skewed distribution that has a tail toward the lower values. Symmetrical distributions have a skewness = 0.

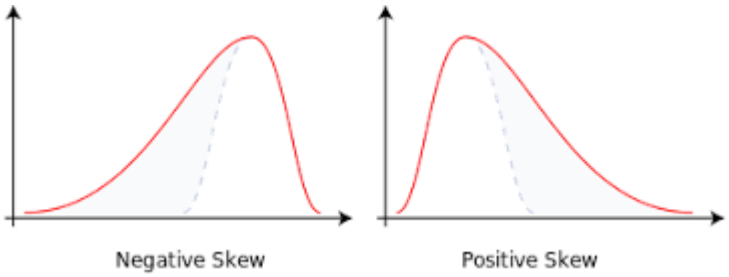

Figure 19. Skewed Data distribution types.
(Source: https://en.wikiversity.org/wiki/Skewness)

Kurtosis is a statistical measure that describes the peakedness of the data. Kurtosis > 3 is described as leptokurtic with a thin high peak and thicker tails. If kurtosis is < 3 has a platykurtic distribution which is broad and flat with thinner tails. The kurtosis similar to a normal distribution is = 3 and termed mesokurtic.

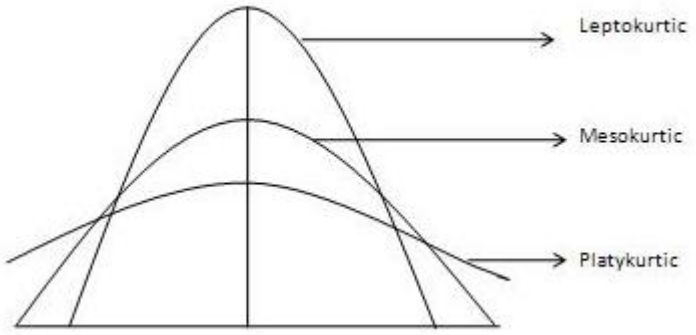

Figure 20. Types of kurtosis.
(Source: https://www.tutorialspoint.com/statistics/kurtosis.htm)

Correlation Coefficient

Multivariate data analysis is conducted to compare variables or elements against each other by means of XY scatterplots. The intensity of the linear relationship between two variables is expressed by the correlation coefficient (r). The closer the correlation coefficient it to +1 or -1 the higher the intensity of the linear relationship between the two variables of interest, with $0 < r \leq 1$ indicating a positive relationship and $-1 \leq r < 0$ indicating

a negative relationship. A correlation coefficient tending to 0 indicates no linear relationship.

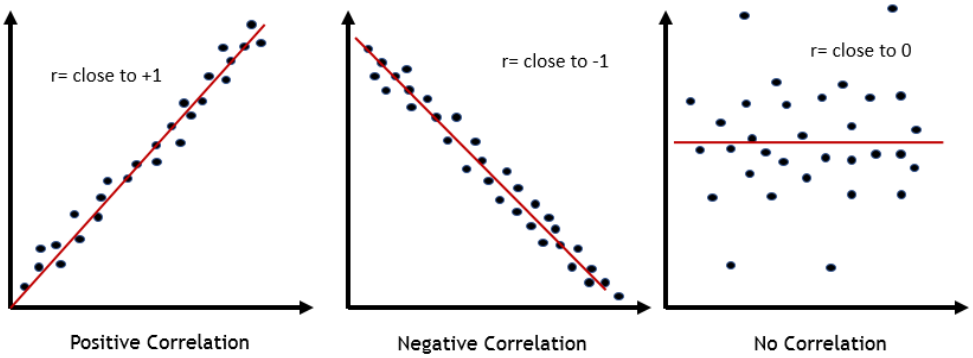

Figure 21. Correlation coefficient relationships of multivariate data analysis

3.3. Variography

The research study applies geostatistics to calculate the estimation variance as a determinant for the best estimate from various sampling designs. The estimation variance that is calculated depends on the variography. Spatial continuity of the grade is defined by the variogram model which is used in geostatistics to quantify how samples relate to each other in space according to the distance and orientation (Coombes, 2008). Therefore, the next step in the process is to conduct variography on the validated sample dataset per domain. Once the variogram is defined, possible sample layout patterns can be evaluated to obtain the optimal sampling option.

In order to apply geostatistics, the following is additionally assumed and inferred (Clark and Harper, 2000):

- The samples measured are from a homogenous population and the characteristics at sampled locations are the same at the unsampled locations.
- The values at the unsampled location are related to the values at the sampled location.

Samples of a mineralised deposit are termed regionalised variables that are spatially distributed. The correlation of regionalised variables are separated by a distance have a structure that is applied in geostatistics. The aspect of both the local randomness of the of two independent grade values and their

spatial correlation is expressed by a random function (Journel and Huijbregts, 1978). For this to apply, the samples or regionalised variables must be defined according to a homogenous zone or domain (Armstrong, 1998). The expected value of the mean and the variogram are not changing between locations within the domain (Ronald, 2016).

A semi-variogram that characterises the spatial continuity of regionalised variable sample data is expressed by the function (Journel and Huijbregts, 1978):

$$\gamma(h) = \frac{1}{2N_h}\sum_{i=1}^{N_h}[Z(x_i + h) - Z(x_i)]^2 \qquad \text{Equation 6}$$

The semi-variogram is half the averaged squared differences between Z(x) and Z(x+h) at any two points separated by the distance and direction vector (h) where N(h) is the number of pairs.

Semi-variograms plots were modelled in Snowden Supervisor software with available drill hole data for each domain.

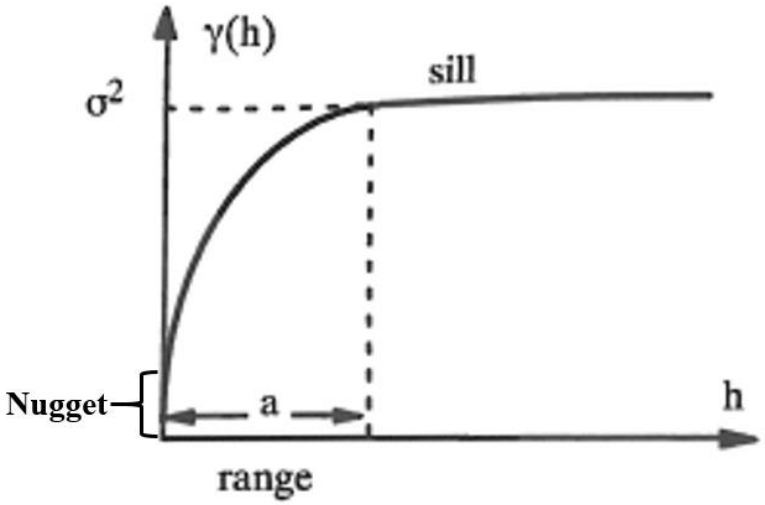

Figure 22. Plot of a Semi-Variogram (Armstrong, 1998).

Attributes of a modelled variogram include the range, sill and nugget. The range value from the modelled semi-variogram is the distance beyond which samples are no longer spatially correlated (Coombes, 2008). The structure up to the range defines the continuity between samples that are spatially

correlated. Coombes (2008) describes the sill is the population variance(σ^2), which is that point beyond which the sample grade difference is not dependent on their separation distance. Refer to Figure 22. The nugget is the inherent difference between two closest spaced samples that are taken adjacent to each other. The nugget is typically calculated from a downhole variogram however in this 2D PGE environment it is determined from the directional variograms of closely spaced samples. In many geological ore bodies, the continuity of mineralisation in the deposit can be different along different directions which is known as anisotropy (Kushavand et al., 2007; Coombes, 2008). To account for directional spatial variability, model variograms can be modelled

for the different directions. Mineralisation in an orebody with no preferred directional continuity is referred to as isotropic, and therefore a single variogram is modelled for the element.

3.4. Kriging Variance

Kriging is defined by Armstrong (1998) as an estimation method that gives the best unbiased linear estimate with minimum variance. This geostatistical estimation technique termed kriging originated from work by Krige in 1951 and 1952 in the Witwatersrand gold mines (Krige and Kleingeld, 2005; Krige, 1999). Krige discusses the issue of conditional bias whereby high grades are overestimated and low grades underestimated. Georges Matheron was cited by Krige (1999) for the derivation of an estimator of weighted moving averages of the data which are linear combinations of the data formulated as:

$$Z^*(v) = \sum \lambda_i Z(X_i) \qquad \text{Equation 7}$$

The estimate Z*(v) is the estimate at location v and is the sum of the regionalised stationary variable values $Z(X_i)$ which are weighted. The λi are the weights providing the unbiased estimate. Kriging applies the spatial variability component by introducing the semi-variogram and a Lagrange multiplier to ensure unbiasedness in the estimate. The weighting factor allows for more reliance on samples that are closer in comparison to distant

samples (Armstrong, 1998). Kriging provides a measure of confidence on the estimation error in the estimate known as the kriging variance which are designed to a minimum variance (Coombes, 2008; Armstrong, 1998) to give the best estimate. The estimation variance is given by

$$\sigma_{\epsilon}^{2} = Var[(Z - Z(v)^{*})^{2}] \quad \text{Equation 8}$$

The kriging variance is independent of the data values and is only dependent on the semi variogram model and the data configuration in relation to the location to be estimated. Therefore, as described by Clark and Harper (2000) the confidence depends on the relationship between each sample value and the unknown target block (V) to be estimated and the relationship between all possible samples (x_i) including itself. The kriging variance is also a function of the optimum sample weights and the Lagrange multiplier (μ) and is expressed as follows:

$$\sigma_{K}^{2} = \sum_{i} \lambda_i \bar{\gamma}(x_i, V) - \bar{\gamma}(V, V) + \mu \quad \text{Equation 9}$$

Various drill spacing configurations were tested using the concept of the drill hole spacing analysis, mentioned in Section 2.5, whereby the calculated kriging variances are used as an indicator of the best estimate (Bertoli et al., 2013 and Williams et al. 2015).

3.5. Cost Comparison

Further analysis of the all the sampling designs will be based on the additional criteria of costing. The purpose of this additional consideration assists with the selection of a grid option that would be deemed optimal and weighs the cost benefit of the sampling strategy suggested from the geostatistical results. The sampling process cannot be solely based on the outcomes of geostatistical work and independent of costing. An optimal strategy includes an understanding of the potential financial commitment required.

3.6. Methodology Summary

Firstly, the mechanised mining layout for Styldrift I was studied in order to gain an understanding into the mining production method which included an

underground visit and observation of the physical sampling process. This information was used to design the sampling grid patterns taking the mining layout into account to ensure practicality.

Secondly, exploratory data analysis was completed on existing available sampling data in order to validate and check the representation for its use in this study. Classical statistical methods were employed to confirm the definition of the local facies types and understand the variability to domain areas of similarity to achieve stationarity.

The continuity of these facies domains were characterised by modelling this spatial continuity relationship with variography which are used in the geostatistical evaluation. Estimations were run on the various grid spacings with the kriging variance calculated and analysed as a measure of confidence into the optimal option analysis.

Lastly, an implicit cost comparison, based on the number of samples identified for the various sampling strategies, was reviewed to understand the cost benefit of each sampling grid. This approach assisted in concluding the final recommendation on the implementation of the optimal sampling strategy that met technical, practical and financial requirements for Styldrift I.

4. RESEARCH RESULTS AND DISCUSSION

Results on the data analysis and outcomes of the geostatistical evaluation are reported and discussed in this section of the report. The reported data analysis includes the investigation of the statistical data and facies delineation for domaining. The results for geostatistical evaluation of the various sample spacings is detailed and described for option analysis.

4.1. QAQC Accuracy Results of Sample Data

The dataset used in this study for domain verification and conducting variography for spatial continuity analysis have been analysed for accuracy and precision to verify the reliability of the data. African Mineral Standards

(AMIS) CRMs or standards, which are internationally ISO17034 accredited were inserted into the sample batches and analysed separately per assay standard and element according to their unique known certified values. The standard tolerance limit is 3 times the certified standard deviation (σ), which is added to the certified mean value to calculate the upper limit and subtracted to calculate the lower limit. The tolerance limit for a blank is 10 times the detection limit. This is the accepted range for results. All values that fall outside of this range are outliers. The assay values reported by laboratory for these known values are evaluated according to these limits by means of control charts which can be found in the Appendix of this report. Control charts depict how the standard or blank plots in reference to the certified mean, upper and lower limits. The analysis of the CRMs with large outlier percentages of greater than 10 % would be of concern. The AMIS produced certified reference materials indicate a high level of accuracy with low outliers and therefore the data is accepted as accurate.

4.2. QAQC Precision Results of Drill hole Data

The performance of precision testing by twin stream analysis is tabled below per element with scatter plot diagrams from Figure 23 to Figure 26 depicting the correlation between the two legs for Platinum, Palladium Rhodium and Gold.

Leg 1 versus Leg 2 for all elements have very high correlation coefficient values either equal to or close to 1. For all elements more than 80 % of the data lies within a 10 % precision.

Table 1. Precision results analysed for drill hole data per prill element

Element	N pairs	Leg 1 Average	Leg 2 Average	Average % Relative Difference	R Value	% Outliers	% Data within 10% Precision
Pt	5495	2.59	2.60	-0.94	0.99	0.08	91.87
Pd	4705	1.62	1.63	-0.71	1.00	0.06	91.97
Rh	2296	0.64	0.64	-0.51	0.99	0.00	92.33
Au	1473	0.32	0.32	1.28	0.95	0.00	82.28

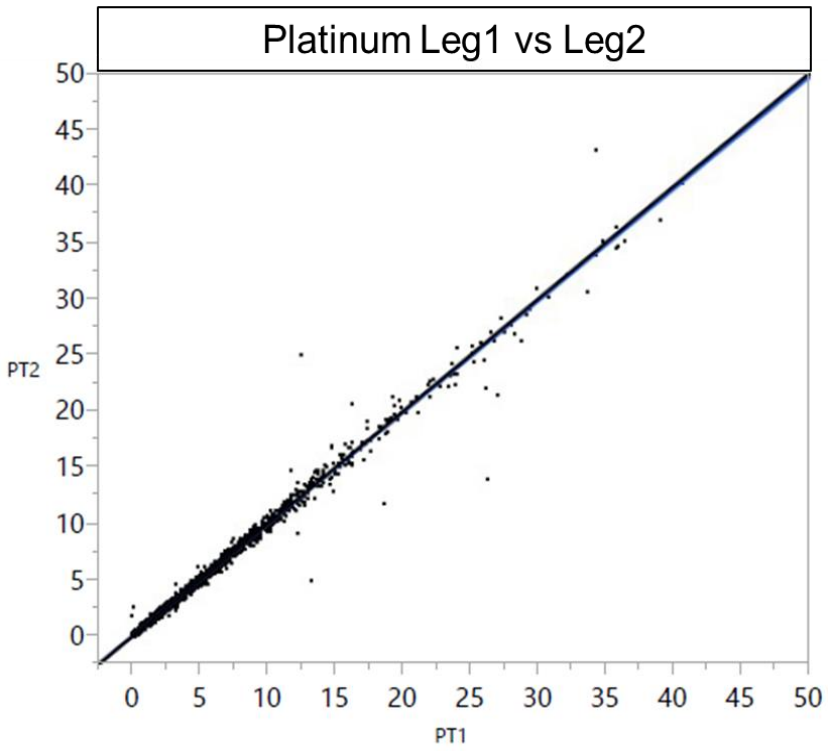

Figure 23. XY Scatterplot of Pt (g/t) twin stream comparison

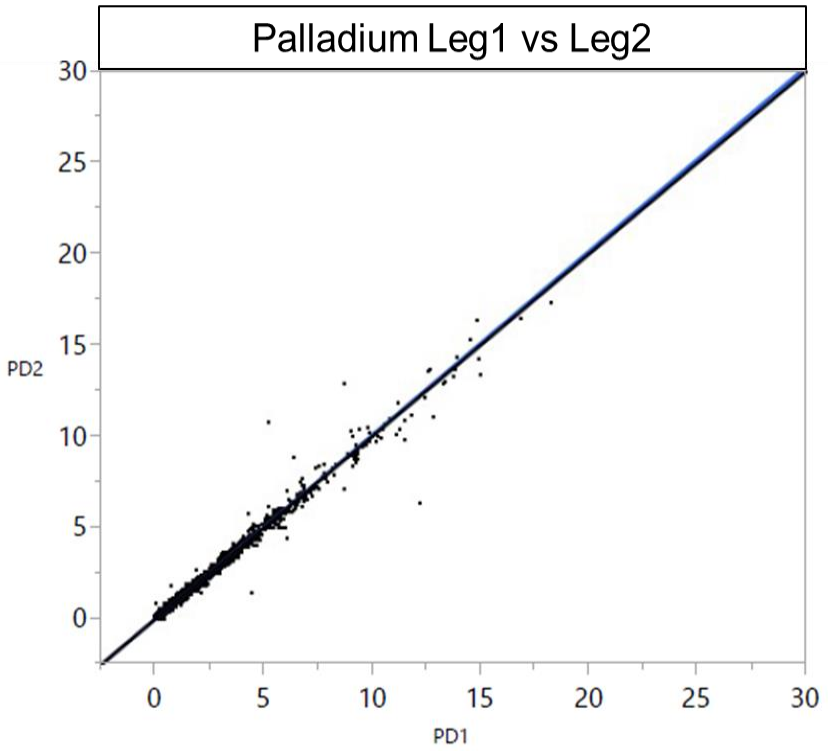

Figure 24. XY Scatterplot of Pd (g/t) twin stream comparison

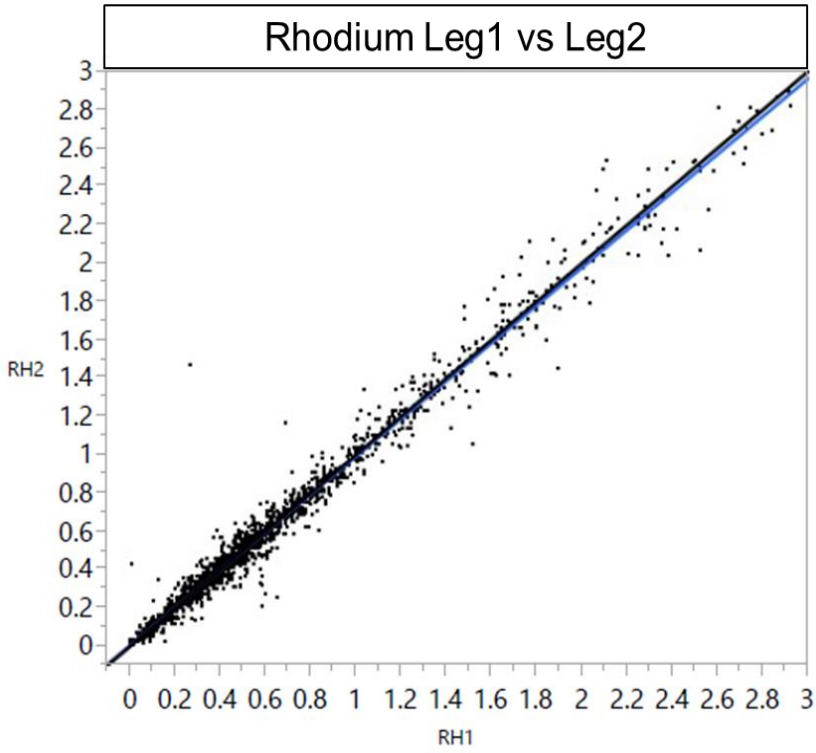

Figure 25. XY Scatterplot of Rh (g/t) twin stream comparison

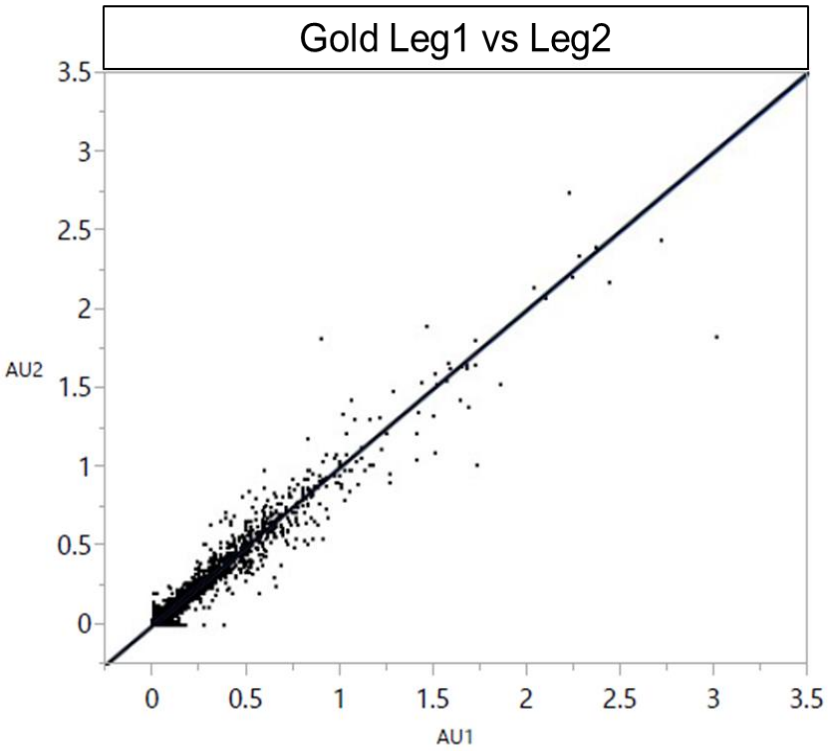

Figure 26. XY Scatterplot of Au (g/t) twin stream comparison

The twin stream results for all the elements produced 91.87 %, 91.97 %, 92.33 % and 82.28 % of the data for Pt, Pd, Rh and Au respectively, within

a 10 % precision and very little outliers. These precision results are excellent with data well within the ideal range of over 80% requirement within 10 % precision and less than 5 % outliers.

It can therefore be concluded from the QAQC analysis that the data is both precise and accurate and acceptable for use in the estimate.

4.3. Prill Relationships and Regression

Rhodium (Rh) is not assayed for and is determined when the sum of the 3E (Pt, Pd and Au) is less than 1.5 g/t. Rhodium relationships with the other individual PGE's were reviewed in order to obtain realistic rhodium estimates to add to those prills without a rhodium value for a 4E sum calculation. Rhodium to platinum showed a good correlation of 0.838 (r), refer to Figure 27 below. The formula for the regression is Rh (g/t) = 0.0736 x Pt (g/t) + 0.0123 which was obtained from the correlation of 723 samples and applied to calculate the missing Rh (g/t) grade values for 268 samples.

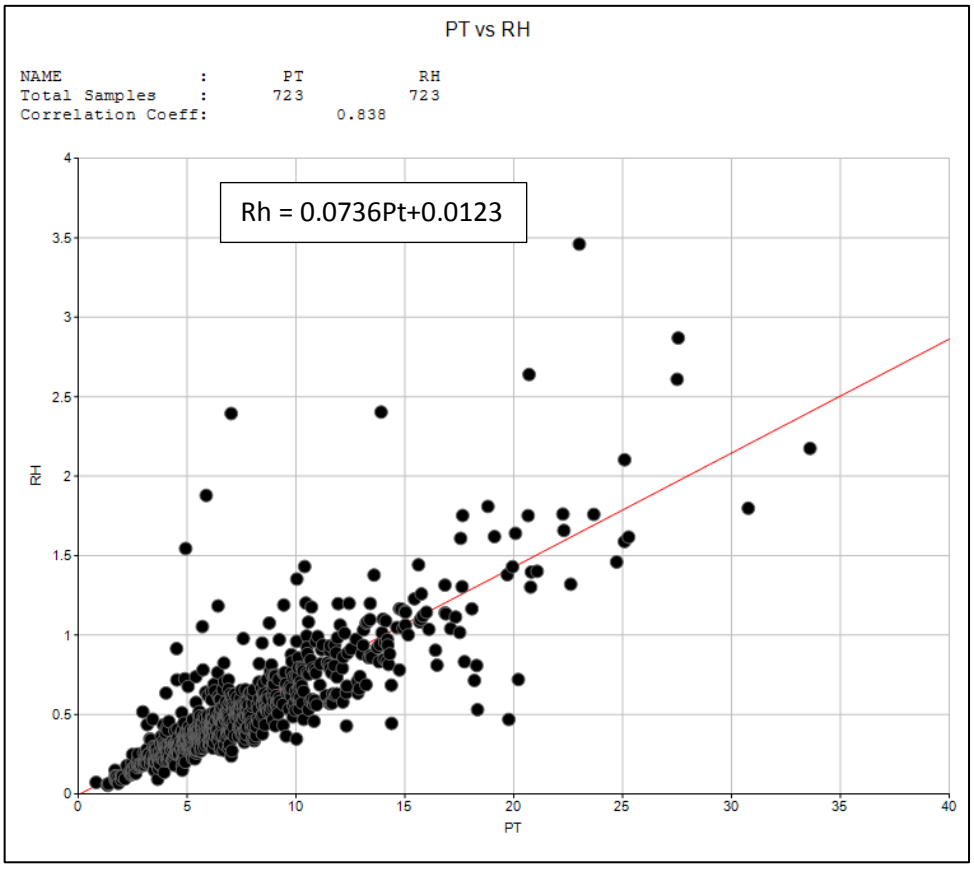

Figure 27. Platinum versus Rhodium scatterplot of the Merensky Reef

4.4. Drill hole Data Statistics Investigation and Domain Inspection

The use of geostatistics for sampling design in this study is dependent on the variogram and therefore stationarity of the grade element or variables is a requirement. According to the recommendation of Atkinson and Llyod (2007) that a global design would be sub-optimal and therefore suggests segmentation, local facies domains are checked ensuring continuity and sample correlation. Ronald (2016) explains the robustness of a domain is tested through exploratory data analysis to establish stationarity and representation of a single population.

A total of number of 729 drill hole sampled data was extracted for investigation and examination of the delineated local facies types to apply as domains for use in modelling the variography and to produce the estimation results for a sampling grid for each area. The distribution of 525 and 204 intersections within the Central Reef facies and the Normal Reef facies, respectively, is displayed in Figure 28.

All available sampling data that did not pass inspection as per the validations outlined in Section 3.2.1 were excluded from use in the study. Composite grade plots and constructed histograms per domain (Figure 29 to Figure 46) of grade values for the 4E PGE grade and each individual prill (Pt, Pd, Rh and Au) is presented for the statistical analysis of the data.

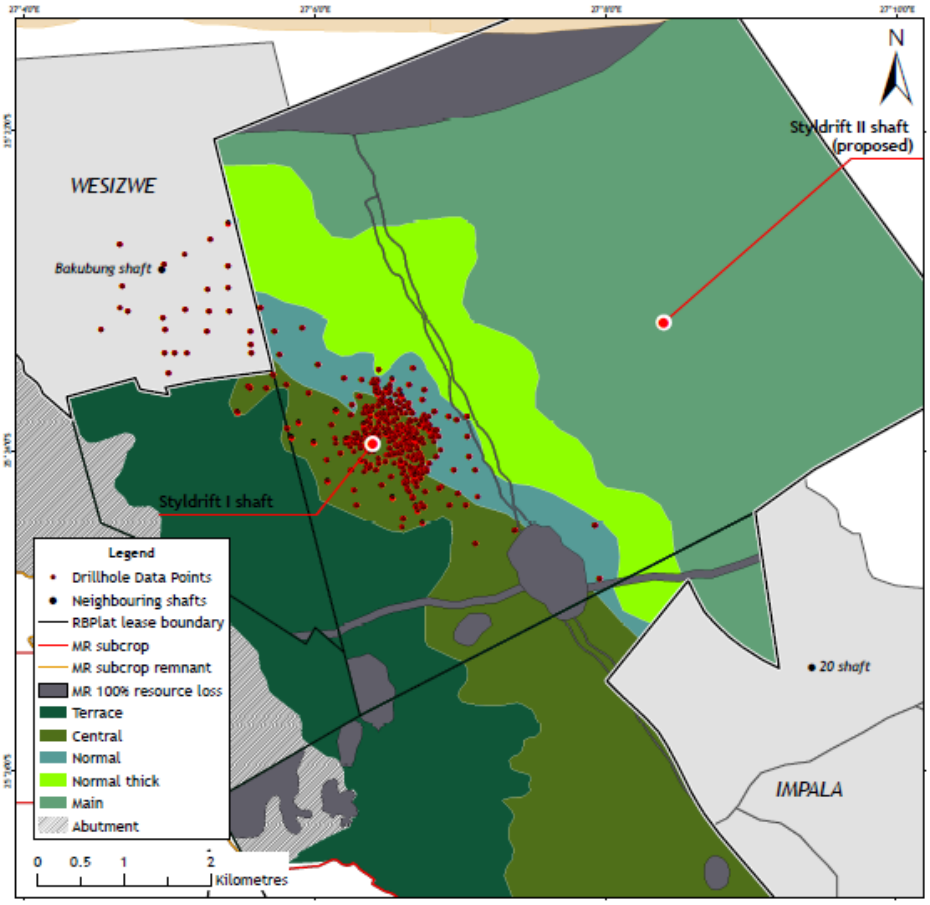

Figure 28. Available drill hole data used for domain analysis and variography

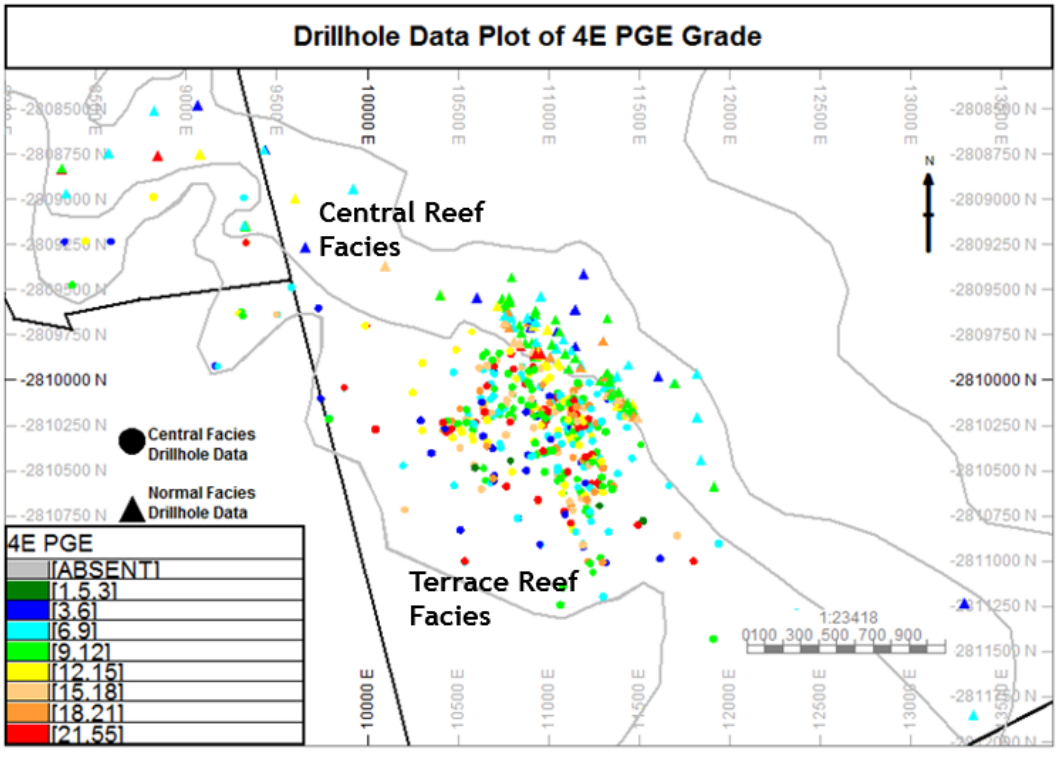

Figure 29 4E PGE grade (g/t) of drill hole intersections by reef facies types

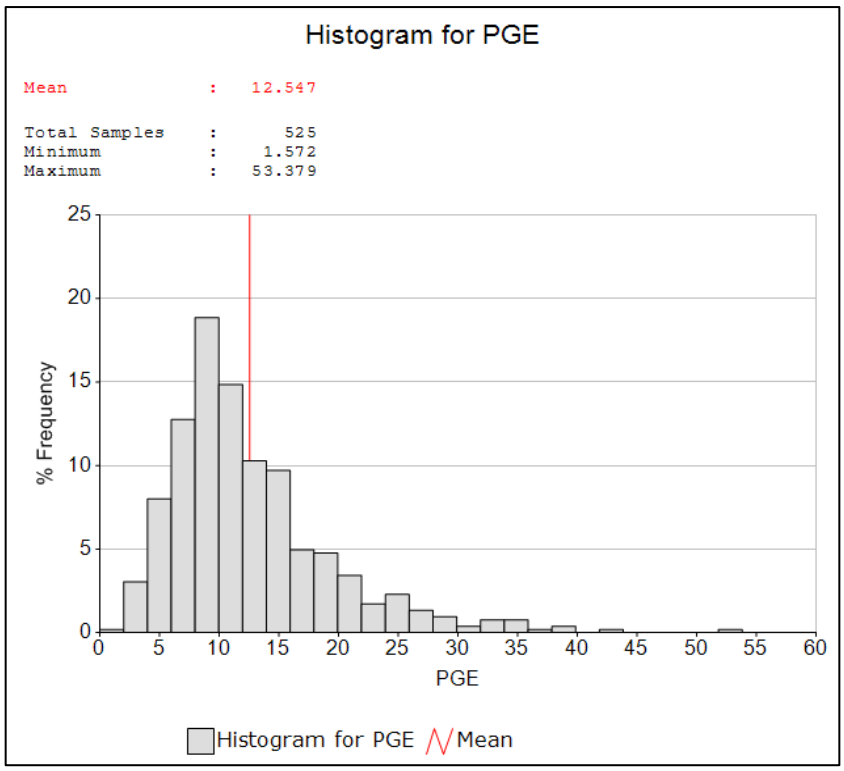

Figure 30. Histogram of the 4E grade (g/t) for the Central Reef Facies

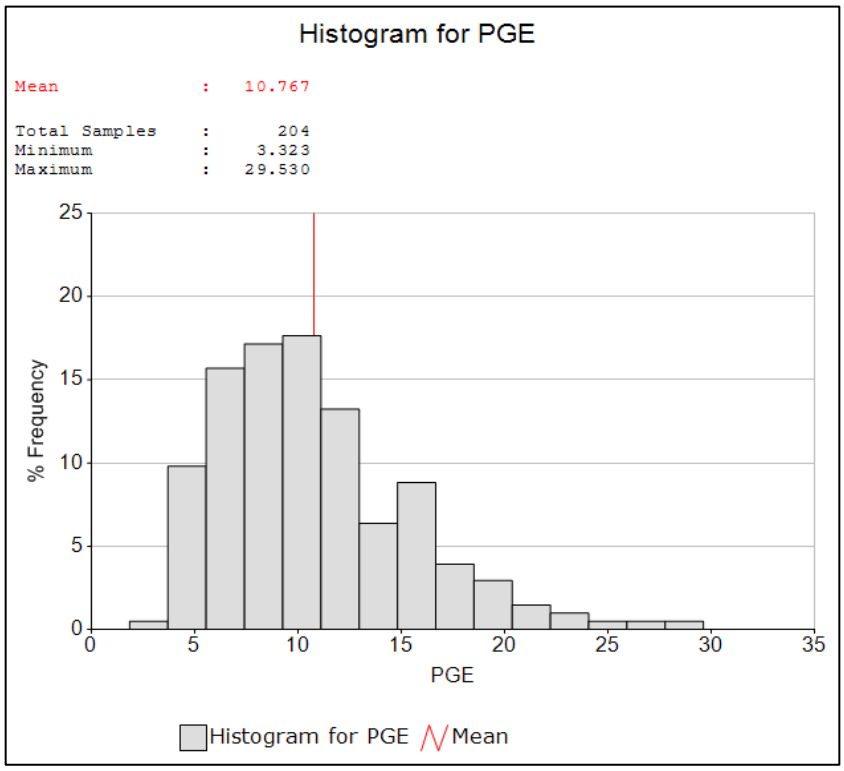

Figure 31. Histogram of the 4E grade (g/t) for the Normal Reef Facies

Table 2. Tabulation of the 4E PGE Grade (g/t) statistics per facies

4E PGE (g/t)	Facies Type	
Statistic	Central Reef	Normal Reef
Number of SAMPLES	525	204
MINIMUM	1.57	3.32
MAXIMUM	53.38	29.53
RANGE	51.81	26.21
MEAN	12.55	10.77
VARIANCE	47.77	21.94
STANDDEV	6.91	4.68
SKEWNESS	1.61	1.06
KURTOSIS	3.95	1.31

The Central Reef facies has a higher mean grade value of 12.55 g/t for the 4E PGE grade compared to the 10.77 g/t for the Normal Reef facies. Minimum and maximum values also have a larger range for the Central Reef facies as well as greater variability with a high standard deviation of 6.91g/t. Both domains display positively skewed distributions however Normal Reef facies displays a flatter platykurtic profile compared to the Central Reef facies leptokurtic higher and thinner peak.

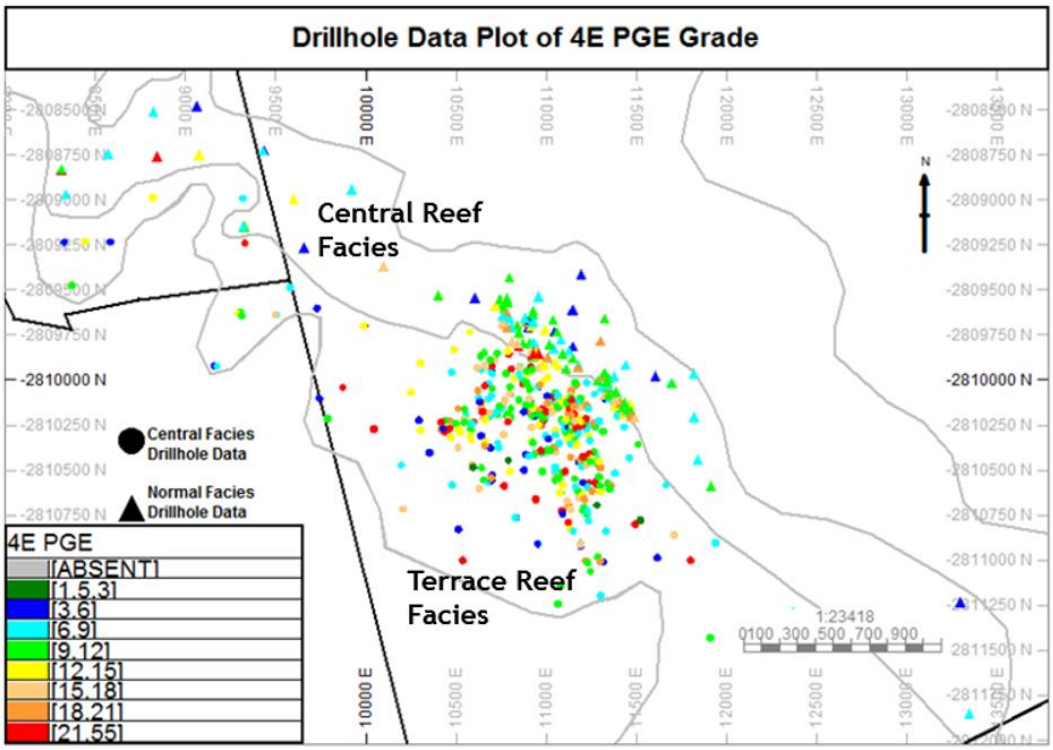

Figure 32. Pt Grade (g/t) of drill hole intersections by reef facies types

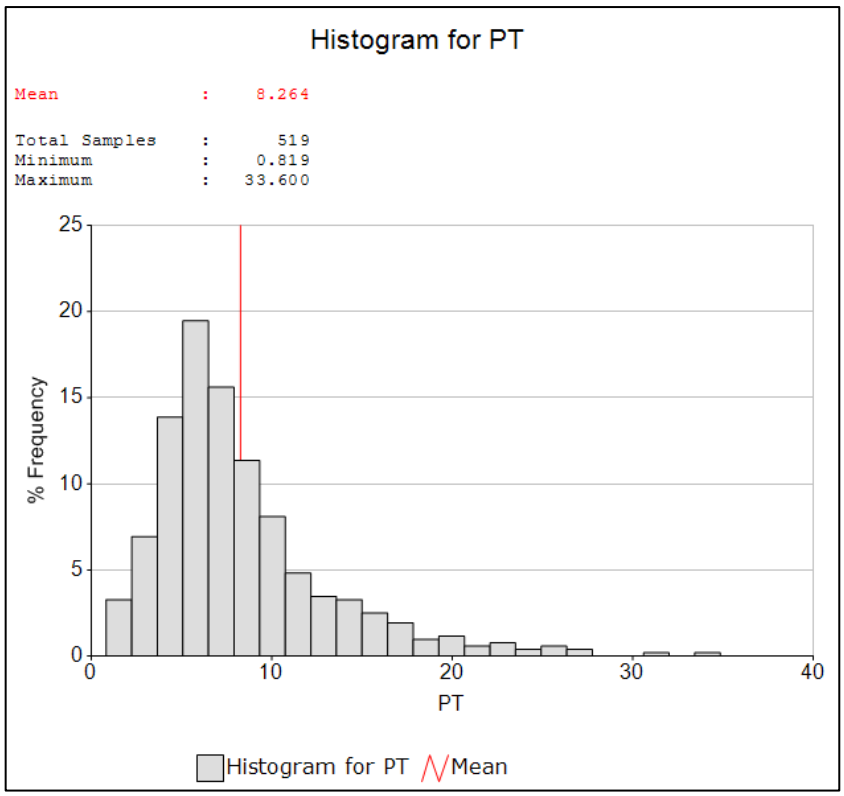

Figure 33. Histogram of the Pt Grade (g/t) for the Central Reef Facies

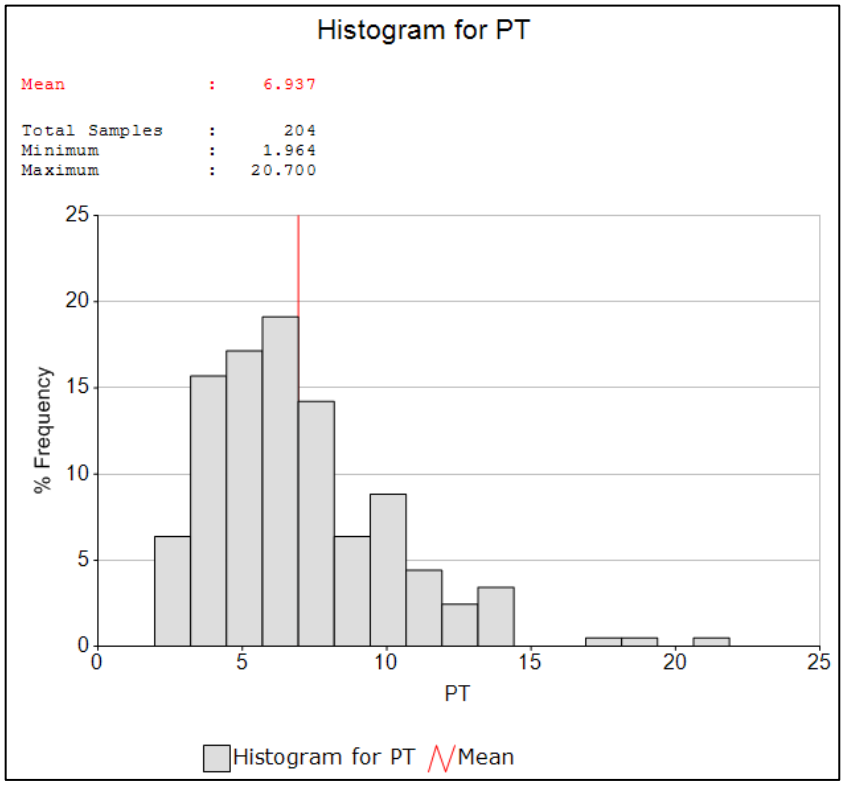

Figure 34. Histogram of the Pt Grade (g/t) for the Normal Reef Facies

Table 3. Tabulation of the Pt Grade (g/t) statistics per facies

Pt (g/t)	Facies Type	
Statistic	Central Reef	Normal Reef
Number of SAMPLES	519	204
MINIMUM	0.82	1.96
MAXIMUM	33.60	20.70
RANGE	32.78	18.74
MEAN	8.26	6.94
VARIANCE	24.00	9.91
STANDDEV	4.90	3.15
SKEWNESS	1.64	1.21
KURTOSIS	3.54	1.98

A slightly higher mean and standard deviation value is calculated for Central Reef facies with a larger range of platinum values. Both domains have skewness values greater than 1 displaying their positively skewed grade distributions in Figure 33 and Figure 34. The kurtosis of platinum for the Central Reef facies is 3.54 with a high peak and thin long tail (Figure 33). Normal Reef facies kurtosis is less than 3 which indicates a flatter grade distribution.

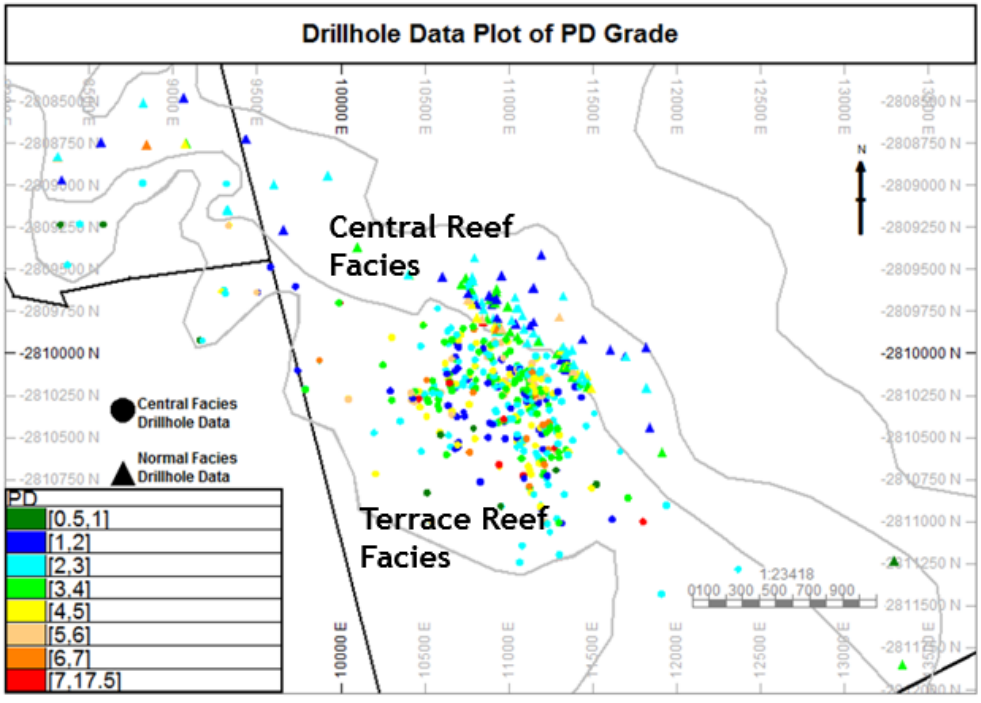

Figure 35. Pd Grade (g/t) of drill hole intersections by reef facies types

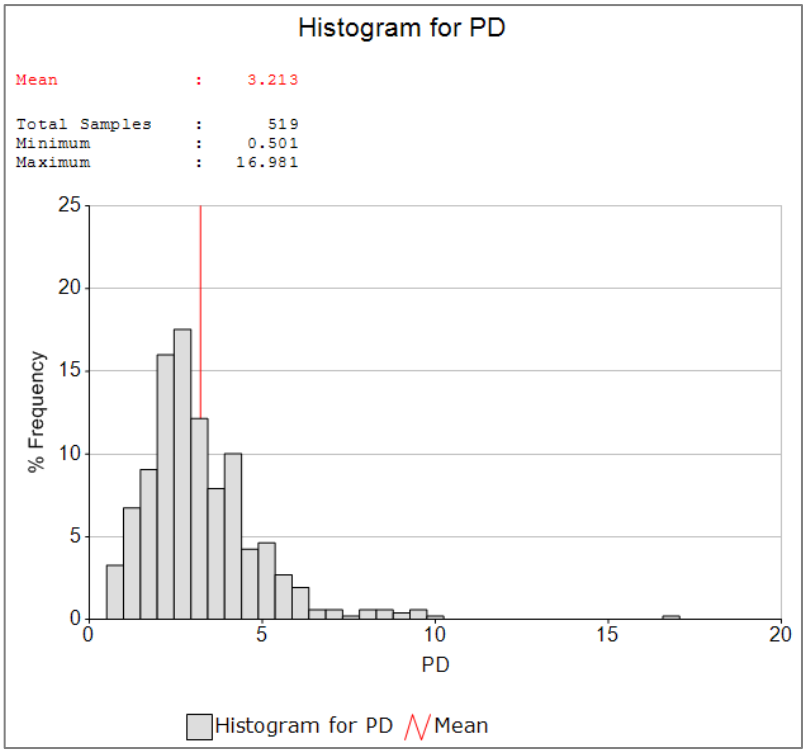

Figure 36. Histogram of the Pd Grade (g/t) for the Central Reef Facies

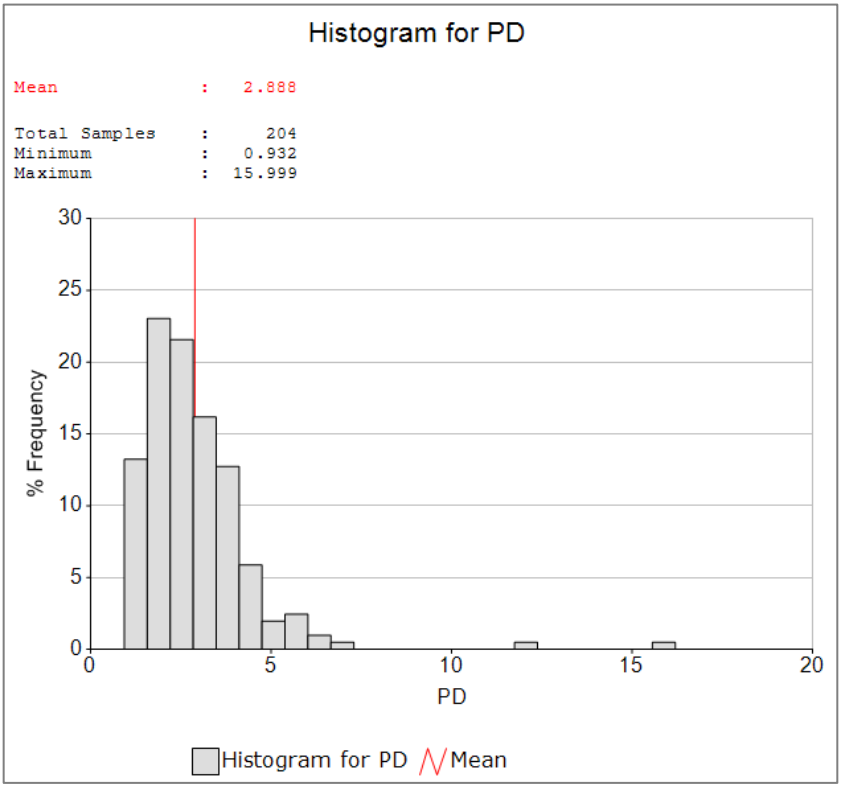

Figure 37. Histogram of the Pd Grade (g/t) for the Normal Reef Facies

Table 4. Tabulation of the Pd Grade (g/t) statistics per facies

Pd (g/t)	Facies Type	
Statistic	Central Reef	Normal Reef
Number of SAMPLES	519	204
MINIMUM	0.50	0.93
MAXIMUM	16.98	16.00
RANGE	16.48	15.07
MEAN	3.21	2.89
VARIANCE	2.83	2.59
STANDDEV	1.68	1.61
SKEWNESS	2.09	3.74
KURTOSIS	9.84	24.20

Palladium for both Central Reef facies and Normal Reef facies domains display similar ranges and mean values of 3.21 g/t and 2.89 g/t, respectively. Therefore, standard deviation values calculated as a measure of variance from the mean value is very close for both domains. Both have positively skewed distributions with leptokurtic profiles.

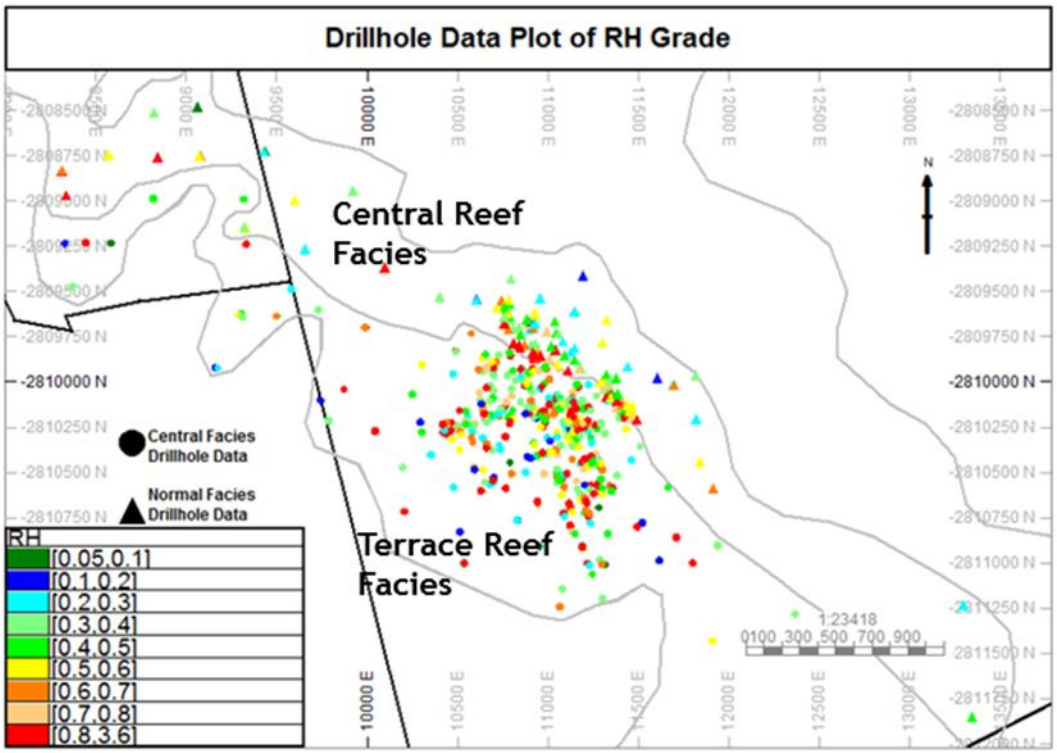

Figure 38. Rh Grade (g/t) of drill hole intersections by reef facies types

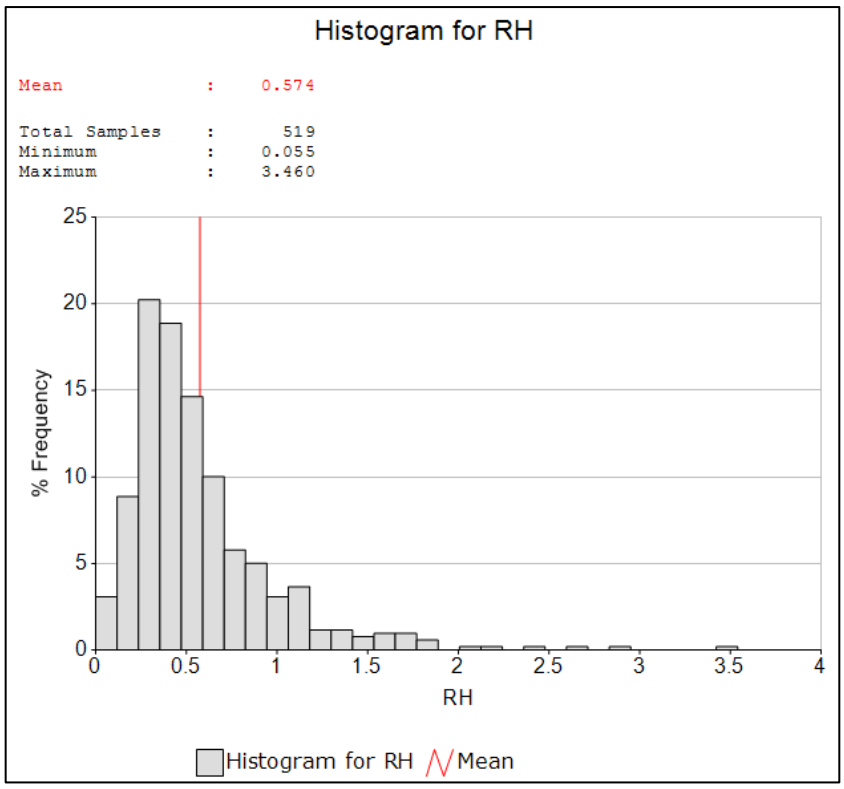

Figure 39. Histogram of the Rh Grade (g/t) for the Central Reef Facies

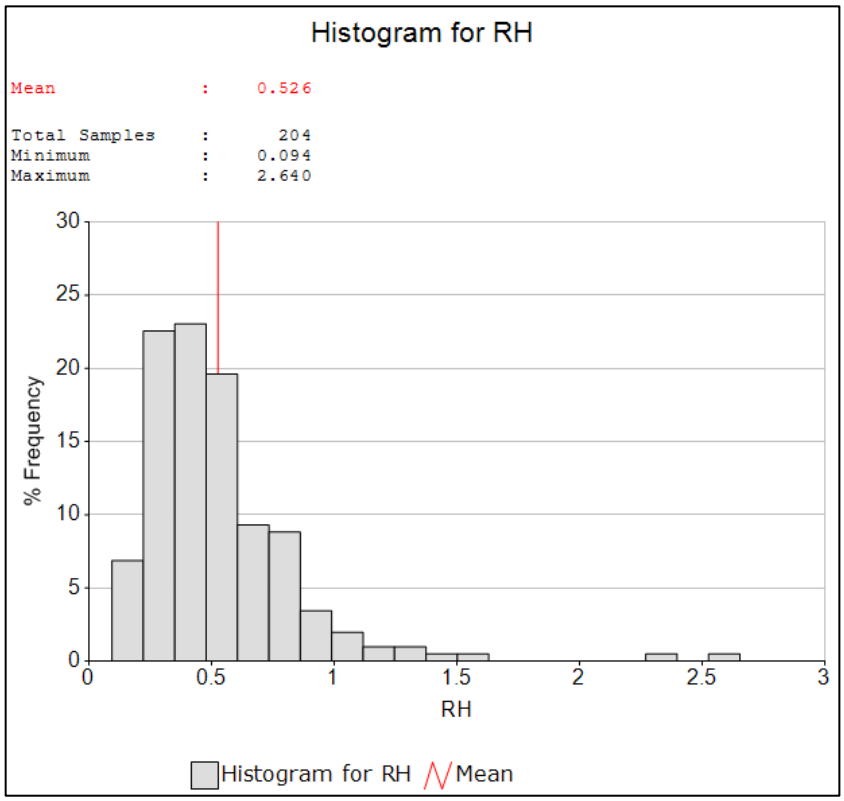

Figure 40. Histogram of the Rh Grade (g/t) for the Normal Reef Facies

Table 5. Tabulation of the Rh Grade (g/t) statistics per facies

Rh (g/t)	Facies Type	
Statistic	Central Reef	Normal Reef
Number of SAMPLES	519	204
MINIMUM	0.06	0.09
MAXIMUM	3.46	2.64
RANGE	3.41	2.55
MEAN	0.57	0.53
VARIANCE	0.17	0.10
STANDDEV	0.41	0.32
SKEWNESS	2.36	2.83
KURTOSIS	8.94	13.33

Rhodium for the Central Reef facies has higher maximum grade (g/t) compared to Normal Reef (Figure 40) but fairly close mean (μ) and standard deviation values. The rhodium grade distributions for the two respective reef facies, shown in Figure 39 and Figure 40 are positively skewed and are peaked to the left.

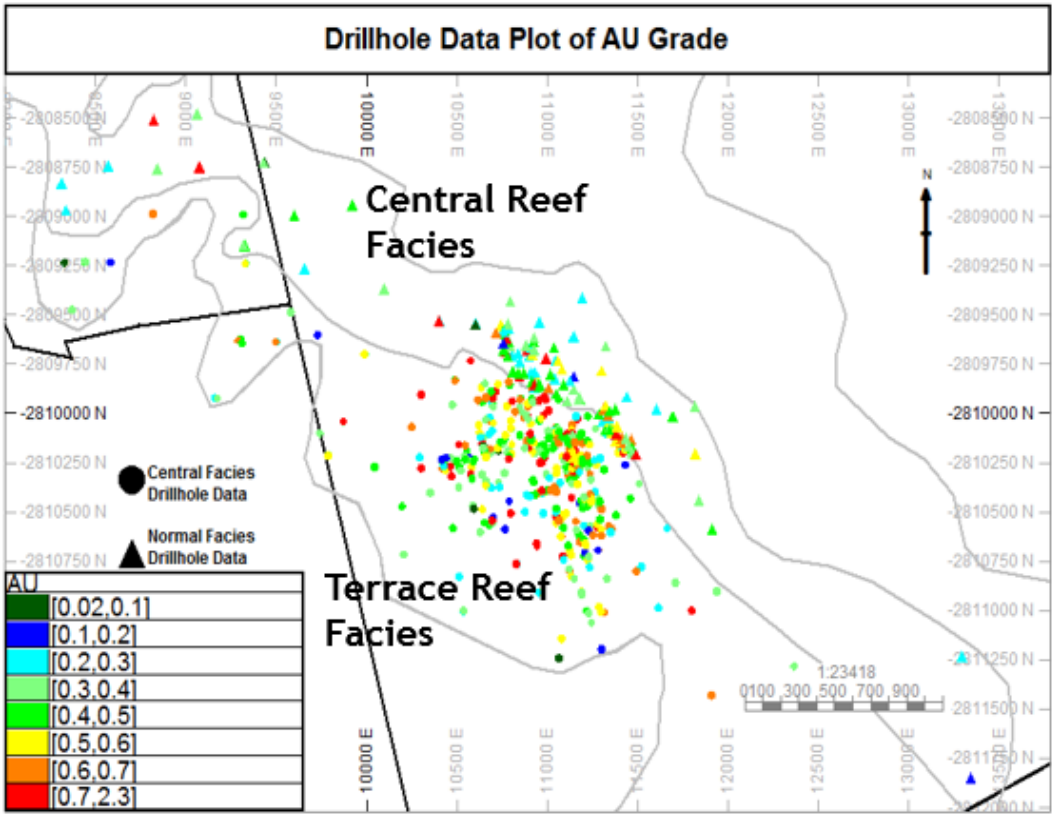

Figure 41. Au Grade (g/t) of drill hole intersections by reef facies types

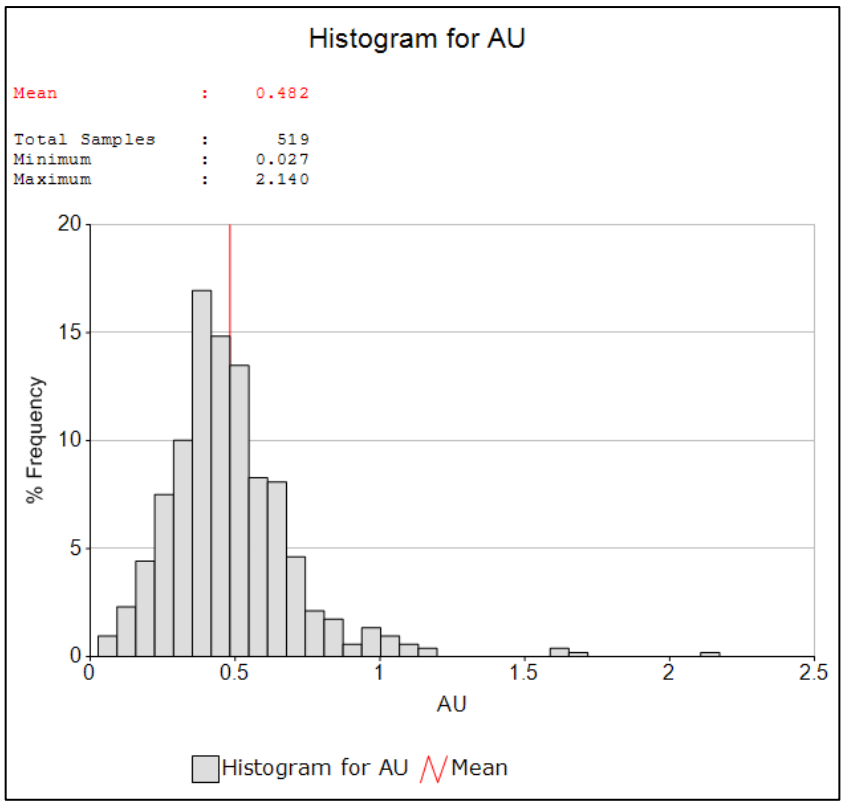

Figure 42. Histogram of the Au Grade (g/t) for the Central Reef Facies

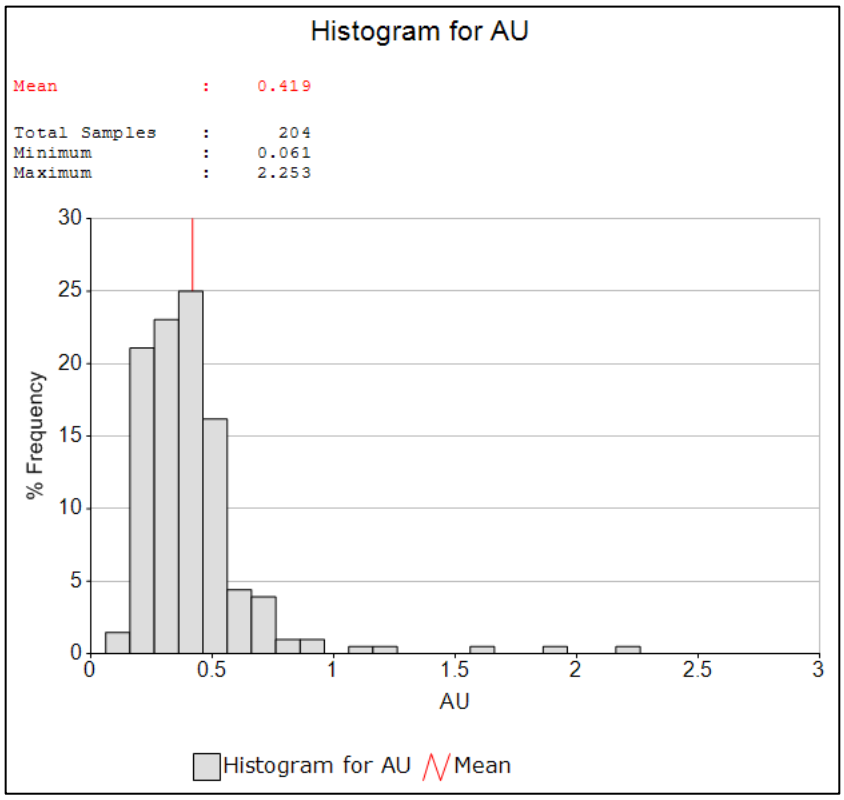

Figure 43. Histogram of the Au Grade (g/t) for the Normal Reef Facies

Table 6. Tabulation of the Au Grade (g/t) statistics per facies

Au (g/t)	Facies Type	
Statistic	Central Reef	Normal Reef
Number of SAMPLES	519	204
MINIMUM	0.03	0.06
MAXIMUM	2.14	2.25
RANGE	2.11	2.19
MEAN	0.48	0.42
VARIANCE	0.05	0.06
STANDDEV	0.22	0.25
SKEWNESS	1.98	3.65
KURTOSIS	9.22	19.84

Minimum and maximum range data for gold (g/t) are very close and comparable for both domains. Therefore the mean (μ) and standard deviation are similar. The distributions are positively skew with a kurtosis > 3. These peaked profiles are presented in Figure 42 and Figure 43.

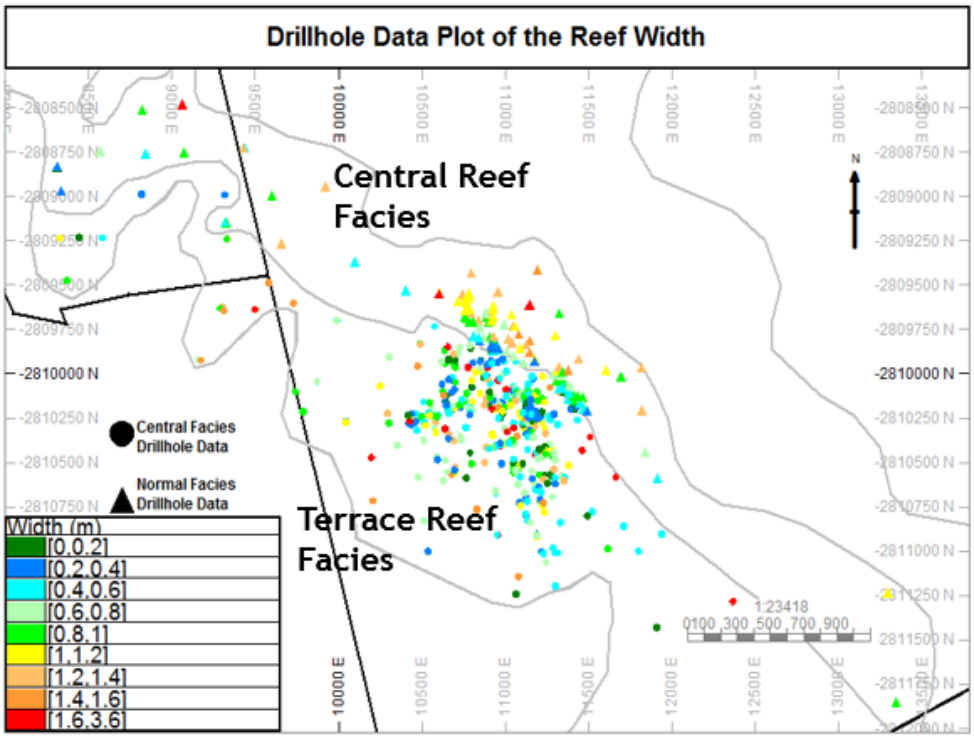

Figure 44. Reef width (m) of drill hole intersections by reef facies types

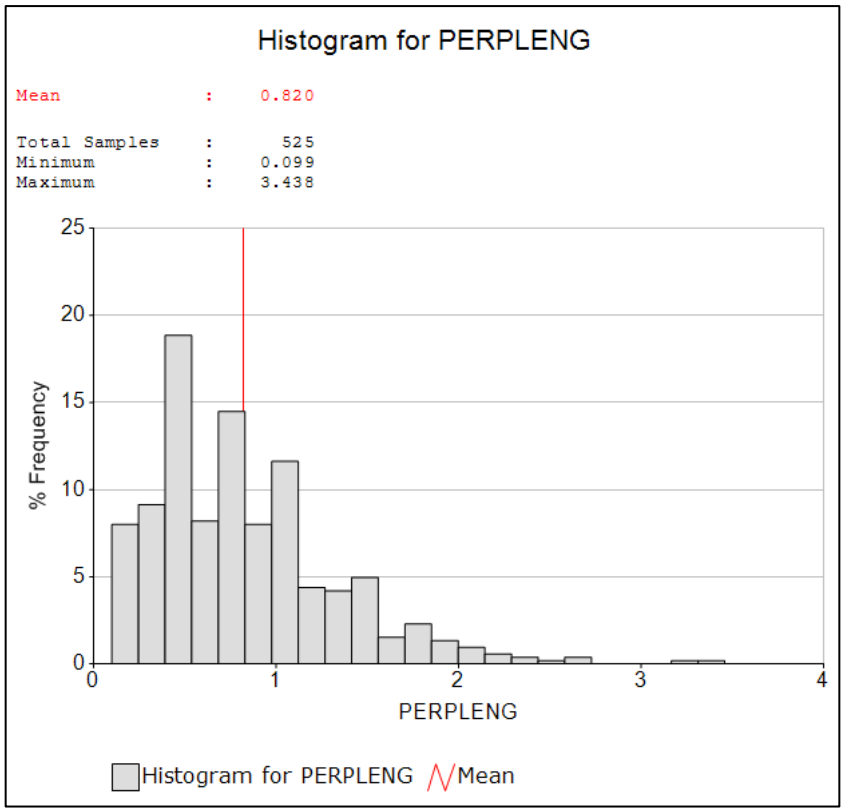

Figure 45. Histogram of the width (m) for the Central Reef Facies

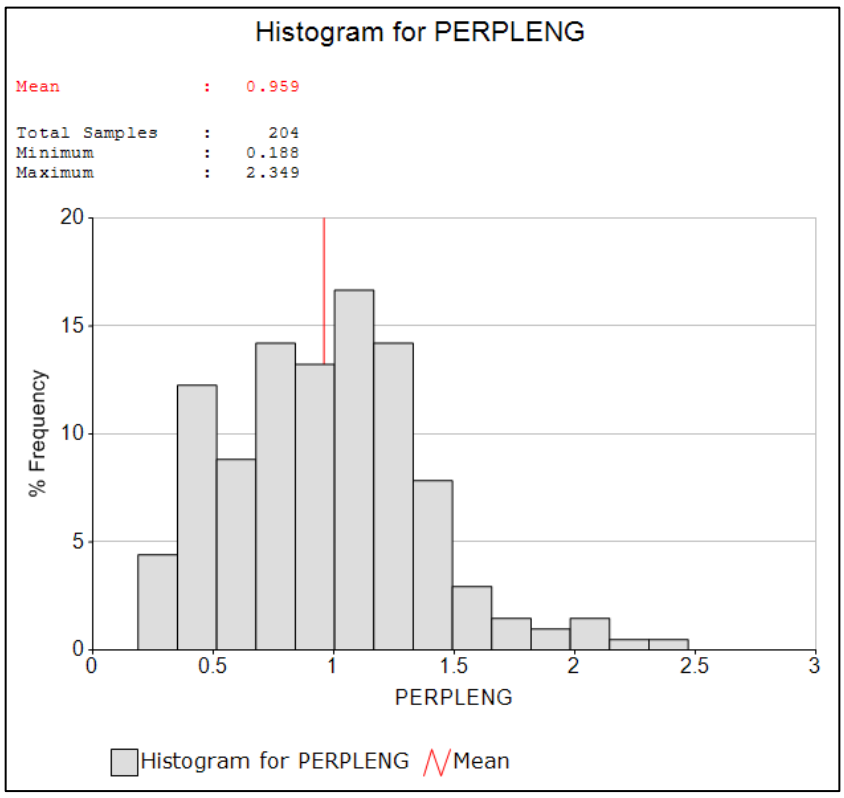

Figure 46. Histogram of the width (m) for the Normal Reef Facies

Table 7. Tabulation of the reef width (m) statistics per facies

Reef Width (m)	Facies Type	
Statistic	Central Reef	Normal Reef
Number of SAMPLES	525	204
MINIMUM	0.10	0.19
MAXIMUM	3.44	2.35
RANGE	3.34	2.16
MEAN	0.82	0.96
VARIANCE	0.26	0.17
STANDDEV	0.51	0.41
SKEWNESS	1.26	0.46
KURTOSIS	2.42	0.37

The perpendicular reef width (m) composites or true reef width composites calculated by correcting for dip during compositing is plotted is in Figure 44. Central Reef has lower and higher range of reef widths in comparison to the Normal Reef and a slightly lower mean (μ) value. Normal Reef is slightly positively skew with a value of 0.46 just greater than 0, the value for a symmetrical distribution. Kurtosis values indicate platykurtic profiles for both domains with Normal Reef displaying flatter than Central Reef in Figure 45 and Figure 46.

The exploratory data through this classical statistical analysis of the 4E PGE grade, individual prill splits (Pt, Pd, Rh and Au) and reef width confirms the distinction between the two facies types and supports the difference in the observed physical characteristics. Therefore, individual sampling patterns may be required for the two facies areas. Interrogation of the data per domain shows the data is well grouped and domaining is sufficient for use in the estimation meeting the stationary assumptions required for the variography mentioned in Section 3.3. The statistical properties are representative of the geological and physical properties and justify the domain definition according to the facies that were determined.

4.5. Variogram Contour Analysis and Modelling the Semi-variogram

Though the exploratory data analysis was done on all the individual prills (Platinum, palladium, rhodium and gold), only the 4E PGE grade (g/t) will

be used for the estimate, as this grade element is the sum of all the individual prills. 4E PGE grade (g/t) is commonly the variable used in the platinum industry. Variograms were created for the 4E PGE grade (g/t) per facies domain.

The lag spacing as described by Coombes (2002) is the incremental separation distance at which the variogram is calculated and should be equivalent to the sample spacing. This lag distance (h) is used to group or bin pairs of points. and is chosen with the consideration of the size of the domain. Figure 47 below demonstrates the parameters used to calculate the directional variogram. The azimuth defines the direction and it's tolerance is the limit for which points are considered. Similarly, the bandwidth limits the number of unrelated points.

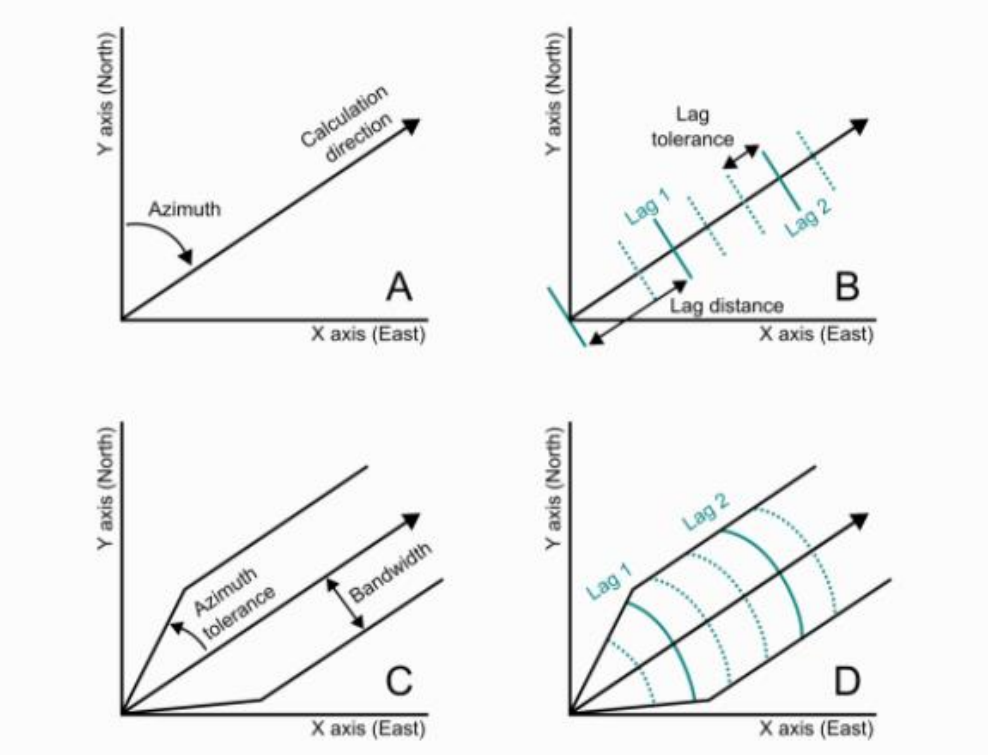

Figure 47.Diagram illustrating the variogram calculation parameters (Source: https://geostatisticslessons.com/lessons/variogramparameters)

The semi-variogram for both facies (Central Reef Facies and the Normal Reef Facies) were modelled in conjunction with the analysis of the variogram contour maps.

A tool used to determine directional anisotropies are contour maps of the variogram surface. The centre point of the map corresponds to the origin of the variogram (Kushavand et al., 2007). Variograms are plotted according to distance within a direction and the variogram values are then contoured. Refer to Figure 48 illustrating construction of a variogram map.

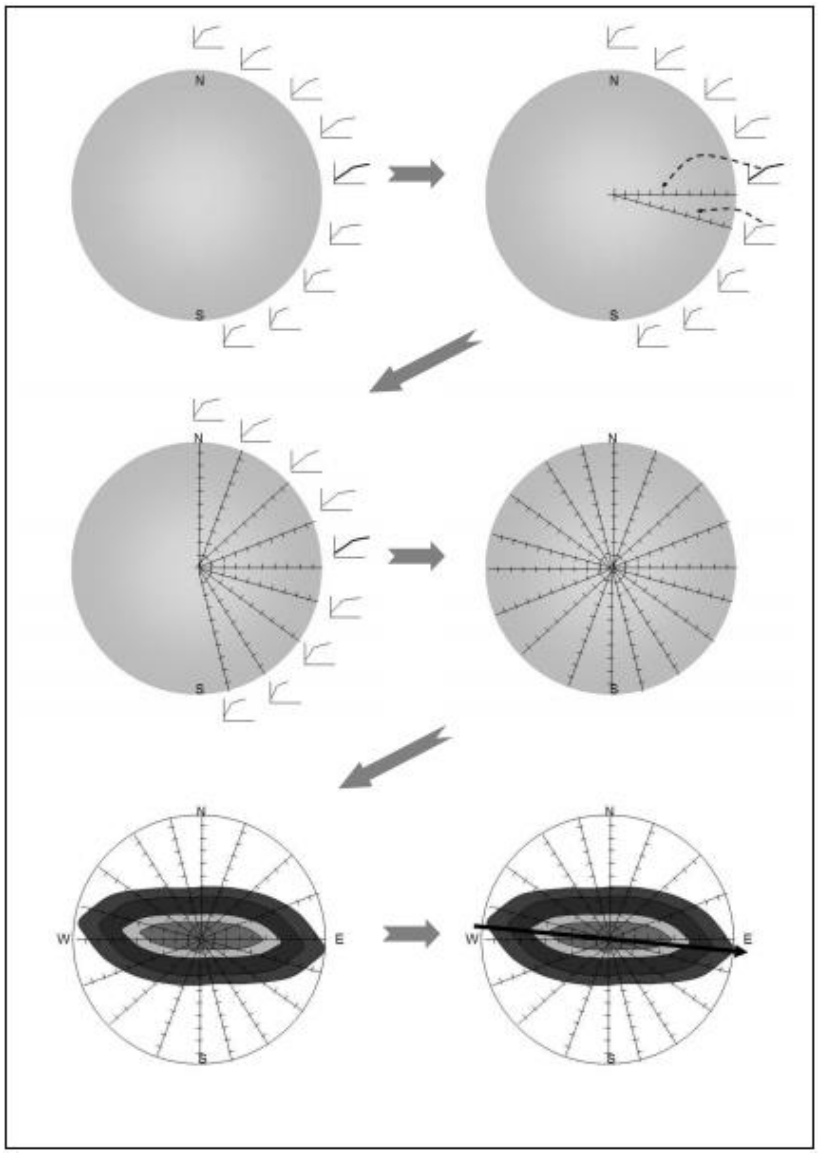

Figure 48. Construction of variogram contour maps, (Coombes, 2008)

Variogram contour maps were created within the Datamine software package for the 4E PGE grade (g/t) per facies domain. The variogram maps for the Central Reef Facies Domain appear in Figure 49 and that of the Normal Reef facies domain in Figure 50 below.

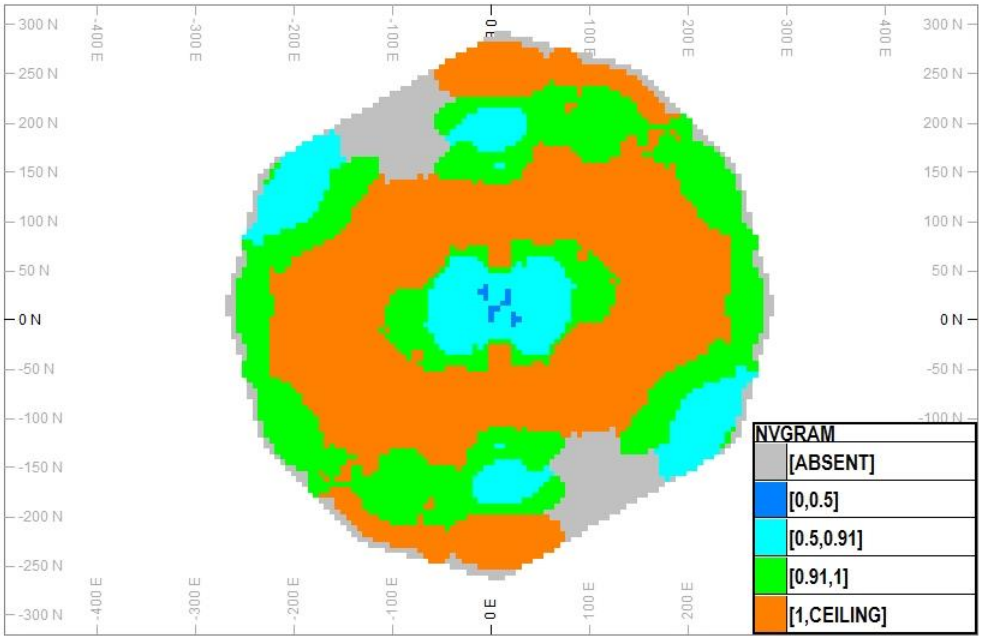

Figure 49. Variogram Contour map of the Central Reef Facies

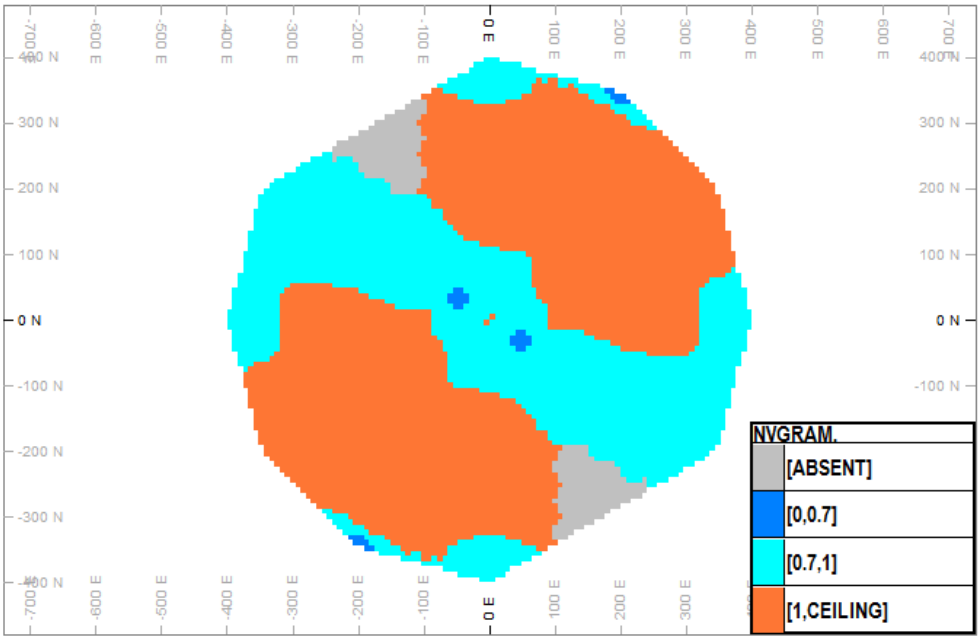

Figure 50. Variogram Contour map of the Normal Reef Facies

For both facies domains only a single omni-directional semi-variogram was modelled as there was no evidence of any strongly developed directional anisotropy. The sense of direction indicated in Figure 50 is an artefact of the spatial distribution of the data in the elongated narrow Normal Reef Facies domain that strikes in NW-SE direction, refer to Figure 11. With no spatial dependence changes in a specific direction, the variography is therefore

isotropic. Artefacts are one of the common seen problems when modelling semi-variograms (Armstrong, 1984).

A lag of 25 m was selected for the Central Reef Facies (Figure 51) due to the close sample spacing ranging between 30 m and 50 m and a 50 m lag applied for The Normal Reef Facies (Figure 52) that has sampling distanced 50 m to 200 m apart. The lag tolerance was set at half the lag spacing with 6 number of lags for Central Reef Facies domain and 8 number of lags for the Normal Reef Facies domain. The number of lags are determined by the data coverage within the volume of the domain.

The variograms were calculated in 5 directions from 0^0-180^0 with an azimuth increment of 30^0. The spatial relationship of 4E PGE grade (g/t) for the Central Reef Facies domain model (Figure 51) shows a 50 % nugget effect, reflected in dark blue on the variogram contour map (Figure 49). The semi-variogram model has a double structure with the first structure exhibiting 41 % of the variability in the first 33 m (light blue on the variogram contour map) and the last 10 % of the of the variability up to the 135 m range (green on the variogram contour map).

The orange contour shading depicts the high variability beyond this total sill or variance in Figure 49. The Normal Reef Facies semi-variogram (Figure 52) is modelled with a high 70 % nugget effect indicated in dark blue on the variogram contour map (Figure 50) and a single structure for the remaining 30 %, where the variogram reaches the sill at 200 m (light blue on the variogram contour map).

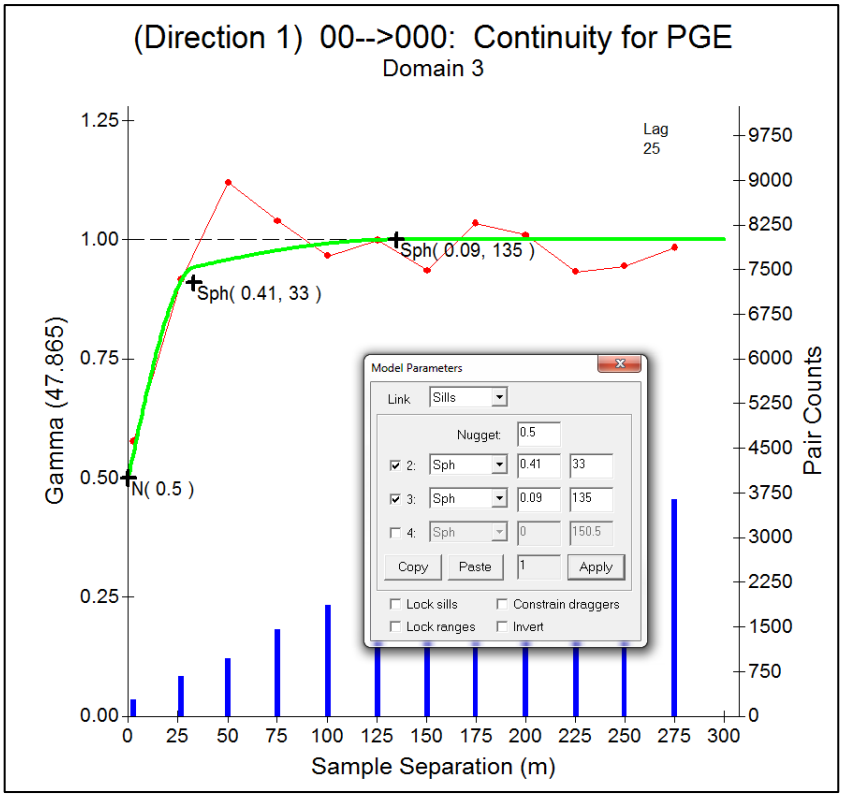

Figure 51. Semi-Variogram for 4E PGE grade of the Central Reef Facies

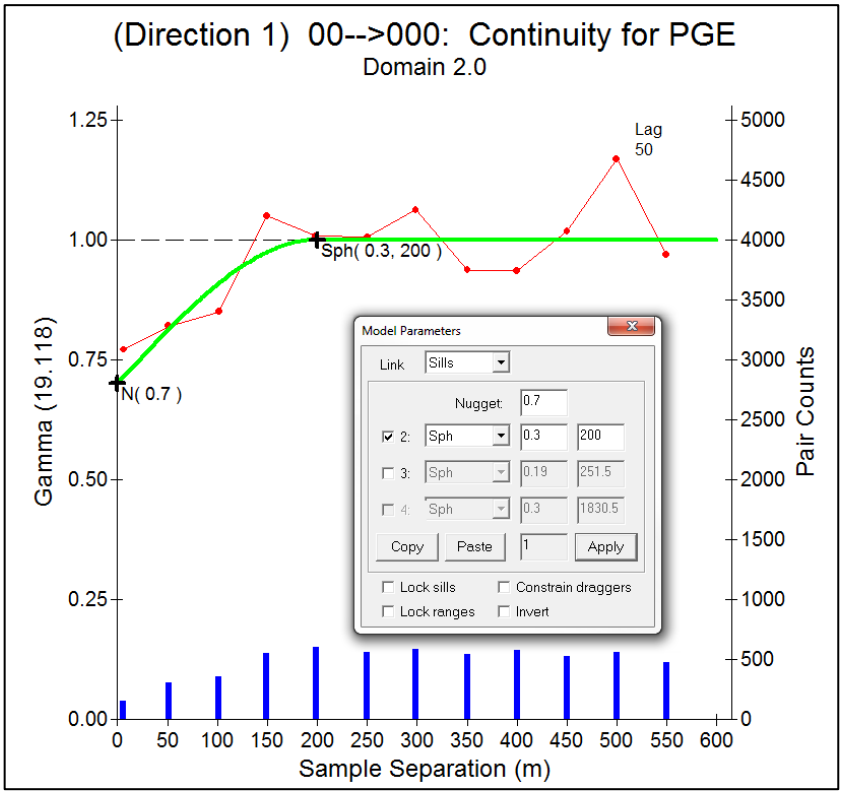

Figure 52. Semi-Variogram for 4E PGE grade of the Normal Reef Facies

4.6. Sampling grid, versus estimation block positions

The current mine design for Styldrift I is a bord and pillar mining method by means of trackless mechanised equipment. This mining method was designed to optimally extract the reef which is flat dipping and stable with wide mineralised areas. The layout consists of 8 m x 8 m bords, splits and pillars. The mining method governed the sample spacing options for testing. Possible configurations of the mine design were used to ascertain possible multiple spacing options along the direction of mining and to accommodate sampling on the face and pillars, as demonstrated illustrated in Figure 53 to Figure 58.

The following sampling grids were configured (m along strike x m across strike) :

- 8 m x 8 m (Figure 53)
- 16 m x 16 m (Figure 54)
- 32 m x 8 m (Figure 55)
- 32 m x 16 m (Figure 56)
- 32 m x 32 m (Figure 57)
- 40 m x 40 m (Figure 58)

Nine blocks with a 50 m x 50 m dimension (A, B, C, D, E, F, G, H and I) in the unmined face ahead were considered in the evaluation. In Figure 53 to Figure 58 the indicated direction of stope mining is from left to right on the right and the system of advance is upwards away from the mined out area that is sampled. Blocks A, B and C are located just in front of the sampled face 25 m ahead. Blocks D, E and F are positioned 75 m ahead of the mining face and blocks G, H and I is positioned 125 m from the sampled face. Blocks B,E and H are centrally located ahead of the mining face position. Blocks A, D and G positioned on the left hand side of the mining face and C, F and I is on the right hand side.

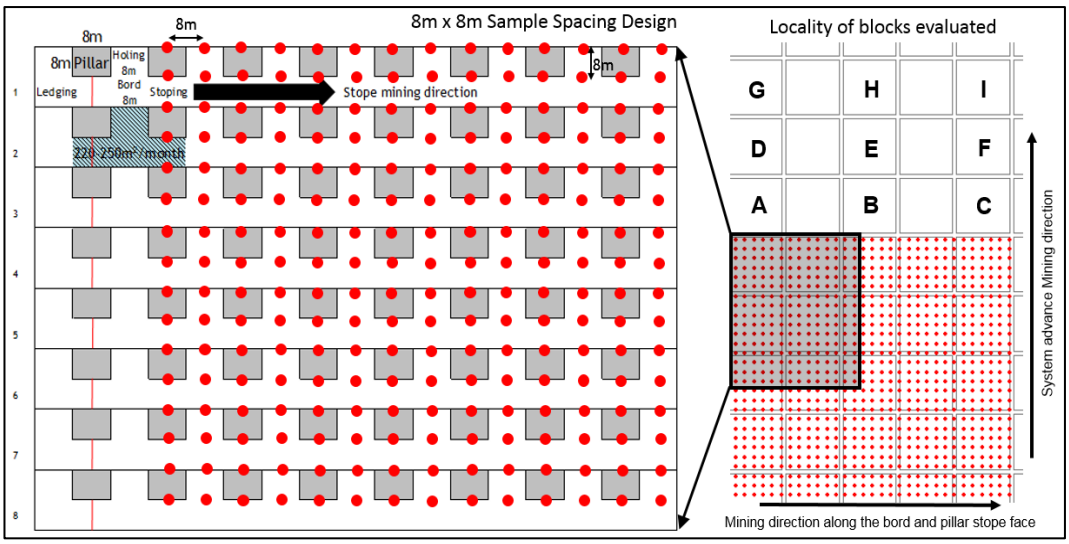

Figure 53. 8 m x 8 m sampling (left) and locality of blocks to be estimated (right).

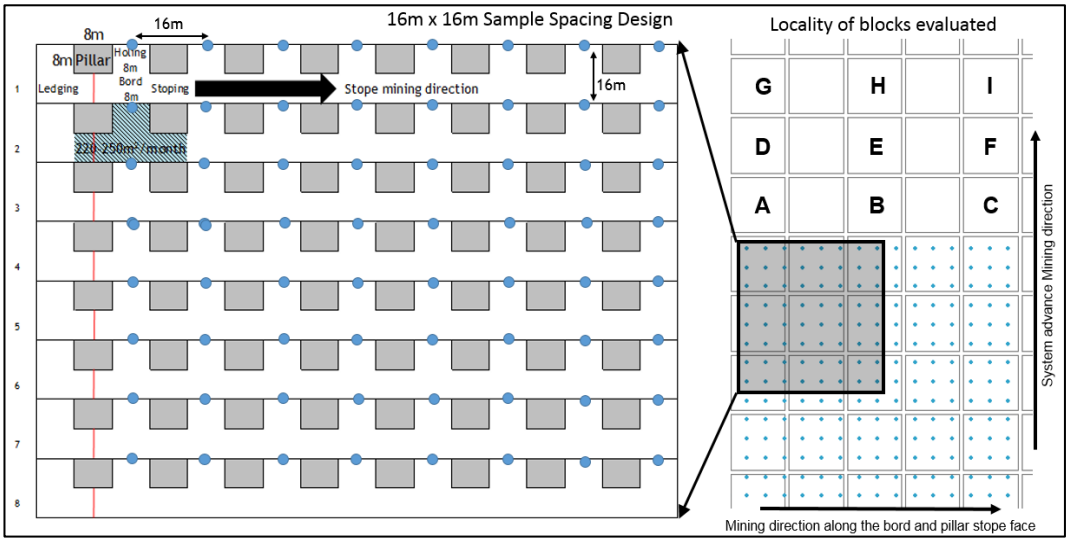

Figure 54. 16 m x 16 m sampling (left) and locality of blocks to be estimated (right).

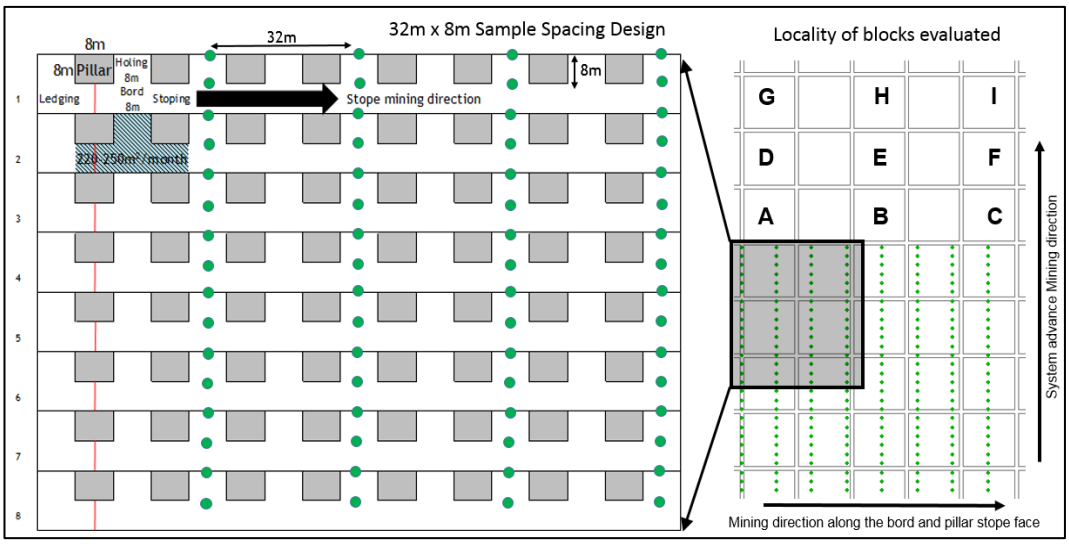

Figure 55. 32 m x 8 m sampling (left) and locality of blocks to be estimated (right).

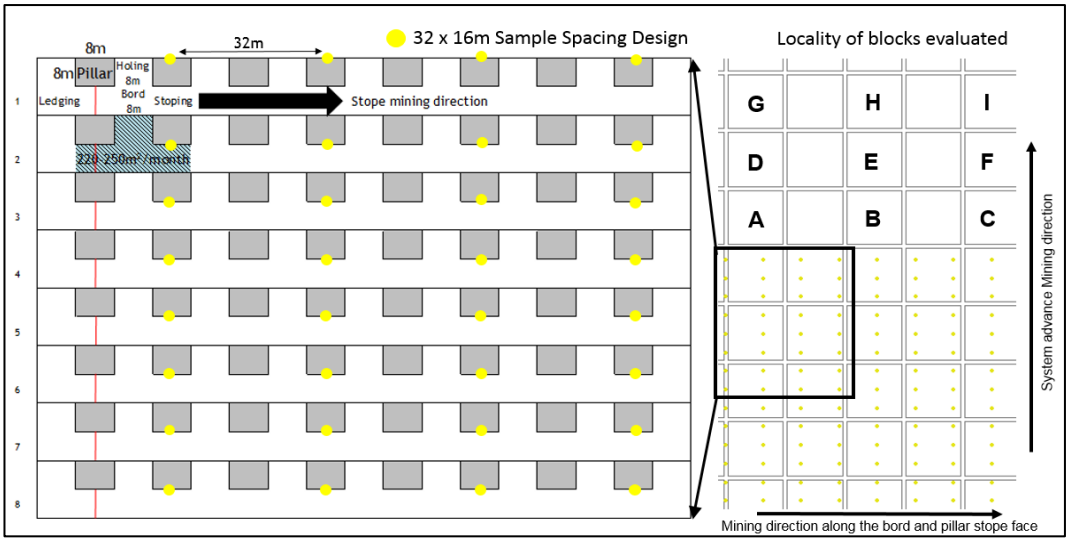

Figure 56. 32 m x 16 m sampling (left) and locality of blocks to be estimated (right).

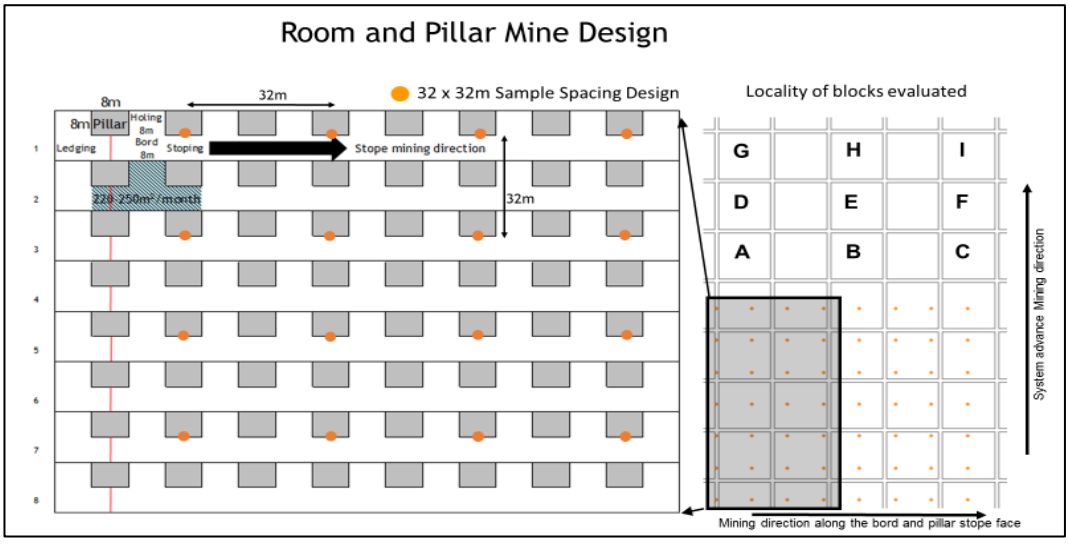

Figure 57. 32 m x 32 m sampling (left) and locality of blocks to be estimated (right).

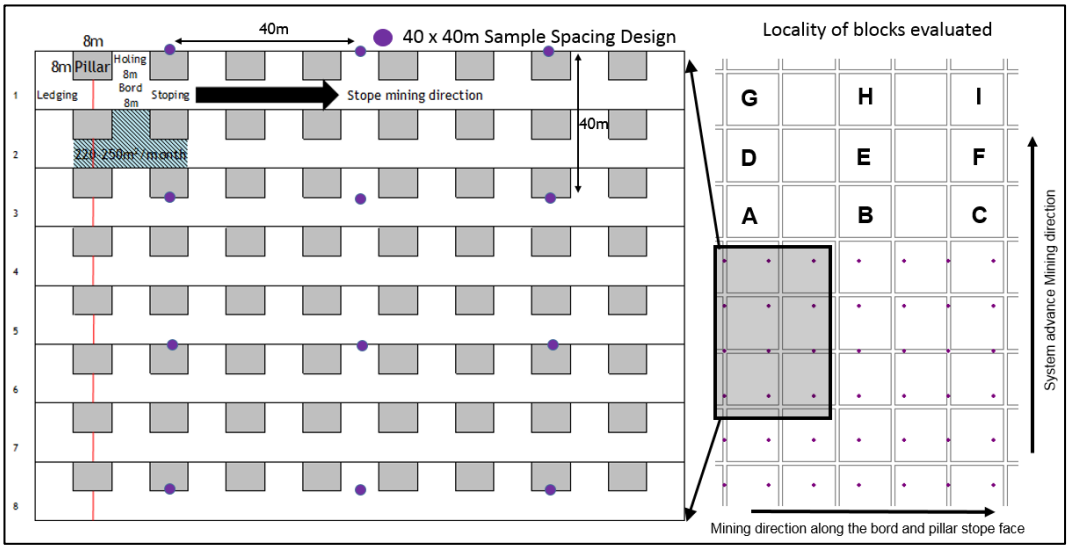

Figure 58. 40 m x 40 m sampling (left) and locality of blocks to be estimated (right).

4.6.1 Central Reef Facies Kriging Variance Analysis per grid

The kriging variance results of the evaluated test blocks (A, B, C, D, E, F, G, H and I) from applying the six sampling options within the Central Reef Facies are summarised and tabulated in Table 8. These estimate results from all the individual sampling option iterations are plotted in Figure 59.

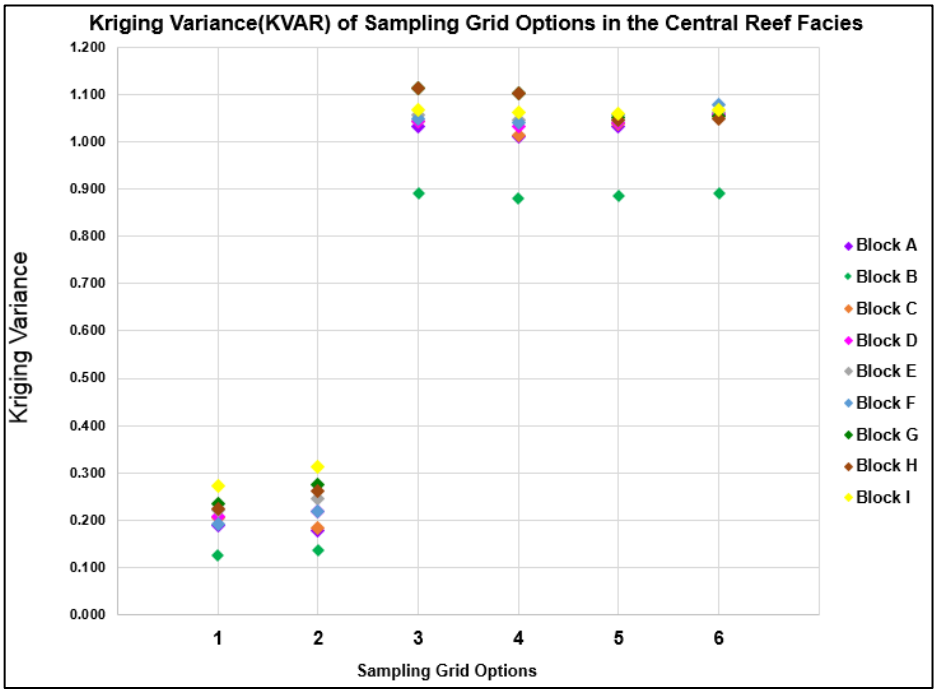

Figure 59. Plot of the kriging variances for the six grid sampling options for each of the blocks evaluated (A-I) in the Central Reef Facies.

The sampling options that produced the lowest minimum kriging variances for each tested block is highlighted in green in Table 8. The option that produced the second lowest value is highlighted in yellow.

Table 8. Kriging variance results for the six grid options from the evaluation of the individual nine blocks estimated (A-I) in the Central Reef Facies.

Central Facies Kriging Variance for PGE grade estimation per Block									
Grid Option #: Sampling Grid	**A**	**B**	**C**	**D**	**E**	**F**	**G**	**H**	**I**
1: 8 m x 8 m	0.188	0.126	0.205	0.207	0.222	0.193	0.235	0.224	0.272
2: 16 m x 16 m	0.178	0.136	0.183	0.219	0.244	0.220	0.274	0.262	0.313
3: 32 m x 8 m	1.032	0.890	1.042	1.043	1.055	1.049	1.113	1.112	1.068
4: 32 m x 16 m	1.011	0.880	1.014	1.031	1.046	1.040	1.101	1.102	1.062
5: 32 m x 32 m	1.032	0.885	1.037	1.041	1.047	1.056	1.050	1.046	1.059
6: 40 m x 40 m	1.055	0.889	1.053	1.060	1.061	1.077	1.053	1.048	1.066

Option 1 evaluation has the lowest kriging variances for majority of the blocks (A, C, D, E, F, G, H and I). The option 2 configuration of 16 m x 16 m highlighted in yellow is a close second for the lowest variance values to that of option 1. This can be observed easily from the graphical display of Figure 59. Grid option 1 (8 m x 8 m) and grid option 2 (16 m x 16 m) plots the lowest of less than 0.4 for all tested blocks of the various sample spacings. Options 3, 4, 5 and 6 produce high kriging variances for all blocks tested with values greater than 0.80.

4.6.2 Normal Reef Facies Kriging Variance Analysis per grid

The kriging variance results of the evaluated test blocks (A, B, C, D, E, F, G, H and I) from applying the six sampling options within the Normal Reef Facies are summarised and tabulated in Table 9. These estimate results from all the individual sampling option iterations are plotted in Figure 60.

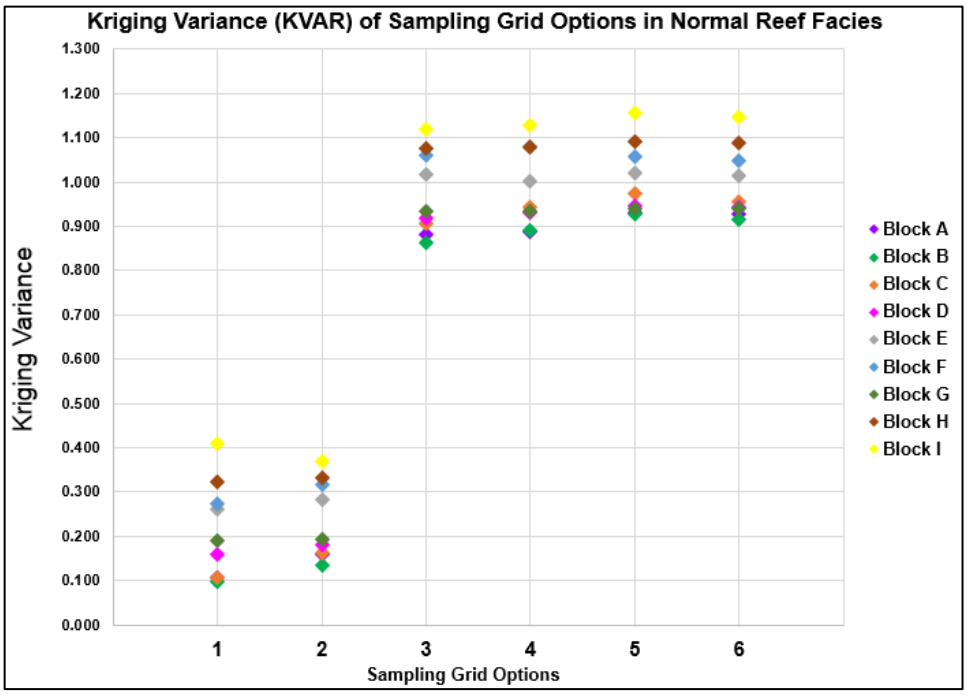

Figure 60. Plot of the kriging variances for the six grid sampling options for each of the blocks evaluated (A-I) in the Normal Reef Facies.

The sampling options that produced the lowest minimum kriging variances for each tested block is highlighted in green in Table 9. The option that produced the second lowest value is highlighted in yellow.

Table 9. Kriging variance results for the six grid options from the evaluation of the individual nine blocks estimated (A-I) in the Normal Reef Facies.

Normal Reef Facies Kriging Variance for PGE grade estimation per Block									
Grid Option : Sampling Grid	**A**	**B**	**C**	**D**	**E**	**F**	**G**	**H**	**I**
1: 8 m x 8 m	0.106	0.098	0.107	0.158	0.261	0.273	0.189	0.324	0.411
2: 16 m x 16 m	0.161	0.134	0.164	0.180	0.282	0.316	0.193	0.332	0.369
3: 32 m x 8 m	0.882	0.864	0.905	0.919	1.016	1.060	0.935	1.076	1.120
4: 32 m x 16 m	0.889	0.891	0.943	0.930	1.002	1.079	0.933	1.078	1.127
5: 32 m x 32 m	0.932	0.929	0.975	0.947	1.021	1.057	0.942	1.091	1.157
6: 40 m x 40 m	0.926	0.915	0.956	0.944	1.015	1.048	0.940	1.089	1.146

Option 1 evaluation has the lowest kriging variances for majority of the blocks (A, B C, D, E, F, G and H). The option 2 configuration of 16m x 16m highlighted in yellow is a close second for the lowest variance values to that of option 1. This can be observed easily from the graphical display of Figure 60. Grid option 1 (8 m x 8 m) and grid option 2 (16 m x 16 m) plots the lowest of less than 0.5 for all tested blocks of the various sample spacings. Options 3, 4, 5 and 6 produce high kriging variances for all blocks tested with values greater than 0.80.

4.7. Cost Comparison of the tested sampling strategies

The following analysis is a review of the sampling designs based on costing. The purpose of this additional consideration assists with the selection of a grid option that would be deemed optimal. The sampling process cannot be solely based on the outcomes of geostatistical work and independent of costing. An optimal strategy includes an understanding of the potential financial commitment required.

Calculated kriged variances from the evaluation were shown to be optimal for grid option 1 (8 m x 8 m) and grid option 2 (16 m x 16 m) configurations for both facies domains. When comparing the results from option 1 and option 2, the kriging variance values for majority of the tested blocks is slightly lower for the 8 m x 8 m sampling grid. The total number of samples from the various sampling densities required to test each sampling configuration over a fixed area of 62 500 m^2 is plotted against the calculated kriging variances in Figure 61 and Figure 62.

This defines the sampling scheme with the least effort or cost for a given precision which is deemed optimal from the geostatistical evaluation. The number of samples is used as a factor to evaluate sampling costs as the increasing number of samples is directly related to added expenses that will be incurred. The number of samples over the test area of 992 samples for option 1 is almost 4 times more than that of the 256 samples used for the 16 m x 16 m sampling configuration. Refer to Figure 61 and Figure 62 for comparison. The 8 m x 8 m sampling density is intensive and may prove to be logistically impractical and time consuming. The difference in the 16 m x 16 m calculated kriging variances is only slightly higher than the 8 m x 8 m. kriging variance values for grid option 2 occur within a small range of 0.136 and 0.313 for Central Reef Facies (Figure 61) and 0.134 and 0.369 for Normal Reef Facies (Figure 62). There is marginal precision gained with the 8 m x 8 m sampling configuration when comparing to a 16 m x 16 m sampling pattern is therefore not worth the inconvenience and additional sampling costs. The optimal value add would be gained from achieving a similar estimation precision for less sampling.

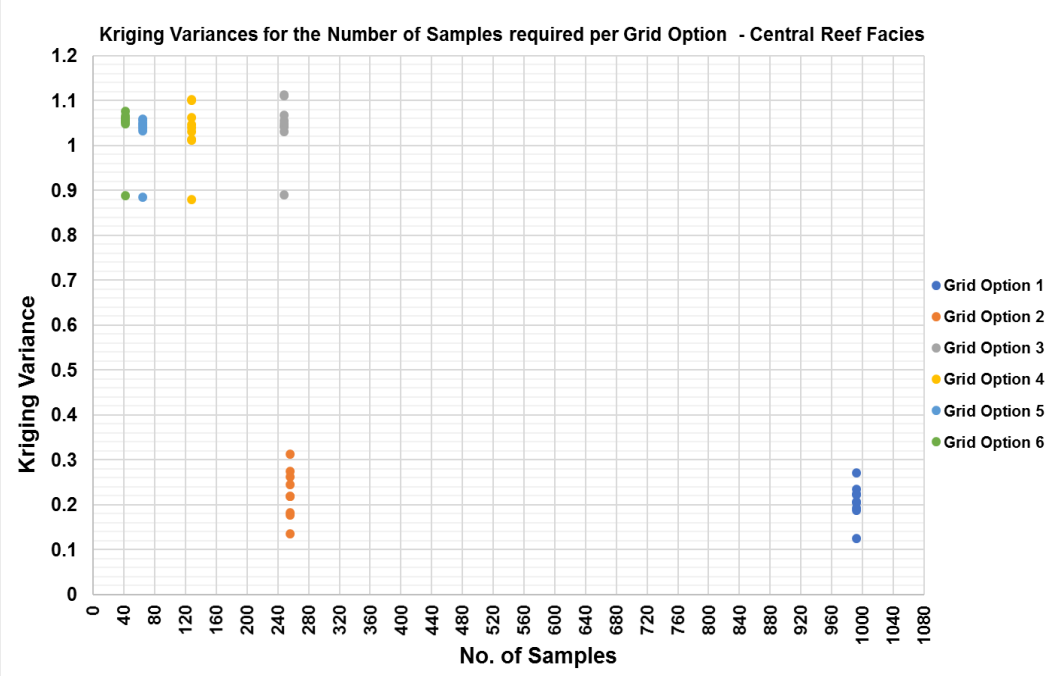

Figure 61. Kriging variance plot relative to the number of samples tested per sampling grid option for the Central Reef Facies

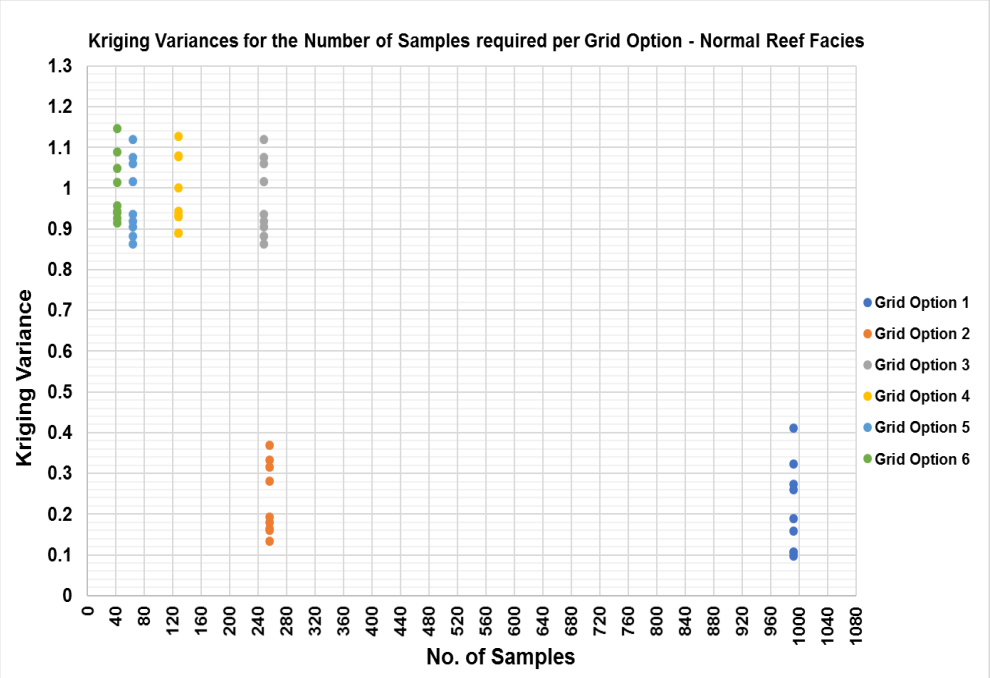

Figure 62. Kriging variance plot relative to the number of samples tested per sampling grid option for the Central Reef Facies

4.8. Summary of Results

QAQC analysis of the existing data used for the variography and facies domain analysis confirmed that the sample is representative by means of accuracy and precision testing. Regressions were conducted for missing rhodium prills based on the strong correlation to platinum. The exploratory data analysis of the grades and widths supported the differences observed between the two facies types and substantiated its separation as domains for the evaluation. Variogram contour maps for both facies domains indicated no evidence of anisotropy, therefore omni-directional variograms were modelled. Analysis of the kriging variance results in both facies domains for the nine tested blocks using six sample spacing options showed that option 1 (8 m x 8 m) and option 2 (16 m x 16 m) as the optimal options corresponding with the lowest estimation error values. A further assessment of the sample densities required as an indication of cost expressed the large number of samples required for option 1 which would be uneconomic and a similar precision can be gained from option 2 for a quarter of the cost. It can be concluded that the optimal sample spacing for both the Central Reef Facies and Normal Reef Facies is option 2 with a 16 m x 16 m grid.

5. CONCLUSIONS AND RECOMMENDATIONS

The objective of the study was to establish an optimal sample spacing for Styldrift I by applying geostatistical techniques with the consideration of what is practical and economic. Mining operations are continually plagued by the misclassification of ore reserves and poor reconciliation which in turn has major economic impacts due to decisions based on incorrect or poor information. As a result, the use of geostatistical techniques to evaluate the quality of various sampling options by calculating the kriging variance that measures the precision was successful in determining the optimal option.

Drill hole sample analysis was an easy, objective and practical method to use for investigating the optimal sample spacing. As with any geostatistical tool it is dependent on the stationarity of your data and therefore areas with different characteristics will employ different variography. If the continuity is different for local facies types, one does not necessarily apply the same sample spacing. The research study highlighted that the implicit cost consideration component is a key determining factor as the marginal precision gained for some sample strategies is not worth the additional cost, inconveniences in terms of time or effort and workers safety due to extended exposure at the underground mining.

Drill hole sampling analysis can be limited in its flexibility, it is therefore recommended that further work can be implemented based on applying a conditional simulation approach for the sample spacing analysis. Conditional simulation can be applied to assess numerous sampling options over the total area in a relatively short time. Englund and Heravi (1993) demonstrated the advantages of conditional simulation that can apply several parameters simultaneously. The conditional simulation possibility did not form part of this research study. Nonetheless this technique could be explored in future sampling design investigations.

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APPENDIX

QAQC Control Charts

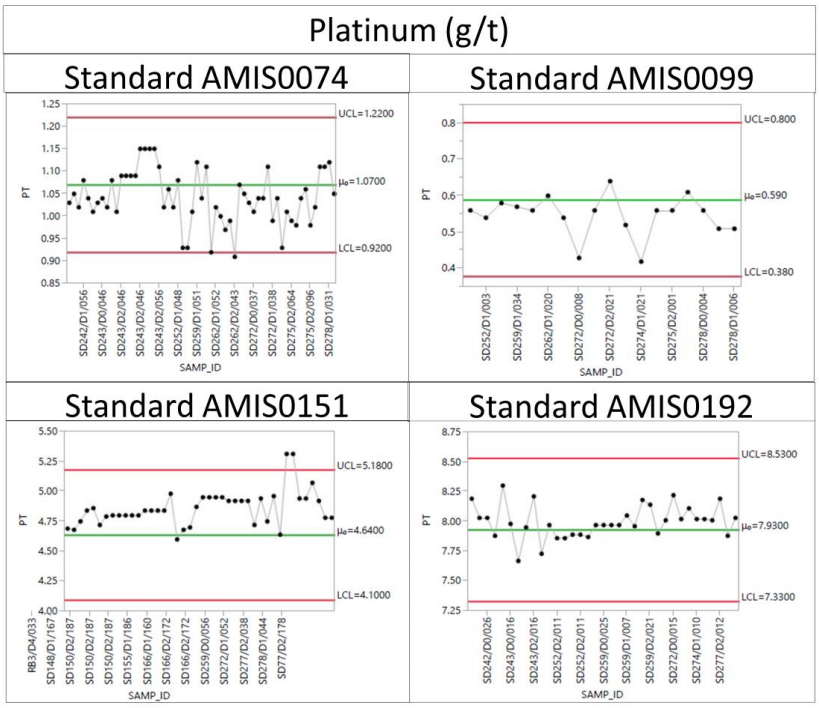

Figure 63. Control chart results for Platinum certified reference material standards

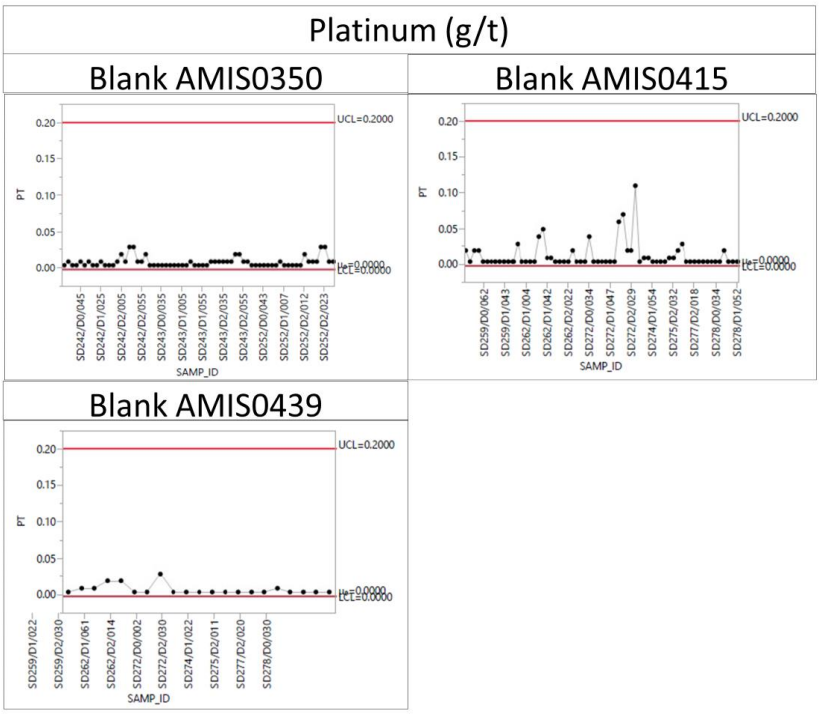

Figure 64. Control chart results for Platinum certified reference material blanks

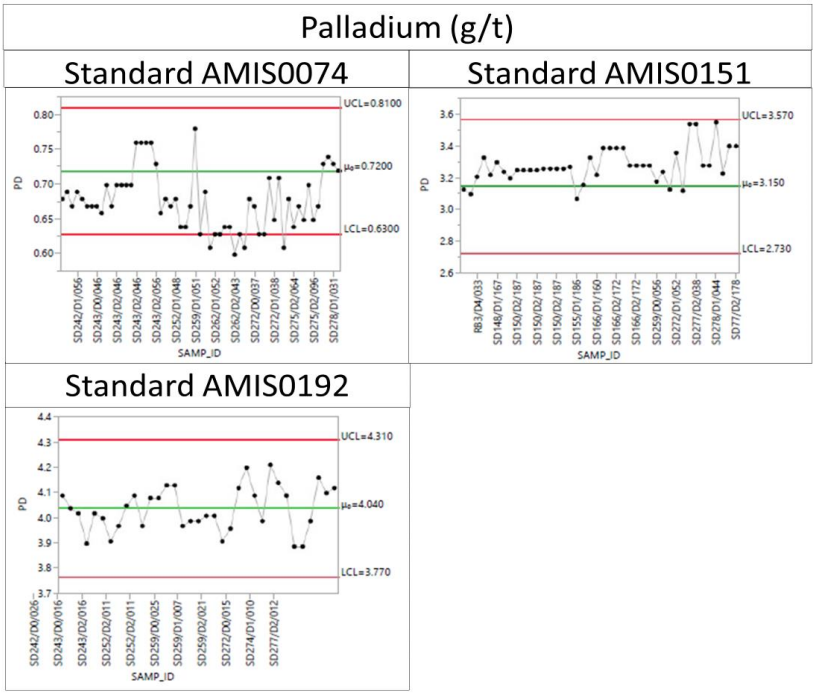

Figure 65. Control chart results for Palladium certified reference material standards

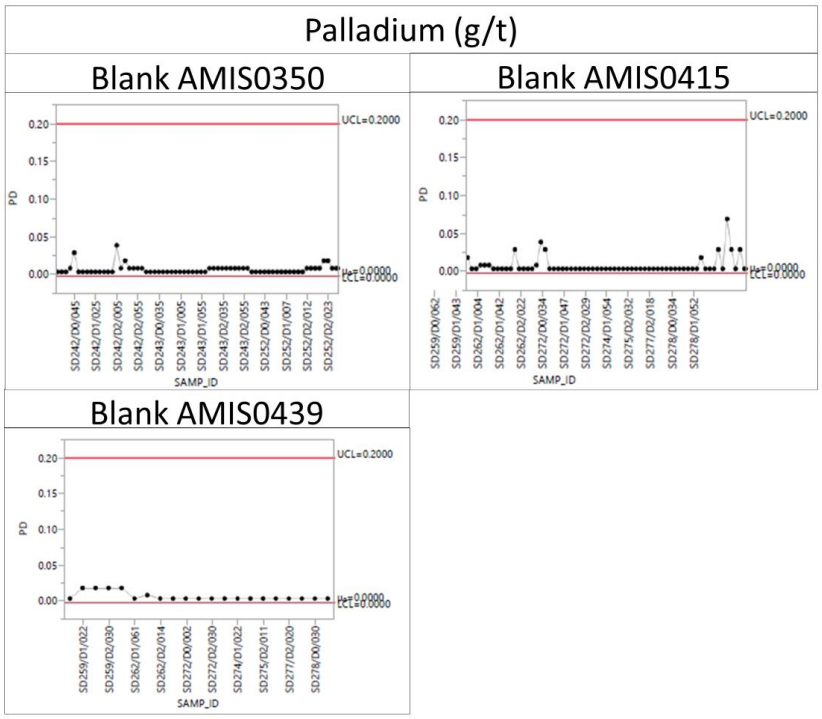

Figure 66. Control chart results for Palladium certified reference material blanks

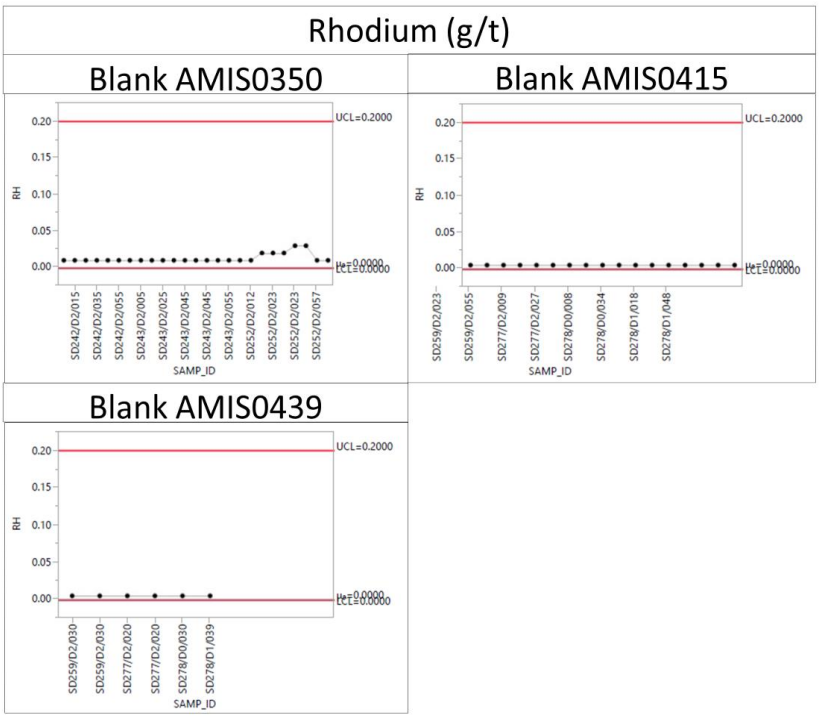

Figure 67. Control chart results for Rhodium certified reference material blanks

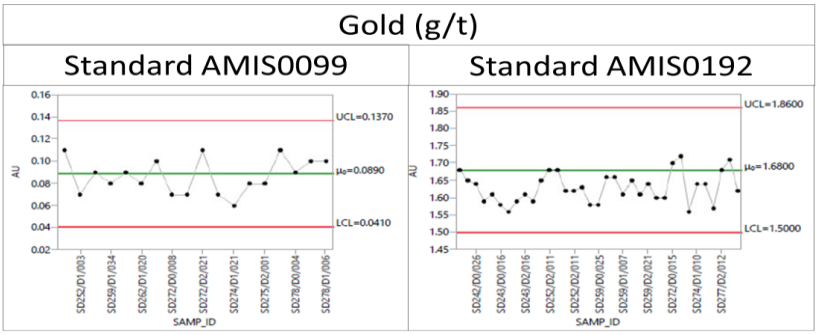

Figure 68. Control chart results for Gold certified reference material standards

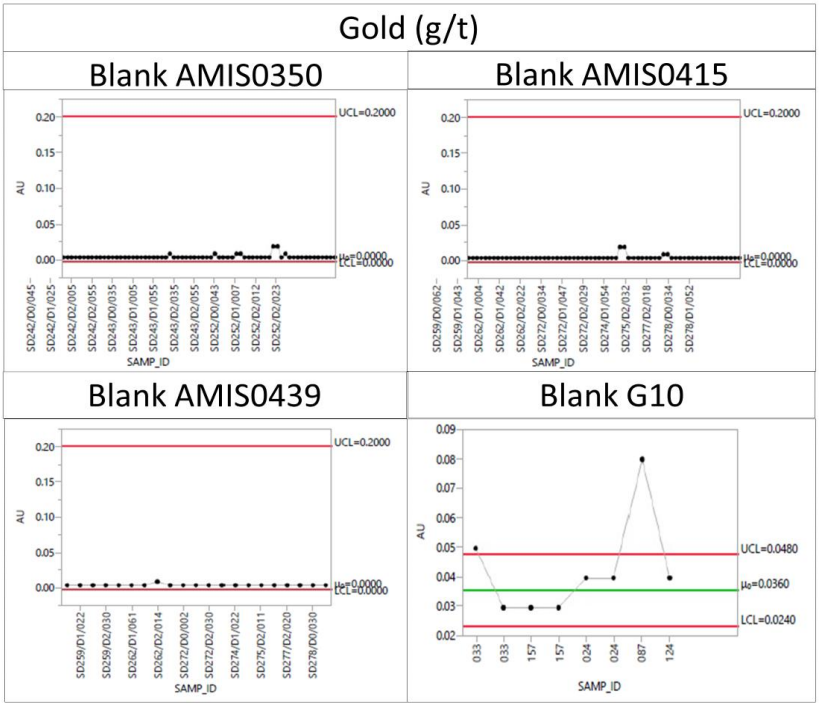

Figure 69. Control chart results for Gold certified reference material blanks