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Ref: 52045

12 January 2007

Company Announcements Office
Australian Stock Exchange Limited
20 Bridge Street
SYDNEY NSW 2000

By Electronic Lodgement

Dear Sir/Madam

NI 43-101 Report – Langer Heinrich Uranium Project

We wish to advise that Paladin Resources Ltd has lodged a technical report, pursuant to the NI 43-101 reporting obligations in Canada. This report details the resource update recently estimated by mineral resource specialists, Hellman and Schofield Pty Ltd. This report has been lodged with SEDAR* and can either be accessed from the following link www.sedar.com under the Company's profile or on the Paladin website.

Yours faithfully
Paladin Resources Ltd

JOHN BORSHOFF
Managing Director

**The System for Electronic Document Analysis and Retrieval (SEDAR), relating to the electronic filing of securities information as required by the securities regulatory agencies in Canada.*

Langer Heinrich, Namibia Resource Estimation

Technical Report
(Effective Date: 8th January 2007)

Langer Heinrich Uranium Pty Ltd

Langer Heinrich, Namibia Resource Estimation

Technical Report

(Effective date: 8th January 2007)

Langer Heinrich Uranium Pty Limited

Authors:	David Princep	Principal Geologist	(MAusIMM, B.Sc.)
Date:	8th January 2007		
Copies	Langer Heinrich Uranium Pty Limited	(2)	
	Paladin Resources - Perth	(1)	

Primary Author
David Princep

The reader is advised to read the Disclaimer (Section 3 of this Document)

LANGER HEINRICH

LOCATION

LATITUDE 22°47´ TO 22°49´ S
LONGITUDE 015°16´ TO 015°25´ E

Prepared for
Langer Heinrich Uranium Pty Ltd

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1 Summary

Hellman & Schofield Pty Ltd (H&S) was retained by Langer Heinrich Uranium (Pty) Ltd (LHU) to undertake re-estimation of U_3O_8 resources at the Langer Heinrich prospect, Namibia. Those resources are to enable updated life of mine planning studies to be undertaken. H&S has previously estimated resources for the Langer Heinrich deposit.

Langer Heinrich is a calcrete-hosted secondary uranium deposit associated with valley-fill sediments in an extensive Tertiary palaeodrainage system. It is located about 90 kilometres due east of Swakopmund, in the Namib Desert of central Namibia. The large-scale, hard-rock Rössing uranium mine is located about 40 kilometres north-west of Langer Heinrich. Uranium occurs as carnotite, an oxide mineral containing both uranium and vanadium, deposited as thin films lining cavities and fracture planes and as grain coatings and disseminations. The deposit extends over a 15km length. Mineralisation is near-surface, between one and thirty metres thick, and between 50 and 1,100 metres wide depending on the width of the palaeovalley.

The deposit was discovered in 1973 after a government-sponsored airborne radiometric survey of the area. Between 1974 and 1980 General Mining Union Corporation Limited (Gencor) undertook extensive percussion and diamond drilling, excavated a series of bulk sample test shafts, mined a large-scale costean and trial open pit, operated a trial dry screening plant and undertook detailed metallurgical, engineering and hydrological studies. The project was mothballed in the mid-1980's after a fall in the uranium price. It was acquired by Acclaim Exploration NL in 1998. That company completing infill RC drilling over a portion of the deposit and a pre-feasibility study in 1999-2000. Again the project was put on hold due to prevailing uranium prices.

Paladin Resources Ltd acquired the operating company, Langer Heinrich Uranium (Pty) Ltd and its assets in August 2002. Paladin has reconstructed all available drill hole data into a digital database that has been extensively checked and validated. The majority of sample data available to inform resource estimates derive from work undertaken by Gencor. That work appears to have been undertaken to a high standard and comparisons of U_3O_8 grades in Gencor's drilling to grades in test shafts and XRF assays and radiometric logging of Acclaim and Paladin drill holes largely support the reliability of the historic data. There were, however, relatively few data available to assess the reliability of U_3O_8 grades from Gencor percussion drill samples below the water table.

Although water flow rates are reportedly low the quality of those samples were regarded as questionable, as a consequence Paladin undertook an extensive drilling program in the area of Detail 1 during the second half of 2004 to prove up the resources below the water table. Paladin subsequently undertook drilling campaigns in 2005 to upgrade and extend Details 1, 2 and 7 and in 2006 to infill and extend Details 3, 4, 5 and 6. Resources defined by drilling on a regular 50m x 50m spacing or closer have been consigned to Measured category. Areas in which 50m drill coverage is incomplete have been consigned to Indicated category and mineralisation in areas drilled at 100m x 100m spacing have been allocated to Inferred category.

Variograms of U_3O_8 grades indicate that the continuity of grades is relatively poor over even quite short distances, not unlike that observed in some gold deposits. This is backed up by comparisons of nearest neighbour samples in drill holes and test shafts. However the overall continuity of mineralisation, the geological continuity, is quite strong in plan-view. Variograms based on areas of close-spaced sampling in Detail 1 have been used to guide modelling of the short-scale continuity of U_3O_8 grades in other areas.

Resources have been estimated at a number of cut-off grades using Multiple Indicator Kriging with block support correction. Primary model panel dimensions are 50mE x 50mN x 4mRL. Estimates assume that grade control sampling at about 5mE x 5mN x 1mRL will be available prior to mining and a selective mining unit of approximately 5mE x 5mN x 2mRL. Estimates for the entire deposit are summarised in the table below.

Cut off	Measured			Indicated			Inferred		
kg/t	**tonnes**	**kg/t**	**t U_3O_8**	**tonnes**	**kg/t**	**t U_3O_8**	**tonnes**	**kg/t**	**t U_3O_8**
0.10	39,457,372	0.44	17,419	30,930,569	0.34	10,663	97,201,522	0.35	34,146
0.20	27,427,118	0.57	15,701	18,453,039	0.48	8,835	55,628,450	0.51	28,118
0.25	**22,718,853**	**0.64**	**14,634**	**14,456,305**	**0.55**	**7,936**	**43,397,492**	**0.58**	**25,360**
0.30	18,880,824	0.72	13,574	11,520,570	0.62	7,124	34,700,152	0.66	22,961
0.35	15,803,502	0.80	12,568	9,233,807	0.69	6,377	27,835,964	0.74	20,718
0.40	13,338,122	0.87	11,640	7,504,693	0.76	5,725	22,718,844	0.83	18,791
0.45	11,389,882	0.95	10,804	6,181,120	0.83	5,160	18,779,095	0.91	17,105
0.50	9,810,200	1.02	10,049	5,143,487	0.91	4,662	15,765,281	0.99	15,660
0.55	8,497,910	1.10	9,352	4,337,562	0.98	4,234	13,460,104	1.07	14,438
0.60	7,378,487	1.18	8,700	3,688,109	1.05	3,858	11,643,691	1.15	13,382
0.65	6,444,162	1.26	8,110	3,151,823	1.12	3,517	10,117,076	1.23	12,411
0.70	5,704,680	1.33	7,607	2,745,240	1.18	3,241	8,833,413	1.31	11,542

Total Langer Heinrich estimated resources

In all tables where resource estimates are detailed, metal content in terms of t U_3O_8 are based on contained metal in the ground and take no account of mining or metallurgical recoveries, mining dilution or other economic parameters.

The assumed degree of selectivity that can be achieved during mining is regarded as somewhat conservative. Gencor's trial mining has demonstrated that highly selective mining can be achieved at relatively low production rates. Picking of upper and lower ore contacts will be important in mining and the use of technology such as laser or DGPS excavation control may considerably reduce mining dilution.

2 Introduction and Terms of Reference

2.1 Terms of Reference

This report is to comply with disclosure and reporting requirements set forth in the Toronto Stock Exchange (TSX) Company Manual, National Instrument 43-101, Companion Policy 43-101CP, and Form 43-101F1.

The report complies with Canadian National Instrument 43-101, for the 'Standards of Disclosure for Mineral Projects' of December 2005 (the Instrument) and the resource and reserve classifications adopted by CIM Council in August 2005. The report is also consistent with the 'Australasian Code for Reporting of Mineral Resources and Ore Reserves' of September 2004 (the Code) as prepared by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Mineral Council of Australia (JORC).

2.2 The Purpose of this Report

This report was prepared in support of updated resource estimations announced on the Toronto and Australian Stock Exchanges by Paladin Resources Limited, the parent company of Langer Heinrich Uranium (Pty) Limited on the 24th November 2006. The current report reflects material changes to previous mineral resources arising from the completion of an infill drilling program in the area of Details 1, 2 and 7 in 2005 and Details 3, 4, 5 and 6 in 2006 resulting in a new mineral resource estimate.

2.3 Principal Sources of Information

In addition to a number of site visits undertaken by the author to the Langer Heinrich Uranium Project, the most recent being in December 2006, the author has relied extensively on information compiled for the bankable feasibility study (BFS) on the Langer Heinrich Uranium project and the previous Independent Technical Report on Langer Heinrich. The BFS was compiled by Minproc Pty Ltd (Minproc) with contributing sections from H&S, Minproc, Mining Solutions Consultancy and SoftChem.

The author has made all reasonable enquiries to establish the completeness and authenticity of the information provided and identified, and a final draft of this report was provided to LHU, along with a written request to identify any material errors or omissions, prior to lodgement.

2.4 Qualifications and Experience

The primary author of this report is Mr David Princep, who is a professional geologist with over 16 years experience in the mining and evaluation of mineral properties within Australia and overseas. Mr Princep is currently employed as Principal Geologist with Paladin and is a Member of the Australasian Institute of Mining and Metallurgy (AusIMM), and has the appropriate relevant qualifications and experience to be considered a Competent Person as defined in the JORC Code and a Qualified Person as defined in Canadian National Instrument 43-101. Mr Princep has visited the Langer Heinrich Uranium Project on a number of occasions, the last being in December 2006.

2.5 Independence

The material changes being reported in this document do not require demonstration of independence. As a consequence, this report is being authored and published by Paladin Resources Ltd.

2.6 Abbreviations

A full listing of abbreviations used in this report is provided in *Table 1* below.

List of Abbreviations

	Description
$	Australian dollars
"	inches
µ	microns
3D	three dimensional
AAS	atomic absorption spectrometry
Au	gold
bcm	bank cubic metres
BFS	Bankable feasibility study
CC	correlation coefficient
cfm	cubic feet per minute
CIC	carbon in column
CIL	carbon-in-leach
cm	centimetre
cusum	cumulative sum of the deviations
CV	coefficient of variation
DTM	digital terrain model
E (X)	easting
EDM	electronic distance measuring
EV	expected value
g	gram
g/m^3	grams per cubic metre
g/t	grams per tonne
HARD	half the absolute relative difference
HDPE	high density poly ethylene
H&S	Hellman & Schofield
HQ_2	size of diamond drill rod/bit/core
hr	hours
HRD	half relative difference
ICP-MS	inductivity coupled plasma mass spectroscopy
ID	Inverse Distance weighting
ID^2	Inverse Distance Squared
IPS	integrated pressure stripping
IRR	internal rate of return
ISO	International Standards Organisation
ITS	Inchcape Testing Services
kg	kilogram
kg/t	kilogram per tonne
km	kilometres
km^2	square kilometres
kW	kilowatts

	Description
kWhr/t	kilowatt hours per tonne
$l/hr/m^2$	litres per hour per square metre
LHU	Langer Heinrich Uranium
LM2	Labtechnics 2kg (nominal) pulverising mill
M	million
m	metres
Ma	thousand years
MIK	Multiple Indicator Kriging
ml	millilitre
mm	millimetres
MMI	mobile metal ion
Moz	million ounces
Mtpa	million tonnes per annum
N (Y)	northing
NaCN	sodium cyanide
NPV	net present value
NQ_2	size of diamond drill rod/bit/core
°C	degrees centigrade
OK	Ordinary Kriging
oz	troy ounce
P80 -75µ	80% passing 75 microns
PAL	pulverise and leach
ppb	parts per billion
ppm	parts per million
psi	pounds per square inch
PVC	poly vinyl chloride
QC	quality control
Q-Q	quantile-quantile
RAB	rotary air blast
RC	reverse circulation
RL (Z)	reduced level
ROM	run of mine
RQD	rock quality designation
SD	standard deviation
SG	Specific gravity
SMU	simulated mining unit
t	tonnes
t/m^3	tonnes per cubic metre
tpa	tonnes per annum
w:o	waste to ore ratio

Table 1: Abbreviations used in this document

3 Reliance on Other Experts

The author of this report is not qualified to provide extensive comment on legal facets associated with ownership and other right pertaining to Langer Heinrich Uranium Pty Ltd mineral properties, which are included in Sections 1 and 4. The author has relied heavily on review of copies of the various title deeds, tenement and company searches made available by Langer Heinrich Uranium Pty Ltd, encapsulating the rights afforded to Langer Heinrich Uranium Pty Ltd. The author did not see or carry out any legal due diligence confirming the legal title of Langer Heinrich Uranium Pty Ltd to the properties.

The resource estimates included in this report were prepared by independent consulting firm; Hellman & Schofield Pty Ltd. Hellman & Schofield has provided separate certificates of responsibility in relation to Section 17. Those resources relating to Details 1, 2 and 7 were estimated in 2005 by Mr. David Princep, who was at the time a full time employee of H&S. The resources for Details 3, 4, 5 and 6 were estimated in 2006 by Mr. Neil Schofield who is a full time employee of H&S.

The author of this report is not qualified to provide extensive comment on environmental issues associated with the Langer Heinrich Uranium Project, included in Sections 5 and 17. The assessment of data pertaining to environmental issues relies heavily on information provided by Langer Heinrich Uranium Pty Ltd and SoftChem (Environmental Consultants), which has not been independently verified by the author. Comments made by the author rely on the environmental report for the Langer Heinrich Uranium Mine.

4 Property Description and Location

4.1 Location

The Langer Heinrich Uranium Project is located in the Republic of Namibia ("Namibia") in southern Africa. The project site is within the Erongo Region, 180km west of the national capital, Windhoek and 80 km east of the major deepwater seaport at Walvis Bay, a well established city and the main sea port of Namibia, and the coastal town of Swakopmund .

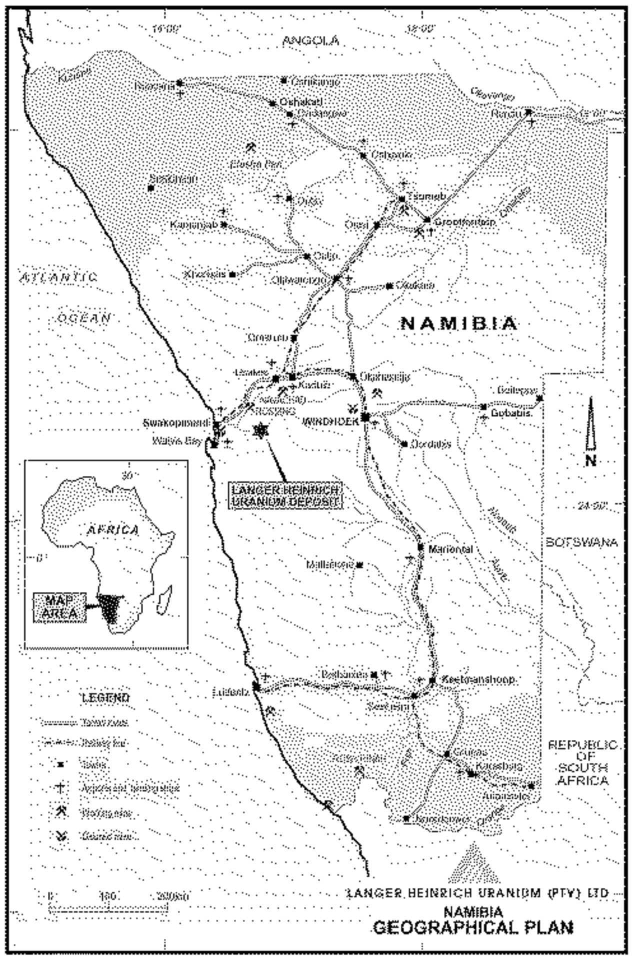


Figure 1: Location Map

Namibia, formerly a protectorate of the Republic of South Africa, became independent on the 21st March 1990. The political system is a stable, multi-party parliamentary democracy. English is the official language and the legislative and fiscal regimes are very similar to those of South Africa. The Namibian dollar is linked at parity with the South African Rand ("SAR").

Namibia is part of the Southern Africa Development Corridor ("SADC") zone and has a population of 1.8M people. The Namibian economy relies heavily on the country's mining industry with mining being its principal export industry and the Namibian Government offers considerable incentives to companies setting up new mining operations. Specifically, title to exploration and mining tenements is clear-cut and certain, and accelerated depreciation and reduced corporate taxation rates are available.

4.2 Description of Licence

The Langer Heinrich Uranium Project is covered by a single Mining Licence, ML140.

No	Longitude			Latitude			Namibian (Gauss Conform) (Central meridian: 15°)	
	Degree	Minute	Second	Degree	Minute	Second	Y coordinate	X coordinate
1	E15	16	21.828	S22	47	40.272	-27999.61748	87998.80466
2	E15	17	49.488	S22	47	40.128	-30499.58848	87998.80466
3	E15	17	49.560	S22	48	9.360	-30499.58848	88898.79167
4	E15	25	7.860	S22	48	8.388	-42999.41874	88898.79167
5	E15	25	8.112	S22	49	29.676	-42999.41874	91398.75559
6	E15	16	22.044	S22	49	30.828	-27999.61748	91398.75559

Table 2: Coordinates of ML 140

The Licence has been surveyed.

In 1992 the new Minerals (Prospecting and Mining) Act (No 33 of 1992) of Namibia (see section 4.2.1) was promulgated and, in accordance with the new act, Gencor applied for a mineral deposit retention licence (MDRL). MDRL 2236 was granted on the 16 August 1994 for 5 years and has been renewed for further 3 periods (of two years). MDRL 2236 was converted to a full mining license ML 140 in July 2005 prior to the commencement of mining in September 2006

The current ML allows for the mining of the deposit to be carried out. To maintain the licence in good standing, the company is required to:

- Maintain the project database,
- Submit annual financial statements to the Government of Namibia, and
- Commence mining operations within a set period of time following the grant of the License.

Title No.	Surface Area (hectares)	Date Acquired	Expiry Date	Commitments
ML140	4,375	26 July 2005	25 July 2030	N$5,000

Table 3: Licence Details

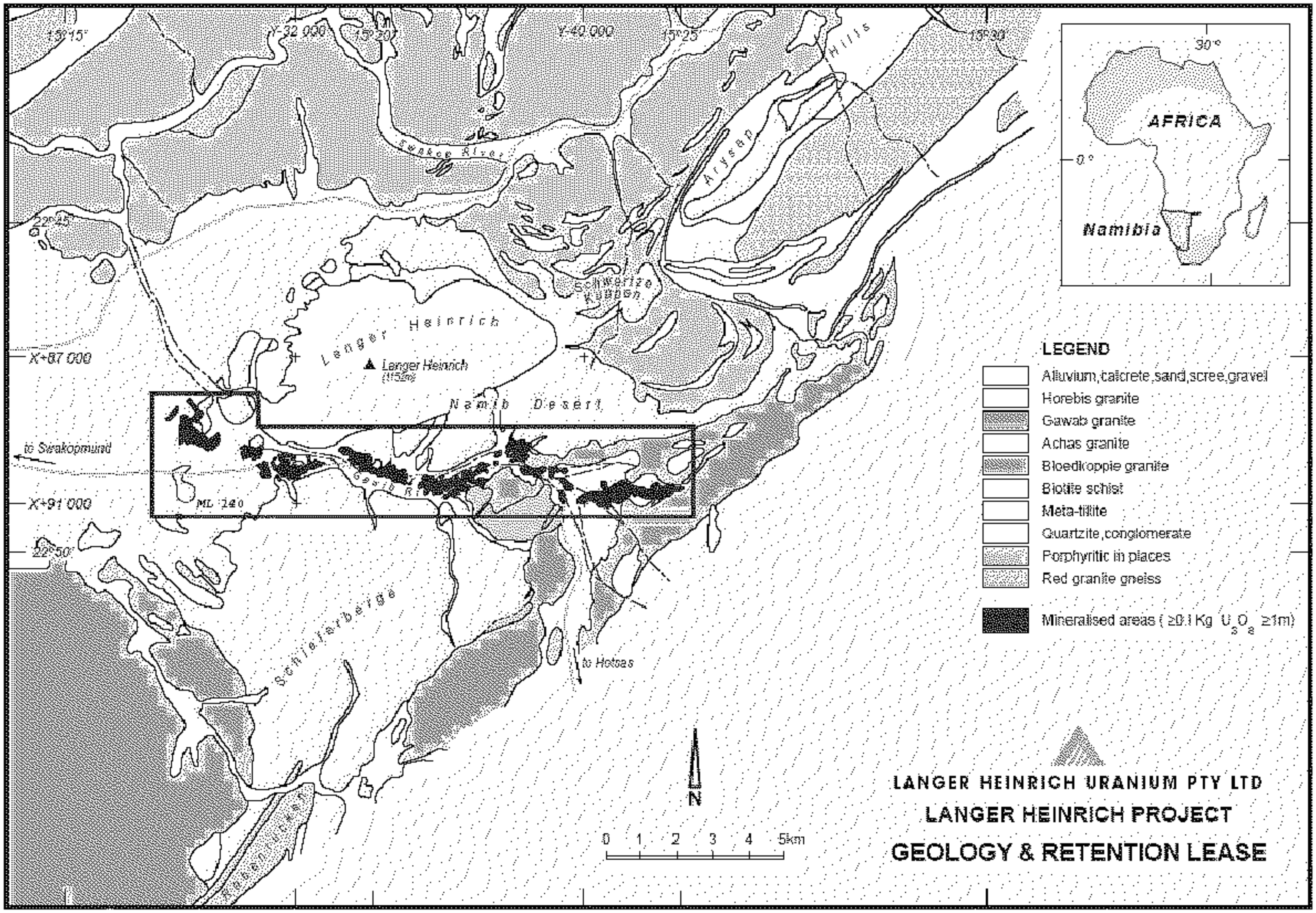


Figure 2: Property map

The licence covers 3,975 ha of State land in the Namib Naukluft Park. Since establishment of the park, numerous prospecting and mining activities have been conducted within it. Environmentally irresponsible behaviour by some operating companies, resulting in long-lasting damage, has led to the establishment of the Policy for prospecting and mining in protected areas and national monuments in 1999 (the term 'protected areas' includes national parks and game reserves). This policy document outlines the procedures to be followed before government takes a decision if a prospecting or mining activity may commence. In addition, any proposed mining project shall also have to adhere to the following 13 principles of environmental management (SAIEA, 2003):

- Renewable resources shall be utilised on a sustainable basis for the benefit of current and future generations of Namibians,
- Community involvement in natural resource management and sharing in the benefits arising there from shall be promoted and facilitated,
- Public participation in decision making affecting the environment shall be promoted,
- Fair and equitable access to natural resources shall be promoted,
- Equitable access to sufficient water of acceptable quality and adequate sanitation shall be promoted and the water needs of ecological systems shall be fulfilled to ensure the sustainability of such systems,
- The precautionary principle and the principle of preventative action shall be applied,
- There shall be prior environmental assessment of projects and proposals which may significantly affect the environment or use of natural resources,
- Sustainable development shall be promoted in land use planning,
- Namibia's movable and immovable cultural and natural heritage including its biodiversity shall be protected and respected for the benefit of current and future generations,
- Generators of waste and polluting substances shall adopt the best practicable environmental option to reduce such generation at source,
- The polluter pays principle shall be applied,
- Reduction, re-use and recycling shall be promoted, and
- There shall be no importation of waste into Namibia.

In addition supplementary conditions, that include the implementation of an environmental rehabilitation programme to the satisfaction of the Directorate of Resource Management - Ministry of Environment and Tourism, have been imposed on MDRL2236.

A Mining Licence must be applied for and granted prior to project development.

4.2.1 Minerals (Prospecting and Mining) Act (No 33 of 1992)

Administrative body: Department of Mines, Ministry of Mines and Energy.

Main objectives: This act regulates reconnaissance, prospecting and mining of minerals. Various licence types, and their implications, are stipulated. The act details reporting requirements for monitoring of activities and compliance to environmental performance, such as disposal methods. The Mining Commissioner, appointed by the Minister, is responsible for implementing these regulations. A Mineral Board has also been established, the functions of which are to advise the Minister and cooperate with other ministries.

Several explicit references to the environment and its protection are contained in the act, which provides for environmental impact assessments, rehabilitation of prospecting and mining areas and minimising or preventing pollution.

Section 91(f) requires that an application for a mining licence contains particulars of:

- The condition of the existing environment;
- An estimate of the impacts and the proposed mitigation measures; and
- Details regarding pollution control, waste management, rehabilitation and minimisation of impacts on adjoining land.

4.3 Ownership

The Langer Heinrich Uranium Project is owned 100% by Langer Heinrich Uranium (Pty) Limited ("LHUPL"), a company registered in Namibia. Through subsidiary holding companies, LHUPL is now beneficially owned 100% by Paladin

LHUPL was formed by a subsidiary of the South African mining house, Gencor and others in 1973 to prospect for uranium in Namibia. The participants had been granted prospecting concessions covering 970km2 in the Namib Desert. The Langer Heinrich Deposit was discovered by Gencor within the concession area in late 1973. Over the following seven years Gencor undertook extensive exploration and test work to evaluate the potential of the Deposit

In 1998 ownership of LHUPL through its parent holding company, Lahndrik Holdings SA, a company registered in Luxembourg, was transferred from Gencor to Acclaim. In July 2002, Acclaim agreed to sell its 100% holding in Lahndrik Holdings SA to Paladin Energy Minerals NL, a wholly owned subsidiary of Paladin, and full ownership of LHUPL and the Langer Heinrich Uranium Project was thus acquired by Paladin

The corporate structure is summarised in the following diagram:

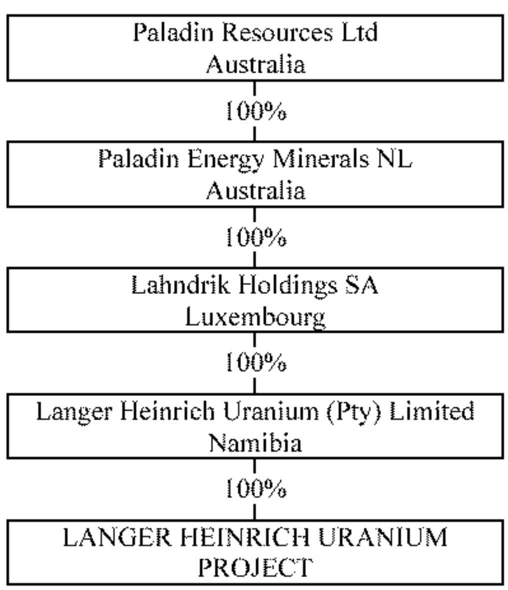


Figure 3: Corporate Ownership

4.4 Fees / Taxes and Assessment of Work Requirements and Liabilities

Work commitments relating to the granting of the Mining License, that mining activities should be commence within 1 month of grant and are substantially in alignment with the budget submitted in the ML application, have been carried out. Expenditure commitments attached to ML 140 are N$5,000 per annum, although these can be modified if the Government wishes to do so.

Ml 140 is subject to a royalty of A$0.12 per kilogram U3O8 sold payable to Redport Ltd., a wholly owned subsidiary of Mega Uranium Ltd as well as a royalty payable to the Namibian government of 2% of the market value of nuclear fuel minerals.

All permits required for the sustained operation of the mine have been granted. There are no outstanding environmental liabilities other than those normally associated with the de-commissioning of the mine site following the cessation of mining and processing activities. Clean up of the historical mining areas within the mining license will take place during the course of normal mining operations.

4.5 Background information on Namibia

See NI43-101, Langer Heinrich, Namibia, Independent Technical Report, Resource and Reserve Estimation, 7th June 2005.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Access

The only land access to the licence area is via district road C28 from Swakopmund (*Figure 4*), but following an existing turn-off approximately 20 km from the plant site. Repairs and grading have been carried out to up-grade the quality of this dirt road from the turn-off to the site.

There are two disused airstrips close to the project site that are capable of handling up to 8-seater light aircraft. One strip is in the middle of the valley near an old exploration camp and the other on the desert floor just west of the Gawib Valley. The former has been re-established as a usable strip (for small twin engine planes only) following the completion of some remedial work. This air strip is now available for use in emergencies or as other needs arise.

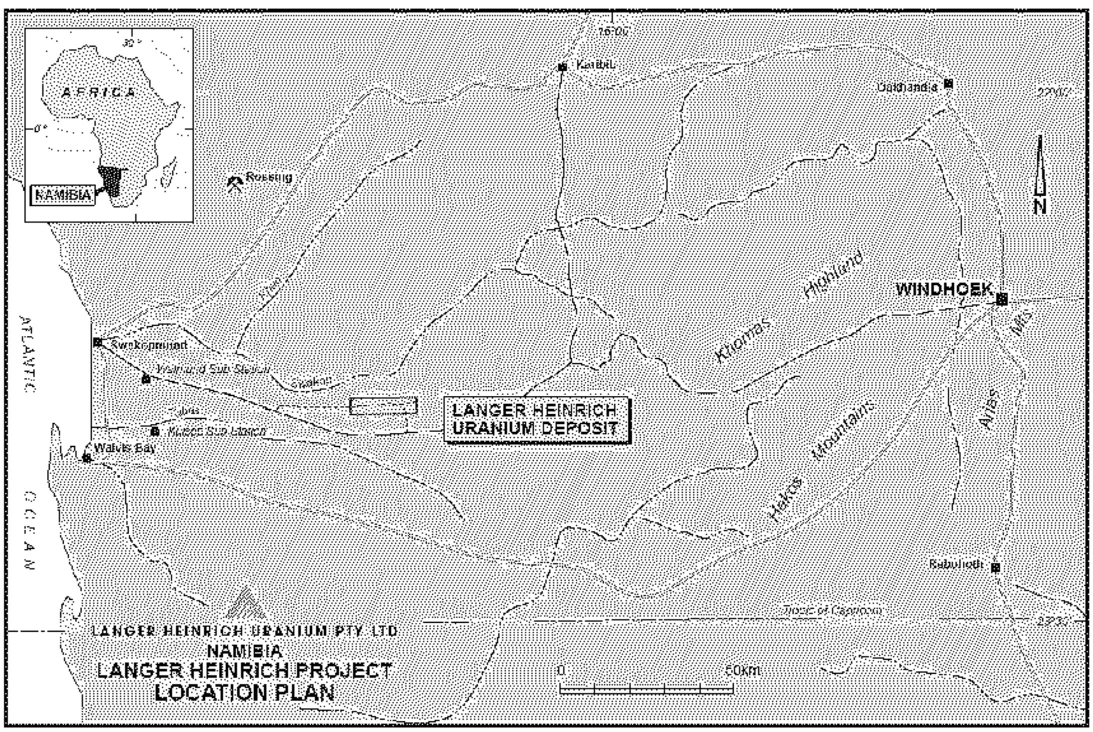


Figure 4: Site Location

5.2 Climate

The Langer Heinrich site is situated in the arid Namib Desert of Namibia, with the climate typical of a desert. In general, the climate is typified by hot and dry conditions with rather cool nights. The wind speed in winter is stronger than in summer, mostly due to the dominant high-pressure system of the inland regions that result in subsiding air drainage to the coastal regions. Climatic assessment has been made based on data obtained over two year period, March of 2000 to June 2002. This represents a short historical data set and therefore does not contain extremes such as wind gusts and maximum and minimum temperatures.

A typical desert rainfall pattern is evident in the data as only two significant rainfall events were recorded during the measurement period. The maximum rainfall recorded in one month was 76 mm (*Figure 5*) with 26 mm during a 24-hour period and 15.7 mm in one hour.

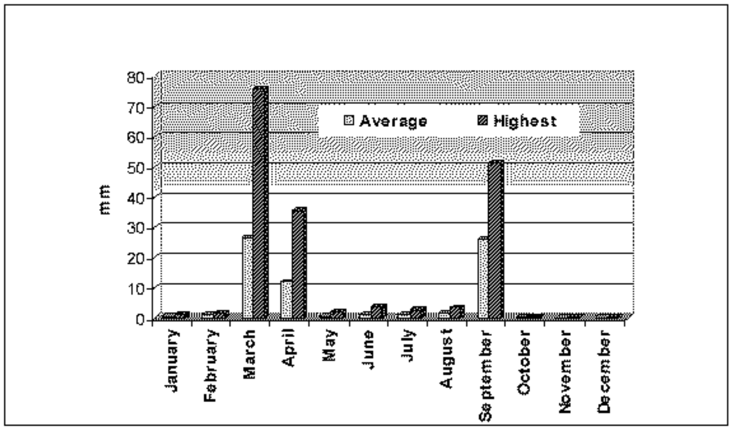


Figure 5: Rainfall data at Langer Heinrich

Due to the typical desert climatology, as measured at the site, the temperature variation can approach and even exceed 30°C on any given day. The lowest hourly temperature measured during the sampling period was 4.3°C and the maximum hourly temperature was 40.9°C. Mean daily temperatures exceed 20°C for most months, with July and September the exceptions (*Figure 6*). Due to the relative close proximity to the coast, frost will be a rare experience at this site.

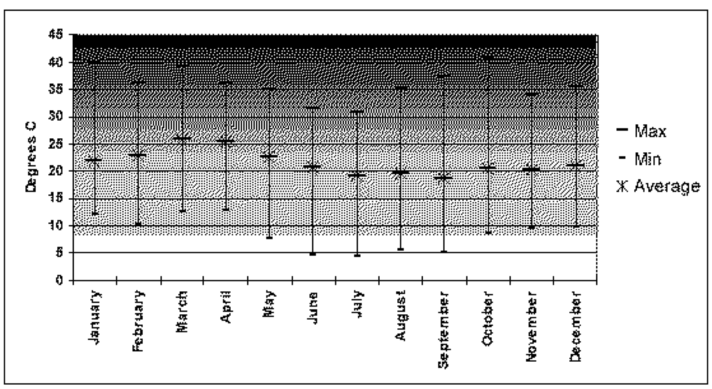


Figure 6: Monthly temperature range at Langer Heinrich

The directional wind pattern at Langer Heinrich shows distinct summer and winter patterns, however, the topographical influence of the nearby mountain ranges and even the greater topographical influence of the high inland Plateau with the site at the border of the coastal plains are evident. In summer (*Figure 7*), the constant southerly experienced on the coast swings to a westerly at the site due to the physical Langer Heinrich mountain in close proximity to the north of the site.

In winter the strong easterly winds is a function of the high-pressure system dominating the Southern African Highveld that causes air to subside and then to drain towards the coast (see *Figure 7*).

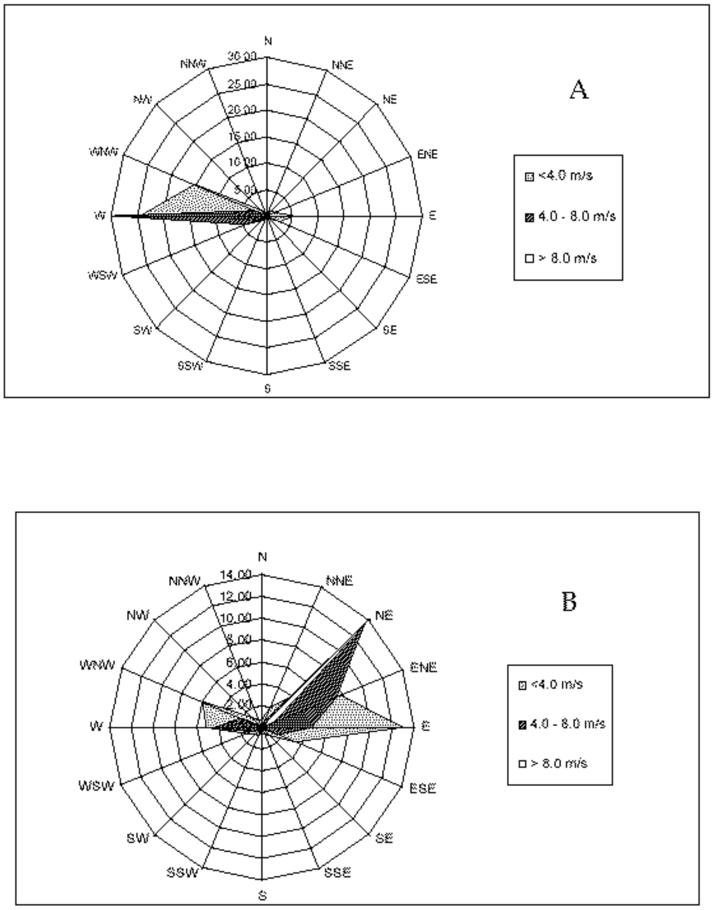


Figure 7: Frequency of wind direction at Langer Heinrich
(in terms of wind speed during A - summer periods, B - winter)

During autumn and spring the transitional state of the seasons is evident as both seasons have elements of the summer and winter wind frequency distribution.

5.3 Local Resources

The population of the Erongo region is 107,629, which is approximately 6.7% of the total population of Namibia (CBS, 2002). Most of the population is to be found in urban areas with 63% living in the towns of Walvis Bay, Swakopmund, Omaruru, Karibib, Arandis, Usakos, Uis and Henties Bay (IDC, 1995). The closest towns to the proposed Langer Heinrich mine are Walvis Bay (popn.~41,000) and Swakopmund (popn. ~25,000).

The water supply is from the Swakopmund terminal reservoir, with a water pipeline to following the district road C28 before turning north-east, following an existing dirt road into the mine site. NamWater has upgraded the terminal reservoir at Swakopmund with additional feed facilities through the upgrading of the existing Kuiseb and Omdel borehole schemes.

Swakopmund and Walvis Bay are linked by a 220 kV power line from the national grid. Power for the project is available from Walmund substation. The power supply for the site runs north from the Walmund substation until it intersects the water supply pipeline. The power line then follows the water pipeline to the site, servicing the intermediate water pump stations en route.

The granting of the Mining License for the project in 2005 also confers all necessary surface access rights to enable sustained mining operations.

5.4 Infrastructure

The national road network connects the Erongo region to the rest of the country via Okahandja, Windhoek and Otjiwarongo. The trunk roads between Windhoek, Okahandja, Swakopmund, Walvis Bay and Omaruru are tarred. Other major connections are gravel or salt roads. The gravel road C14 passes south of the Langer Heinrich deposit.

Railway connections exist between Walvis Bay, Otjiwarongo and Windhoek. This railway network connects further to South Africa. A class A airport is located at Walvis Bay. The harbour at Walvis Bay is one of the key economic features of the region. The harbour has two bulk terminals, cold storage facilities and ship repair and marine engineering services. A border post exists at the harbour as well as at the Walvis Bay airport.

5.5 Physiography

Namibia can be divided into three north-south geographic zones, which run parallel to the coast, namely the Desert Zone, the Scarp and the Plateau.

The Desert Zone commences at the coast and extends eastwards as a steadily rising plain, attaining a height above sea level of about 650 m at the base of the Scarp. The Scarp is a zone of rugged hills composed of outcropping metamorphic rocks and extends inland to around 200 km from the coast where it meets the western edge of the inland Plateau. The 1,500 m high Plateau extends eastwards uninterrupted into Botswana.

The Langer Heinrich deposit is located at the eastern edge of the Desert Zone, in the foothills at the base of the Scarp within the most northerly part of the Namib-Naukluft National Park. The project area is situated within and beneath a 1 km to 2 km wide, flat-bottomed valley, wedged between the Langer Heinrich Mountains on the north (elevation 1,152 m AMSL) and the Schiefer Mountains to the south (elevation 883 m AMSL). The mineralised valley is 710 m AMSL at its high point and descends gradually toward the west to an elevation of 550 m AMSL

The regional area is sparsely vegetated and very poorly watered. The rainfall is low and the rivers are normally dry. Occasionally stormwater entering the rivers in the upland areas reaches the sea. Perennial surface water occurs at a few points in the rivers, but subsurface water is present in the larger rivers all year.

Present day runoff is restricted to the Gawib and Tinkas rivers, which both flow into the Swakop River north of the Langer Heinrich Mountain. The Tinkas River's catches runoff from the eastern end of Langer Heinrich Valley, while the Gawib River drains the western end of Langer Heinrich Valley. The palaeochannel in which the Langer Heinrich deposit is situated extends westward beyond the Gawib River and beneath the Gawib Plain. The surface water catchment of the Langer Heinrich Valley is approximately 80 km2.

The closest surface water gauging station is on the Swakop River at Dorstrivier, north of Langer Heinrich. The flow records from this station, between 1977 and 1987 have been examined. During this period the Swakop River flowed on 3 occasions, February 1980, March 1984 and February 1985.

5.5.1 Flora and Fauna

Three major vegetation zones can be distinguished in Namibia, namely deserts, savannas and woodlands. The deserts are subdivided into the northern Namib, central Namib, southern Namib, desert and succulent steppe (winter rainfall area), and the saline desert with dwarfshrub savanna fringe. The central Namib lies between the Huab and Kuiseb rivers and the project area occurs within this zone.

The central Namib is part of southern Africa's Desert Biome. Macro-vegetation is sparse to non-existent, concentrated mostly in the Swakop River. The hills are bare rock and the valley and desert floors are covered by coarse grit, the finer sands and clays having been winnowed towards the coast by the strong prevailing north-easterly winds. Plant diversity is comparable to other desert regions of the world, but the levels of endemism are remarkably high. Just over 400 plant species, about 10% of the flora of Namibia, occur in the central Namib.

Three species of small native mammal were recorded on the site. All are relatively numerous across their natural range in Southern Africa.

Six species of large mammals were recorded. The gemsbok, springbok, common duiker and klipspringer are relatively common species in Southern Africa. Hartmann's mountain zebra, on the other hand, presently has a discontinuous distribution in small four populations, one of which is in the Naukluft mountains. The only mammalian predator recorded is the brown hyaena.

Five reptile species have been recorded and are likely to be common on the site and in suitable habitat in the wider region. Three of the recorded species are endemic. The Waterberg sand lizard however, has a very restricted range and could be of conservation concern.

Eighteen bird species have been identified although none are considered as endangered within Namibia.

The habitat and environmental features on the proposed mining site and those in the general vicinity are neither unique nor restricted in extent. Those habitats present are represented in the Namib-Naukluft Park, adjacent to the site. The total area that would be affected by the proposed mining activities is relatively small in regional terms, and has been subjected to intense exploration activities in the past.

Most of the native fauna species present at this locality appear tolerant of varying degrees of disturbance and human activity, indicated by their persistence in spite of previous disturbance. No species of conservation interest have been recorded.

6 Project History

The exploration history of the Langer Heinrich Uranium Project is summarised in *Table 4*. It includes work carried out by Gencor and Acclaim during the periods that those companies controlled the Project.

GENCOR	
1973	Discovery of the Langer Heinrich deposit.
1974-76	Extensive drilling - 25,000m percussion drilling, 2,000m diamond drilling to define the initial ore resource.
	32 exploration shafts/pits excavated for resource confirmation.
	Investigations confirmed global ore resources of 80.3 million tonnes grading 0.043% U_3O_8 for 34,520 tonnes U_3O_8 (0.01% cut-off).
	Pre-Feasibility Study completed with positive results.
1977-79	Mining, metallurgical, engineering and hydrological studies undertaken.
	Mining tests completed with excavation of two large trenches excavating 83,400t ore at 0.13% U_3O_8.
	Dry screening plant constructed to test ore processing characteristics.
	Pilot plant established and operated for an 18-month test period.
1980	Full project evaluation report completed backed up with extensive ore resource, metallurgical, mining and engineering work.
1981-1987	Extended drilling to the west that indicates additional mineralisation (Detail 7).
ACCLAIM	
1998	Acclaim acquired the Langer Heinrich Project.
1999-2002	Completed infill RC drilling and Pre-Feasibility Study with positive predicted project outcomes.
	Project mothballed due to uranium market downturn.
PALADIN	
Aug 2002	Paladin acquired the Langer Heinrich Project.
Jan 2003	Completion of a full project review and determination of a development proposal for a revised BFS.
	GRD Minproc, Johannesburg selected as engineers to manage the BFS.
Oct 2005	Infill drilling Details 1, 2 and 7 completed
Aug 2006	Plant construction commenced
Sep 2006	Infill drilling details 3, 4, 5 and 6 completed
Sep 2006	Mining commenced

Table 4: Exploration History of the Langer Heinrich Uranium Project

Details of the various stages of exploration and the programmes that were undertaken are set out below.

6.1 Discovery

Airborne radiometric surveys of the Namib were carried out by the State during 1968 through the Geological Survey of the Republic of South Africa. Based on the results of this initial survey the Ministry of Mines set aside a number of concessions for application by tender. A consortium headed by Gencor was granted the Langer Heinrich concession.

Towards the end of 1973 Gencor carried out a follow-up ground radiometric survey in the Gawib River valley, which verified the airborne results. They identified several radiometric anomalies, which coincided with outcropping occurrences of carnotite in calc-arkosic valley-fill sediments.

6.2 Gencor

6.2.1 Gencor Drilling

Gencor drilled approximately 1,800 open-hole percussion drill holes on a 50m x 50m or 100m x 100m grid-pattern. These holes were drilled to delineate the geology and structure and to allow an estimate of the uranium resources of the Project. They remain the main source of information for the Langer Heinrich Deposit. Drill cuttings were analysed by XRF.

This drilling could not penetrate either the unconsolidated gravel in the present-day, dry Gawib drainage channel or the area beneath some of the conglomerate terrace material. In these areas the orebody remained untested. Delineation of mineralisation below the water could not be established as the wet sandy clay caved in due to groundwater influx. Potential of increasing defined reserves therefore exists as a considerable part of the orebody remains untested.

Gencor drilled 72 diamond drill holes in order to:

- determine the accuracy and limitations of open-hole percussion drilling by twinning a number of the percussion drill holes;
- provide a means of estimating uranium resources below the water table;
- allow estimation of uranium resources, in lateral extensions, beneath the Gawib River bed. (These holes were used in Gencor's 1981 & 1983 internal ore reserve estimates when the high-grade resources of the deposit were calculated); and
- give details of distribution and characteristics of uranium and rock types throughout the deposit.

Core recovery in the early holes was poor in the unconsolidated sections. Later triple tube core barrels were employed and recovery improved with better equipment and drilling practices. Diamond drilling gave precise contacts to mineralisation and a more accurate assessment of grade.

Poor-repeatability of assays from percussion drill hole samples was recognised in the Gencor drilling programs. To evaluate the extent of this problem a program of excavating small pits or shafts was undertaken in 1976. Gencor compared assayed grades of percussion drill holes to the excavated grade of material from 32 pits (shafts) excavated through the mineralisation. The 2m x 1m pits were dug in groups of four, one located at each corner of a 100m x 100m square.

Each pit was sited with an original percussion drill hole in its centre. A further four percussion holes were drilled on the corners of the proposed pit. The percussion drill holes were sampled at 1metre intervals.

A 4.2 tonne bulk sample was collected from each one metre depth interval in the mineralised section of each pit (mineralised sections ranged from 6m to 22m depth) and transported to the bulk sample stockpile site, located nearby. The bulk samples were crushed, coned and quartered, milled and riffled down to 5kg on site. Grab samples, cone and quartered samples and riffled samples were all sent to Group Laboratories in South Africa for analysis.

Statistical evaluation of the pitting and percussion results were interpreted at the time to show that the percussion drilling defined the grades of the mineralisation within a 95% confidence level. It was therefore concluded from this pit sampling programme that the average percussion drill sample analysis is accurate enough to provide an accurate resource estimate for the Langer Heinrich Deposit.

6.2.2 Gencor Trial Mining

Trial mining exercises were undertaken involving excavation of a Mega-Trench and a Test-Pit. These campaigns enabled Gencor to develop satisfactory mining and grade control methods.

The objective of the Mega-trench project was to:

- Determine the distribution of carnotite and its relation to rock type;
- Assess the degree of ore upgrading possible by selective mining;
- Determine the blasting and physical characteristics of the $CaCO3$ cemented clastic sediments;
- Provide ore for a pilot plant;
- Determine the physical characteristics of the ore;
- Determine the clay content and distribution following evidence that clay had a deleterious effect in the liquid-solid separation phase;
- Evaluate grade-control methods using an on-the-spot scintillometer;
- Determine the optimum mining techniques for both clastic sediments and the underlying sandy-clay;
- Assess potential problems in mining that could arise from the presence of groundwater; and
- Aid the selection of mining methods and equipment.

The site for the Mega-Trench was chosen as having ore of low and high grade, thicker than 30m, high and low clay content, ore above and below the water table and ore resting on elevated basement.

Prior to excavation, the centre-line was drilled at 10m intervals, down to bedrock. Drill hole samples from above the water table were assayed by XRF. Down-hole radiometric logging was undertaken so that uranium grade could be estimated in the gritty-clay below the water table.

Blast-holes were drilled on a staggered 3m x 1.5m pattern down to the water table. The water table was between 9m and 14m (in the north) below ground level. Drill hole chips were radiometrically analysed and the drill holes which went beyond the water table were all radiometrically logged down hole. In addition samples from blast-holes, spaced every 6m x 6m were, analysed for U_3O_8, V_2O_5, Cl and suspended solids (clay). This selective assaying resulted in samples from 25% of the blast holes drilled being analysed.

In addition 70° (from the vertical) "smooth-wall" blast-holes were drilled at 2m intervals along the side of the pit. The complete excavation was blasted to the water table prior to excavating and loading. Re-blasts were necessary in several areas to render the ore and waste rock manageable with the small-scale equipment in use.

From June to October 1977, 100,486 tonnes of ore (@ >0.1kg/t U_3O_8) was excavated from the north-northwest striking mega-trench. The trench had planned dimensions of 280m x 22m x up to 23m depth and was excavated to 8m below the water table.

The excavation method used was to rip and load 1m thick horizontal benches to avoid mixing and allow an accurate assessment of grade distribution and effectiveness of selective loading.

Trench Grade Control

After blasting the trench floor was T-probed on a 1m x 1m pattern and marked into areas of different grade. Ore was excavated in 1m thick benches according to its grade category and lithological category. Each lithology and grade category had a specific dump and in total 23 ore dumps were established. Rock <0.1kg/t U_3O_8 was dumped separately as waste.

Truck Grade Control

Each truckload of ore was T-probed at five different points of the load followed by Eberline scanner monitoring. Every 10th truck was sent to the sampling plant and a "representative" grab sample was analysed by XRF.

Systematic weighing of trucks during the Mega-Trench programme gave a dry bulk density, which varied from 1.95 for clay through to 2.25 for the CaCO3 cemented clastics and averaged 2.1g/cm3 for all ore extracted.

During the trenching programme, several small perched pockets of water were encountered. The maximum water flow was 2m3/hour at 606.5m AMSL proving that the rocks have low permeability. Water levels within the excavation rose at a rate of 4cm/day during the first 8 weeks following excavation.

During the trenching operations, water became a problem as the area of the trench was too small to excavate a sump from which to de-water. On a larger-scale operation it was considered water will not offer any significant difficulty to mining operations.

The Gencor conclusions of the trenching operations were as follows:

- sediments occur as sub-horizontal layers 0.5m to 6m thick with undulating contacts;
- there is an increase in grain size and CaCO3 content upwards;

- carnotite is the only uranium mineral present and occurs throughout all clastic rock types. It also occurs in joint and planes of basement rocks but this is not commercially significant;
- natural cut-offs for uranium are 0.1 and 0.3 kg/t; U_3O_8
- estimation of grades using 12.5m x 12.5m grid drilling compared to T-probing following blasting in the trench showed a correlation of ±10%;
- percussion drill holes at 10m x 10m spacing gave excellent correlation to mined ore;
- dilution was 5% with inclusion of internal waste in the ore;
- selective loading is possible excavating 1m benches. Selective loading using thicker benches is possible but must be controlled by drill holes as T-probes only read to 0.2-0.5m below the pit floor;
- bulk mining using 2m to 5m benches is possible but involves trade-off for selective mining and dilution;
- trenching confirmed grade distribution and provided material for pilot plant testing along with correlation of percussion-hole indicated ore reserves to mineable reserves;
- radiometric estimation of uranium grades from trucks is reliable;
- pit-walls at 70o are very stable;
- moisture content of ore was 2% above and 10% below the water table;
- water inflows are not expected to cause any problems during mining operations; and
- dewatering prior to mining is necessary.

The Test-Pit was excavated from May to November 1978 to evaluate on-site upgrading methods using selective loading, screening and radiometric sorting techniques. The pit was sited immediately alongside and to the east of the mega-trench. A total of 80,000t of ore was mined of which 40,000t was passed through a pilot crushing, milling and screening plant erected on site.

The main objectives of the Test-Pit and dry-screening plant operation were to:

- test the effectiveness of screening and radiometric ore sorting;
- provide ore for the pilot plant;
- develop grade-control techniques and evaluate radiometric grade determination;
- develop an understanding of any potential mining problems;
- evaluate exploration and production drilling; and
- evaluate selective loading, blasting, ripping and demarcation of grade boundaries.

This work was geared specifically to enable the design of a model on which mining operations at Langer Heinrich could be based.

The survey system used for the Test-Pit was the same as that utilised for the Mega-Trench. The 00m baseline defined the centre line of the Test-Pit and heights were reduced to Mean Sea Level. The Test-Pit and Mega-Trench were separated by a wall of in-situ bedrock several metres thick. The area selected for the Test-Pit mining was a high-grade portion of the orebody where lithological units are more gritty and sandy than other parts of the Deposit.

The area of the Test-Pit was percussion drilled in early 1978 on a 12.5m x 12.5m grid. Drill-cuttings were radiometrically analysed and assayed by XRF where mineralised.

Almost 75% of the material mined required blasting, the most efficient method being blasting of large areas in situ. Blast-holes were drilled on a 2m x 2m square pattern. The intention was to break all material to -160mm in situ, but this was not achieved and oversize material became a problem in the screening and sorting tests.

The mining contractor, LTA Construction, carried out the blast-hole drilling, blasting, and excavation of the 30m x 100m x 12m deep pit. An International TD25 bulldozer equipped with a one metre single-tyne ripper easily ripped semi-consolidated sandy grit, but lenses of calcareous cemented material caused difficulties.

Ground surface was at an average 626.5m AMSL. An average of 2m of soft sand and loam material was stripped as overburden and taken directly to waste dumps during May and June 1978.

Ore was removed in 1 metre slices from 623m AMSL down to the 611m AMSL. In situ T-probing on a 2m x 1m grid, ripped by the bulldozer, determined grade and delineated blocks for selective loading. Waste, low-grade and high-grade areas were physically demarcated. Grade categories were the same as those used for the Mega-Trench.

Selective loading was carried out using a Poclain RC 200 face-shovel and 10-ton dump trucks. Each truckload required three full shovels of material. Loaded trucks passed under an Eberline Scanner and the loads were T-probed to confirm grade. Grades derived from T-probing of the loaded ore proved good for establishing the grade of run-of-mine product but were not acceptable for screened products. A combination of truck scanning and T-probing was found to be most effective.

The loads were then sent to waste/low-grade dumps or as feed for the screening plant. Every 10th load was weighed on a weighbridge to confirm tonnage calculations.

Major geological boundaries were demarcated using steel droppers. Each lithological ore type was loaded separately in order to test the screening efficiency of various lithological units. Metal balance registers, using radiometric and chemical sampling, were kept to quantify the efficiency of operations from in-pit selective loading to loading the products of screening and sorting.

The conclusions of the test mining operation from the Test-Pit were as follows:

Resource estimation

- the 50m x 50m exploration drilling underestimated the Test-Pit ore reserves; and
- the 12.5m x 12.5m drilling predicted ore reserves accurately and is considered to be the optimum drill hole spacing for mine planning.

Radiometric estimation of U_3O_8 grade

- portable radiometric analytical devices (T-probe) were successful;
- radiometric bore-hole logging was a reliable method of grade evaluation;
- the Eberline truck-scanner was a highly stable, reliable and sensitive grade-evaluation instrument;
- a combination of T-probing and Eberline Truck Scanning was found to back up selective loading, adding greatly to the upgrading of excavated ore; and

- permanent grade control points with scanners and weighbridges should be used as primary checkpoints for confirmation of load grades and tonnage estimation.

Selective loading

- in spite of difficulties caused by poor fragmentation of the cemented conglomerates and grits, selective loading definitely achieved higher grades of ore being fed to the screening plant.
-

Dilution

- dilution was less than 10%.

6.2.3 Gencor Screening and Process Trials

The Screening Plant had a design capacity of 250 tonnes per hour, however, during operation it averaged 80 to 100 tonnes per hour. Over the four months of testing from July to November 1978 40,000t of ore was screened.

The objectives of the operation were to:

- Quantify upgrading by screening of various feed head grades;
- Test the efficiency of belt sorting;
- Supply the pilot plant with upgraded fines for metallurgical tests;
- Supply minus 25mm to minus 160mm material for lump sorting tests; and
- Establish the operating efficiency of the screening plant design.

Conclusions of the screening plant operation were that:

- Carnotite concentrated in the finest fraction, (<3mm) except where there is a high proportion of clayey material;
- Upgrading in the fines was achieved throughout the project;
- A large proportion of the carnotite remained in downgraded +5mm to -25mm and +25mm material;
- Conglomerate ore is best suited to upgrading by screening as the boulders and pebbles containing no U_3O_8 are easily screened out;
- Coarser grits responded well with large amounts of quartz and feldspar in the grit fraction reporting to the +5mm -25mm fraction;
- Calcareous grits were found to have a high-grade component in the +5mm to -25mm fraction;
- Screening of sandy ore was only partially successful due to high fines content;
- Silt and fine sandy ore were not suited to upgrading;
- Simple screening upgraded the ore by a factor of 1.92, however a considerable amount of metal was lost to the waste in this process;
- A moisture content >5% made screening almost impossible;
- Radiometric belt sorters were not particularly effective as the ore was mixed during mining, loading and screening operations; and

- The belt sorter only separated lower grade ore when the grade of the whole truckload was low. Belt sorters remain a potentially advantageous method of feed grade control.

Gencor constructed a one tonne per hour alkaline leach pilot plant. The objectives of the pilot plant study were as follows:

- Establish a grindability index for the ore;
- Confirm the leach parameters established during the laboratory investigations;
- Establish the settling and the checking characteristics of the leach slime;
- Test ion exchange in unclassified pregnant solutions; and
- Test the most effective uranium precipitation methods.

The pilot plant was run over a twelve-month period and resulted in identification of a coherent flowsheet for the project. Liquid-solid separation remained problematic.

6.3 Acclaim Exploration 1999

Following acquisition of the Langer Heinrich Uranium Project Acclaim drilled and re-evaluated the Deposit and completed a positive Pre-Feasibility Study incorporating the results of the additional drilling for resource verification

The primary effort of Acclaim centered on confirming the high-grade resource that had been identified by Gencor. Production using the high-grade portion of the orebody, particularly in the early years, was seen as a key component to bring the deposit into production

With the majority of the old Gencor data acquired, Acclaim's main objective was to prove that the Gencor data could be used and incorporated with data generated by Acclaim in 1999. The Gencor information was entered into a digital database along with data collected in the 1999 programmes. Validation of the old information involved confirming the location and accuracy of all sampling points and determining the accuracy of all sample results.

Surveying was undertaken by Acclaim to validate Gencor's survey base station network and to confirm the location and accuracy of drill holes, pits, Mega-Trench, Test-Pit etc. Internal and external checking procedures of the Gencor laboratory in Springs, Johannesburg were sought and confirmed. Calibration methods and check procedures were confirmed for down-hole radiometric logging. Bulk dry and wet densities were confirmed. Reverse circulation drilling was undertaken to confirm the validity of results form Gencor's percussion and diamond drilling and pitting and to confirm their "reserve estimates".

6.3.1 Acclaim Surveying

Willem Knotze Professional Land Surveyors of 10 Eugene Marais Street, Windhoek, Namibia carried out an evaluation and field check of Gencor's survey work.

A differential global positioning system (DGPS) using total-stations and precise levelling confirmed the high accuracy of seven original survey control stations within the Langer Heinrich valley by surveying and tying into three external Trig stations.

Positions of all Gencor's surveyed points were found to be of a high accuracy. Gencor's plotted positions for percussion drill hole collars agreed with their newly surveyed positions, with a mean error of 545mm within the horizontal plane and 419mm in the vertical plane. Diamond drill hole collar coordinates were found to be precise with errors of less than 100mm. Positions of pits, the Mega-Trench and Test-Pit were similarly found to be surveyed to a high level of accuracy.

Acclaim's 1999 RC drill hole collar positions were surveyed to an accuracy of ±40mm by the DGPS survey work.

Gencor had carried out a low-level aerial photographic survey in 1975. Using the survey control network established at that time (white painted rock aerial markers which are still in place today) a 1:5,000 topographic plan with one metre contours was produced by photogrammetry. A paper print of this plan was scanned, vectorised and tied into the survey control points established by Gencor.

6.3.2 Acclaim Reverse Circulation Drilling

The objectives of Acclaim's 1999 drilling programme were to:

- Confirm the accuracy of the Gencor percussion and diamond drill results;
- Prove that the high-grade core in this part of the deposit was continuous and could be mined as an entity;
- Confirm the size and grade of extensions of mineralisation beneath the current Gawib River drainage channels and below the water table, as predicted from widely spaced diamond drilling carried out by Gencor; and
- Drill out one area on a close-spaced grid pattern to allow a comparison to be made between resource estimates based on Gencor's drill holes and Acclaim's 1999 RC drill holes and the confirmation of both.

A total of 107 RC percussion holes were drilled by Acclaim between 28 May and 29 June 1999. Holes were positioned on the historical grid lines midway between Gencor holes. A number of holes were drilled alongside selected Gencor percussion and diamond holes with some holes being drilled down the existing Gencor drill holes.

The drill rig used was a reverse circulation air-core unit using a 1995 Hydco multipurpose top head drive rig mounted on a late model MAN 4x4 truck with a planetary drive train. The depths of holes varied between 16m and 45m.

Drill chips were collected in a cyclone over 1m depth intervals into pre-numbered polywoven sample bags. Each sample was weighed, lithologically logged and the total-count radioactivity reading noted (using a 1 sec count on a Scintrex GIS 5 hand-held scintillometer). An Auslog, calibrated, total-count radiometric logging system was used by Acclaim as the primary tool to determine equivalent U_3O_8 grade.

The downhole logging succeeded in accurately locating the boundaries of the U_3O_8 mineralisation and has excellent repeatability (shown by multiple recordings). XRF assays from both historic and 1999 drilling compared closely with grade determined from radiometric down-hole logging with differences in modelled grade of only 6%.

Sample selection for chemical analysis was guided by the radiometric down-hole logging results. All samples from mineralized intersections were submitted for analysis including an un-mineralised sample 1m either side of the mineralized interval. Each entire 1m sample was dispatched to SGS analytical laboratories in Springs RSA for analysis.

6.3.3 Paladin 2004 Infill drilling

Paladin undertook infill Reverse Circulation (RC) drilling during the latter half of 2004 (Rich, 2004). This program had a number of objectives:

- Infill existing Gencor and Acclaim drilling within Detail 1 to 50x50m sufficient to meet JORC requirements for "indicated" reserves particularly below the water table.
- Test the theory that the palaeochannel thalweg extends due west from the Detail 1 test pit area to Detail 2.
- Obtain bulk sample material for bench scale process testing.
- Obtain samples from above and below the water table for disequilibrium studies.
- Drill geotechnical holes for the mill and tailings sites.

Drilling statistics are detailed below in Table 5, Two drilling contractors were used, Resource drilling and Drillcon Africa.

	Start	End	Days	Holes	Metres	Average
Resource	September 6	November 16	72	95	4148	57.6/day
Drillcon	September 29	November 20	53	71	2572	48.5/day
Totals				166	6720	

Table 5: Exploration and evaluation history, summary

The drill crews commenced at about 6am and often worked through to 8pm or later, 7 days a week. The low average daily production rates are indicative of the difficulties both contractors encountered, particularly associated with the ground conditions, deep soft sands and down hole drilling conditions, unconsolidated boulders at surface and unconsolidated to weakly consolidated sediments at depth.

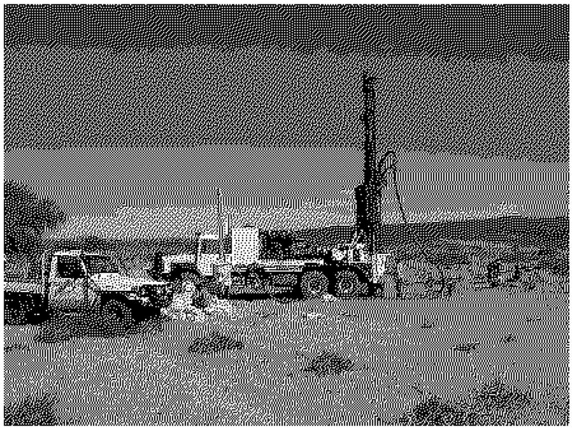
6.3.4 Evaluation

Resource Drilling provided a Smith Capital 14R6H mounted on a Samil 6x6 truck along with a Samil 6x6 support vehicle, Drillcon Africa provided an Ingersol Rand S4W rig but mounted on a double rear wheel drive Ford flatbed truck. The compressor was separate, mounted on a 4x4 Samil along with a 4x4 Samil support truck.

Both rigs used rods of identical outer and inner wall thickness, thereby allowing the same casing factor to be used for all radiometric logging. The Resource Driiling holes were collared with 140mm thick walled (about 10mm) PVC. This was robust enough to allow casing to be advanced with the hammer. Drillcon Africa used thinner walled 140mm PVC and had much trouble with the PVC shattering during the collaring process.

Resource Drilling utilised a drill rig mounted Atlas Copco compressor with common intake manifold for the engine and compressor, the Drillcon Africa drill rig used an Ingersol

Rand compressor which being separately mounted could be parked at some distance from the drill rig and provided a better quality compressed air supply.


Resource Drilling RC rig

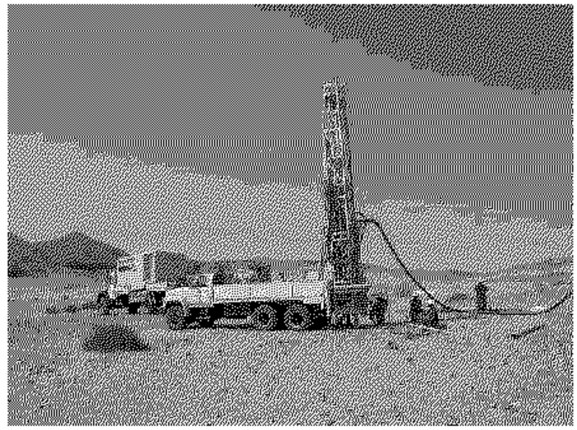

Drillcon Africa RC rig with separate compressor

Figure 8: Drill Rigs used by LHU

Geologically the Detail 1 & 2 areas proved extremely difficult to drill. There were two principal problems:

- unconsolidated boulder conglomerate at surface
- unconsolidated to weakly consolidated sediments at depth

Boulder conglomerates occur at surface and to a depth of about 6m at the eastern end of Detail 1. Only Drillcon Africa encountered these conglomerates. They contain boulders to 0.5m diameter and form an unconsolidated (non calcreted) surface deposit. The problem for the driller is that he must try to drill a hole through this and seat the casing in competent (calcreted) channel sediments. The drill hammer tended to slip off boulder faces, boulders shifted and frequently the upper 1 to 2 m simply blew out leaving a large crater. In these cases the hole was abandoned and redrilled. The second problem was knowing where competent ground commenced. Drillcon Africa tended to case off too early resulting in blow by around the outside of the casing. Again several holes had to be abandoned when this became too extreme to continue.

Unconsolidated sediments at depth were the most serious problem. Gencor were well aware of this reporting that the presence of incompetent, soft, malleable when wet, sediments described as clayey siltstones, found in drill holes below the present day calcareous and gypsiferous duricrust are of importance to both the extraction metallurgy and to mining. The current drilling indicates that these sediments are both clayey and in some instances simply well sorted fine sands and silts. These are often "running" sands which bind around the rods preventing outside return, typically 90% of the air is returned with the sample up the inner annulus, the remaining 10% returns outside the rods to keep the drill hole clear.

Because large quantities of air have been injected into the formation it is usually under pressure. On numerous occasions running sands ran up 2 or 3 rod lengths necessitating tripping the rods and cleaning them out. Even where this material does not run, the drillers still found it very difficult to maintain outside return. Difficulties were often experienced withdrawing the hammer as material built up above the hammer prevented it entering the hole in the overlying competent calcreted sediments.

Clay layers were also a problem. Generally they resulted in the hammer being blocked and having to be withdrawn and cleaned out (often numerous times). Often this occurred right above the basement contact (in some cases in deeply weathered basement saprolite) although puggy clay layers also occur in the channel itself.

Ten compartment plastic chip trays were used to collect samples for the Namibian Department of Minerals and Energy (DME). A sample was collected for every metre drilled. This process was designed to allow the DME to collect a number of reference samples over the deposit.

All holes were radiometrically logged through the rods. All logging was done with standard Auslog 43mm probes. In a number of holes radeometric logging could not be completed to the bottom due to the probe hanging up on dirt or grit in the inner tube. In such cases the probe was removed, and rods reflushed with air. Normally this cleared the blockage however in a few instances the radiometric log had to be terminated above basement. Radiometric data was e-mailed to Australia every evening and deconvolved data was available the following morning Namibian time.

Holes LH 1001 to 1025 were riffle split in their entirety. This proved too cumbersome and slow so for holes LH 1026 onwards only the sections to be sent for geochemical analyses were split. With one rig the splitting crew could keep up with the rig however after two rigs commenced, the splitting crew worked independently. Three identical sets of samples (average sample weight 400-500 g) were split for each metre to be analysed. These were labelled 'A', 'B' & 'C'. The 'C' samples were sent for assay with duplicates being drawn from the 'B' set. The remaining 'A' set was set aside as a reference.

Samples were prepared for submission to the assay lab as a separate task after the drilling was completed. Samples were laid out in order and blanks inserted after the mineralised peaks (located by consulting the deconvolved eU_3O_8 and confirmed by scintillometer). These blanks comprised dune sand collected south of Swakopmund. Under the binocular microscope this material appears to be well sorted quartz sand with minor iron oxide grain coatings. It assays below LLD (<4ppm U_3O_8).

7 Geological Setting

7.1 Regional Geology

The Swakopmund environ forms part of the Damara Belt eugeosyncline. The oldest beds consists of psammitic rocks of the Nosib Group overlain by several thousand metres of pelitic rocks of the Swakop Group and the Khomas Subgroup all of Neoproterozoic Age. Folding, combined with regional granitisation took place between 650 and 500 Ma ago. Some of these orogenic granites, for example Rössing, are uraniferous. A number of subvolcanic to volcanic non-orogenic complexes were emplaced 120 Ma ago and huge volumes of basaltic lava were extruded. Remnants of this basaltic cover have been preserved and are up to 800 m thick. Weathering and erosion of the uraniferous granites was the source of uranium that precipitated to form secondary deposits such as Langer Heinrich.

The lowermost rocks of the Damara Sequence, the pink quartzite of the Etusis Formation of the Nosib Group, form the Langer Heinrich Mountain anticlinorium, a major structure of the area. Overlying these quartzites are schists comprised of rhythmically interbedded fine-grained metapelite, metagreywacke and calcsilicate beds. Collectively these form the Tinkas Member of the Khomas Subgroup, with a maximum thickness of 3,000 m. The uranium mineralised Langer Heinrich palaeochannel is principally eroded into these schists. Locally the base of the Khomas Subgroup is represented by glacial marine sediments of the Chuos Formation of which the thickness varies from 0 to 250 m.

The orogenic Salem granite has intruded the metasediments and covers large areas north of the Langer Heinrich Mountain. Southeast of the mountain, the Bloedkoppie granite, a leucocratic late-to-post-tectonic member of the Salem granite suite, has intruded the metasediments and covers an area of about 25 km2. A portion of this granitic area forms the catchment drainage for the Langer Heinrich valley. On average, it contains 10 – 15 ppm U_3O_8 and values up to 100 ppm U_3O_8 have been measured radiometrically. These rocks are believed to be the source of the uranium in the Langer Heinrich deposit. The Bloedkoppie granite is of the same age as the alaskite of the Rössing mine region and may be genetically related.

The uranium mineralisation is calcrete related and occurs in valley-fill sediments within an extensive roughly east-west tertiary palaeodrainage system. The calcrete consists of interstitial calcium carbonate that was precipitated under arid to semi-arid climatic conditions.

The uranium occurs as carnotite, which is a secondary oxidised carbonate mineral containing both uranium and vanadium. The deposit occurs over a 15 km length with seven drill-indicated higher grade pods (Details 1 to 7, *Figure 9*) occurring within a lower grade mineralised envelope. The carnotite occurs as thin films filling cavities and fracture planes and as grain coatings and disseminations in the calcretised sediments. Mineralisation is near-surface and and the host sequence varies from 0 to 60 m thick, and between 50 m and 1,100 m wide, depending on the width of the palaeovalley.

After calcrete development and mineralisation, parts of these sediments were eroded as a result of uplift that caused rejuvenated river flows - the largest being the Gawib River that in part follows the palaeovalley and has dissected and modified both the calcrete and associated mineralisation. Headwater erosion by the Tinkas and Gawib rivers has eroded 30 m – 40 m of the original palaeovalley sedimentary cover along the central and eastern portion of the orebody, and river capture by the Tinkas River has reversed the direction of the drainage system so that today the most eastern part of the orebody drains eastward into the Tinkas River. A portion of the orebody has also been removed by this erosional process. Where not dissected by these ephemeral drainage systems, the deposit is blanketed by variable thicknesses of river sand and scree. For instance, in Detail 7 this cover exceeds 40 m.

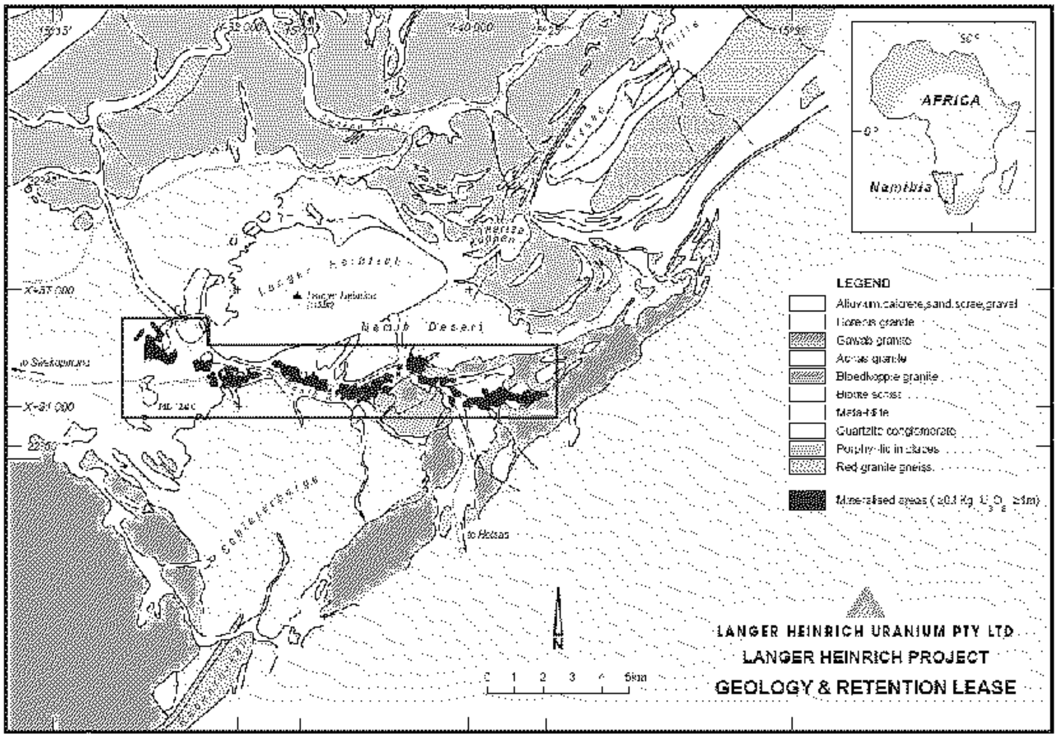


Figure 9: Geological Setting

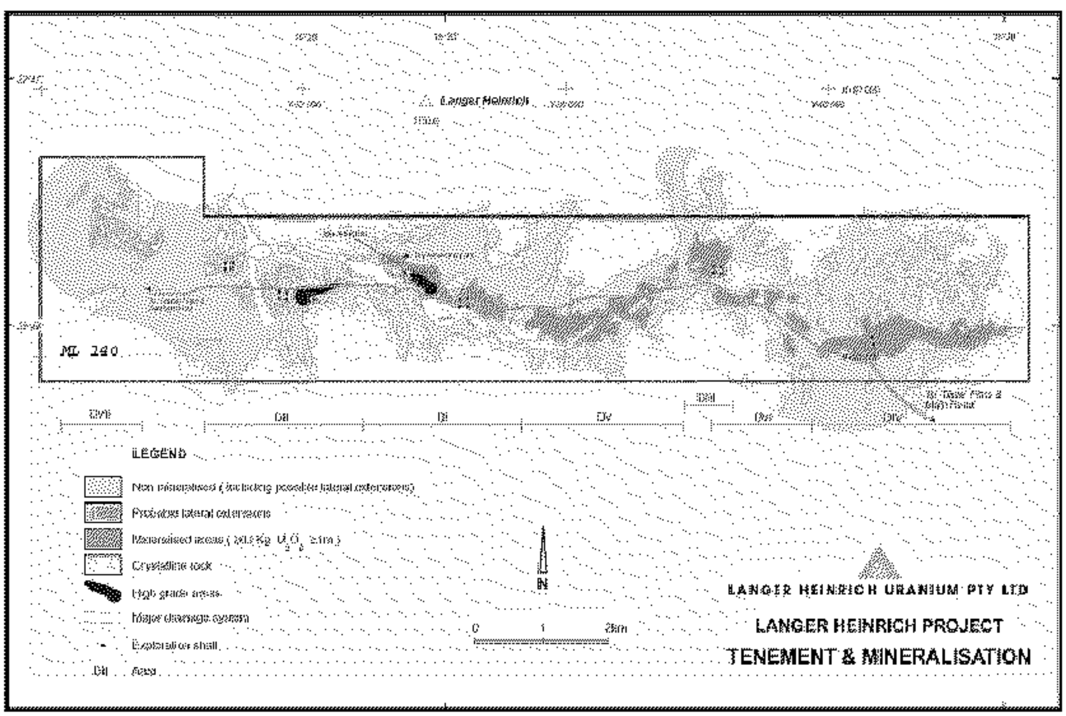


Figure 10: Local Geological Setting

8 Deposit Type

Langer Heinrich is a calcrete related uranium deposit associated with valley-fill sediments occurring within an extensive Tertiary palaeodrainage system. The calcretes are limestone deposits formed as chemical precipitates developed under arid to semi-arid climate conditions. At Langer Heinrich calcretisation has affected a complex sequence of fluvially derived conglomerates, grits, sandstone, silts and clay deposits worked in a braided stream depositional environment.

Beneath the sediments is a rugged basement topography, which occasionally rises high enough to form outliers exposed above the valley sediments. As the basement rocks alternate between erosionally resistant and softer lithologies, the valley width changes from 1 km to 2 km wide, to narrow gorges only a few hundred metres wide.

The fluvial sedimentary sequence comprising the Langer Heinrich Formation is up to 100 m thick and comprises clasts of angular to rounded basement debris in alternating bands of conglomerate, grit, sand, clay-grit and clay. These sediments have undergone variable cementation by calcium carbonate ($CaCO_3$) known as calcrete that precipitated from groundwater moving down the valley. This $CaCO_3$ can comprise up to 15% of the total rock mass.

Uranium mineralisation has been defined along 15 km of the east-west trending palaeovalley and is nearly continuous along this section of the palaeovalley system. The mineralisation is still open to the west where the cover is in excess of 40 m.

9 Mineralisation

Mineralisation at Langer Heinrich consists of sub-horizontal carnotite that has been precipitated within clastic valley-fill rock units, which vary from conglomerates through grits and sands to micaceous claystone. In a general way, ore becomes finer grained with depth and hence ore from beneath the water table is mainly micaceous claystone.

Carnotite is the only uranium mineral reported at Langer Heinrich. It occurs as finely disseminated specks, as blebs up to 20 mm thick and coatings in open pore spaces, which are irregularly distributed within the matrix of all host lithologies within the valley. The carnotite occurs preferentially in the less cemented portions of the host sediments.

The mineralisation occurs as an undulating 1 m to 30 m thick layer, shaped like a subterranean meandering river. At intervals down the valley, thick "pond-like" pods of higher-grade uranium have formed. These are generally located immediately upstream of a narrowing of the valley. These high-grade pods have probably formed where depositional solutions have ponded in basement depressions. The colour scheme for *Figure 11* is as follows: grey represents 0-250ppm, blue 250-400ppm green 400-6500ppm and red >650ppm and the panels are scaled to the mineralisation proportion at a 250ppm cut off.

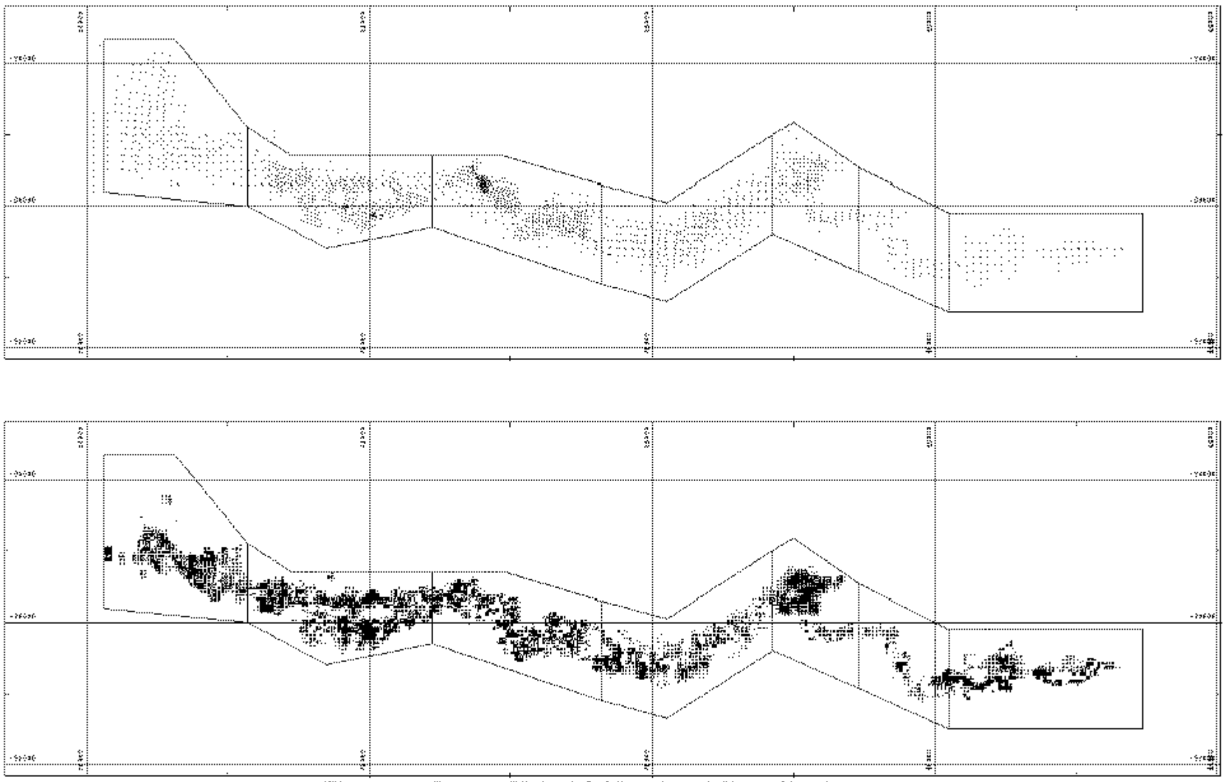


Figure 11: Langer Heinrich Uranium Mineralisation

Grade distribution within a layer, in cross-section, is as a series of broad concentric ellipsoidal shells, with highest grades in a central core and decreasing towards the bottom, top and banks of the channel. Some of these findings may be a function of the drill hole spacing and/or lack of drilling in specific areas. Within these mineralised ellipsoidal shells grade distribution in detail is very erratic or nuggetty. However, the overall continuity of the mineralisation in plan view is quite strong.

Calcium carbonate is an important constituent and the ore can be divided into low and high carbonate categories. Calcium carbonate occurs in concentrations from 5% up to 35% and cements clastic particles together until, at high concentrations, an indurated rock has been formed. The indurated layers are not related to lithology and tend to form sub-horizontal layers of varying thickness. The hard carbonate-cemented layers contain disseminated carnotite, but generally at lower concentrations than in the more porous less cemented sediments.

Below the water table, mineralisation appears to be hosted in an unconsolidated micaceous silt-sand-clay. Mineralogical investigation of the mineralisation shows uranium present only as carnotite, the potassium uranyl-vanadate mineral $[K_2(VO_4)_2(UO_2)_2 \cdot 3H_2O]$. Carnotite is interstitial to the clastic grains, generally as fine-grained flakes, though larger blebs, clumps and open-space coatings are present. Carnotite occurs preferentially in less cemented sections of the ore but shows no relationship to any rock type. It does not occur within the matrix of any of the clasts, whether boulder, sand, or silt size.

10 Exploration

Exploration activities by LHU have been restricted to infill drilling of all Details to increase confidence in the estimation of the mineralisation below the water table. This drilling is detailed in section 11 below.

11 Drilling

11.1 Paladin 2005 Infill drilling

11.1.1 Scope of Drilling Programme

The scope of the drilling programme included drilling work required for planning purposes, infill to increase confidence in resource model and exploration. The drilling was undertaken in the areas Detail 1, 2 and 7 and the drilling was commenced on 15th June by R A Longstaff (Namibia), using a Super Rock 1000 rig adapted for RC drilling and towing a separate Atlas Copco 1000 cfm x 260 psi compressor (*Figure 12*). PVC collars were installed to 6m depth using a conventional hammer and holes were completed using a Reverse Circulation (RC) hammer.

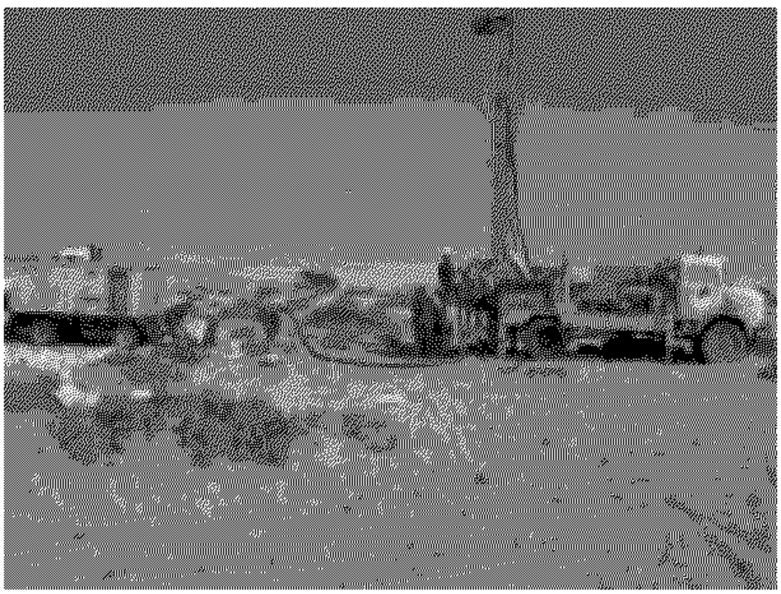


Figure 12: Drill rig on site. Super Rock 1000 rig adapted for RC drilling, towing a separate compressor

Drilling commenced in the north western part of Detail 2, moving westwards into Detail 7. Initially, the drilling progress was slow, since the rig was run by an inexperienced crew and driller, who had problems drilling the difficult ground. The major problem was unconsolidated to weakly consolidated and generally clay rich sediments at depth. However, the production rate improved considerably after an experienced driller was brought to site. Additionally, a second rig was brought onto site in July to prepare the collars for the RC rig. A total number of 244 boreholes and 51 pre-collars resulting in a total number of 11826 m have been drilled along the Detail 1, Detail 2, and Detail 7 areas during this year's RC drilling programme in the period from 15 June to 2 November 2005. The large number of "unused" pre-collars results from the fact the pre-collaring rig collared all planned holes (up to ten in each line). However, results from completed holes have often shown that only three or four holes per line were necessary to define the channel.

All holes were geologically and radiometrically logged. Samples were collected through a cyclone for each meter drilled and split through a riffle-splitter by the drilling crew to give a sample for assay of 3 to 4 kg. Rods and hammers have been lost in holes LH1667 & LH1168. After the completion of the drilling programme the drillers were able to retrieve rods and hammers from both holes.

11.2 Paladin 2006 Infill drilling

11.2.1 Scope of Drilling Programme

The scope of the drilling programme included drilling work required for exploration purposes in order to increase geological confidence, establish indicated mineral resources and increase inferred mineral resources for Detail 5, 3 ,4 and 6.

In the run-up to the drilling programme an electromagnetic survey was conducted by Bittner Water Consult with the aim to localize the main course of the Langer Heinrich palaeo-channel (BIWAC 2006). The drilling work commenced on 5th July by RA Longstaff Namibia (Pty) Ltd, using a Super Rock 1000 rig adapted for RC drilling. An Atlas Copco 1000 cfm x 260 psi compressor was towed separately. A total number of 231 boreholes resulting in a total number of 6355 drill meters have been drilled along Detail 3, 5 and 6 between 5th July and 15th September 2006 (Fig. 1). Initially, boreholes were drilled along 200 m line spacing with a 100 m distance between the boreholes of each line. Boreholes of this first phase of the drilling intersected the basement. Later, infill drilling along 100 m line spacing was conducted in the mineralized areas, which were defined during the first phase of drilling. The boreholes of the second phase of drilling intersected the mineralized zone, but did not intersect the basement in order to save drilling meters. The boreholes were between 4 and 51 m deep with an average depth of 27.5 m. The drilling progress was good with an average daily drilling rate of 119 m; only minor problems with weakly unconsolidated and clay-bearing sediments were encountered. No drilling equipment was lost during the campaign. The boreholes were geologically and radiometrically logged. Drill chip samples were collected through a cyclone for each meter drilled and split through a riffle-splitter by the drilling crew to give a sample for assay of 3 to 5 kg.

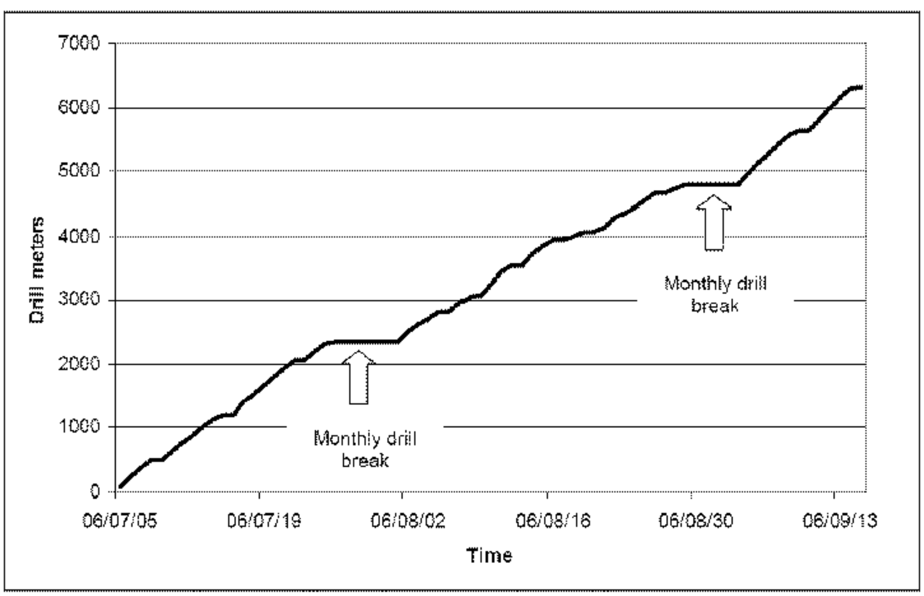


Figure 13: Cumulated metres drill during 2006 campaign

11.2.2 Safety

Both, LHU and RA Longstaff complied with safety standards and wore long trousers and long sleeve shirts as well as appropriate personal protective equipment (PPE) including hard hat, ear plugs, dust masks, safety glasses and safety boots. Radiation exposure was monitored during the drilling campaign. Each member of the drilling crew as well as the LHU staff were equipped with personal dosimeters.

12 Sampling Methods

12.1 Paladin 2005 Infill drilling

12.1.1 Lithological logging

All boreholes were geologically logged. At the drill site, samples were checked for their colour, moisture, weight, and HCl reaction. Additionally, all samples were measured for total gamma count using a Scintrex GIS-5. The sensitivity of Scintrex GIS-5 was tested on the concrete pads at the old Gencor camp site (Fig. 4). After samples were split through a riffle-splitter, RC drill chips were obtained by sieving the reject sample and then lithologically logged. The lithological logging included the description of grain-size, sorting, mineralogy and the definition of a lithological code. Finally, the reject samples were disposed in the old test pit area of Gencor's. The drill chips were stored at the sample container in the vicinity of the old Gencor camp site area. The logging data were captured in Microsoft Excel.

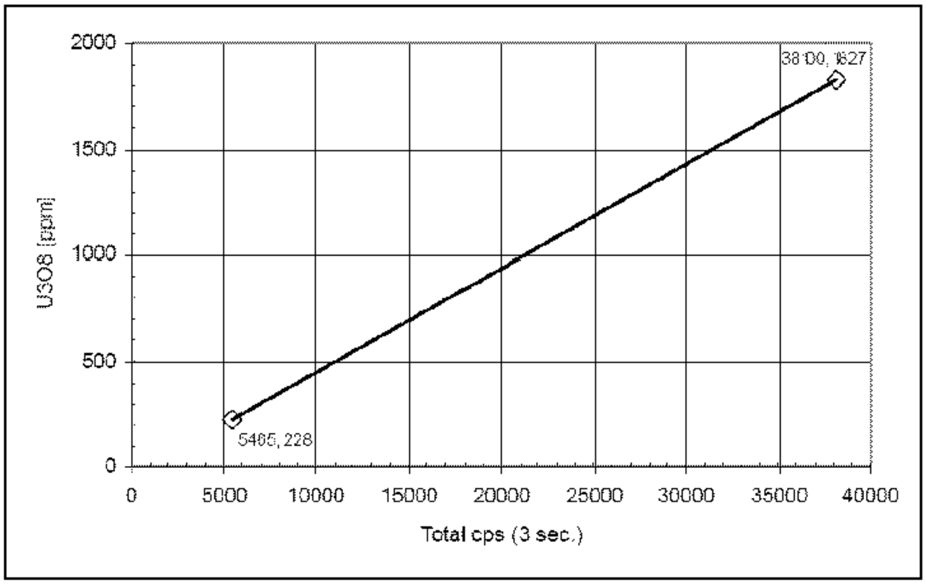


Figure 14: Sensitivity of the scintillometer used during the drilling programme

12.1.2 Downhole logging

Each hole was radiometrically logged through the RC rods after completion of drilling and prior to the rods being withdrawn from the hole. Some holes were logged in the open hole as well in order to confirm the calculation of a casing factor. Recalculation of casing factors in the course of the drilling programme indicated that the rods had a lower factor than the rods previously used in 2004. Finally, calculations by Doug Barrett suggest that a factor of 2.03 should be used instead of the 2.22 factor, which was used initially. Equivalent U_3O_8 values were calculated using the deconvolution spreadsheet provided by consulting geophysicist Doug Barrett.

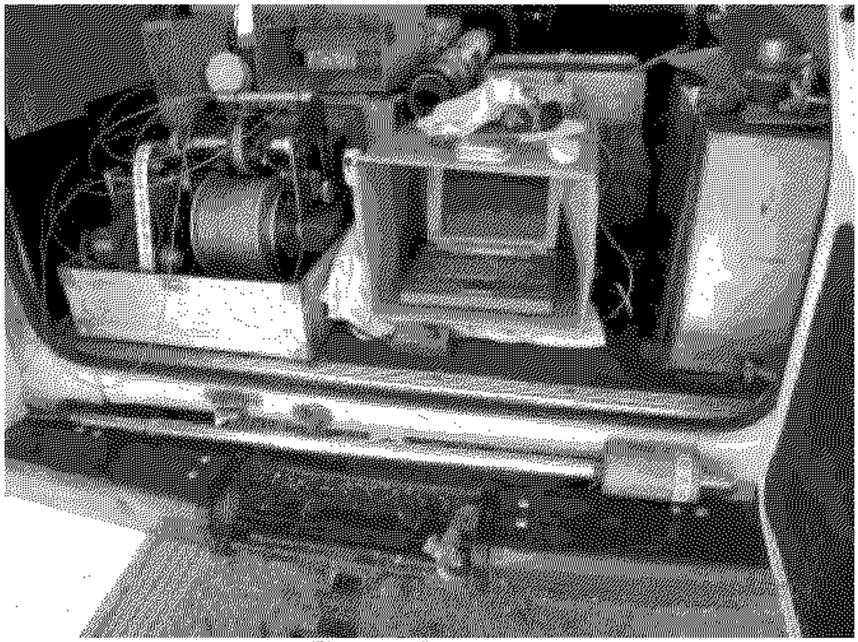


Figure 15: Logger setup

12.1.3 The purpose of sleeve calibrations

As a routine part of the downhole logging operations at LHU, a calibration sleeve supplied by Auslog is read on downhole probes every day before the commencement of logging. These readings are recorded and provide a quantitative check on the correct functioning of the logger. Any variation in the calibration reading outside of the expected statistical range indicates a possible malfunction in the probe or the logger electronics. The readings also provide a longer term quality control check on the condition of the probe between primary calibrations which are made at a calibration facility on an annual basis. For Probe A723, the calibration sleeve gives about 6100 cps. For this count rate, the expected statistical variation at the one standard deviation level would be the square root of 6100, which is 78 cps. Thus at the 95% confidence level (2 standard deviations) one would expect a random reading with the sleeve to lie within the range 6100 or - 156 cps. The variations for the readings in the attached graphs will be less than this since they are each averages of 4 readings.

12.1.4 Graphical representations of the data

The daily sleeve calibration readings, corrected for background, have been plotted in the attached graphs. The first graph shows the change in reading over the measurement period of some five months. It can be seen that the count rate is more or less constant up to the end of August and the probe appears to be functioning normally. After that there is a gradual decline. The consistency of this decline suggests that it is real and not part of random statistical fluctuations. This decline manifests itself in the skew to the left in the histogram, although two of the readings in August seem abnormally low and account for the outliers on the far left hand side of this diagram (*Figure 16*).

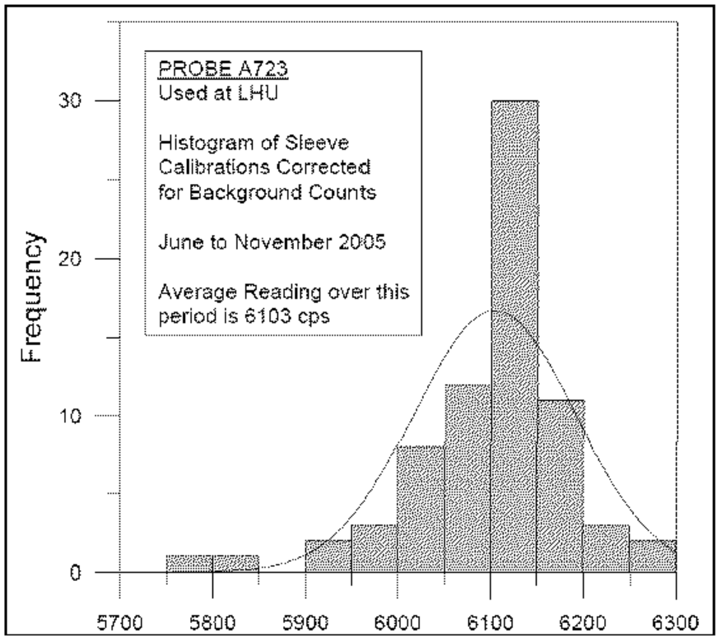


Figure 16: Calibration data of probe A723 for the period from June to November 2005

Such behaviour is normal for gross gamma probes whose sensitivity tends to decrease with time. An annual change of a few percent is not unusual. This probe was last calibrated in September 2004 and was showing signs of a need for re-calibration. It is probable that since the end of August 2005, equivalent uranium grades calculated from logs using A723 are a little low (up to 1.6% low) due to the probe's decreasing sensitivity. At the completion of 2005's drilling, the probe was sent to Geotron in South Africa for servicing and then to Pelindaba RSA for re-calibration. The calibration sleeve should also be read at Pelindaba at this time to provide a new base line reading for the sleeve.

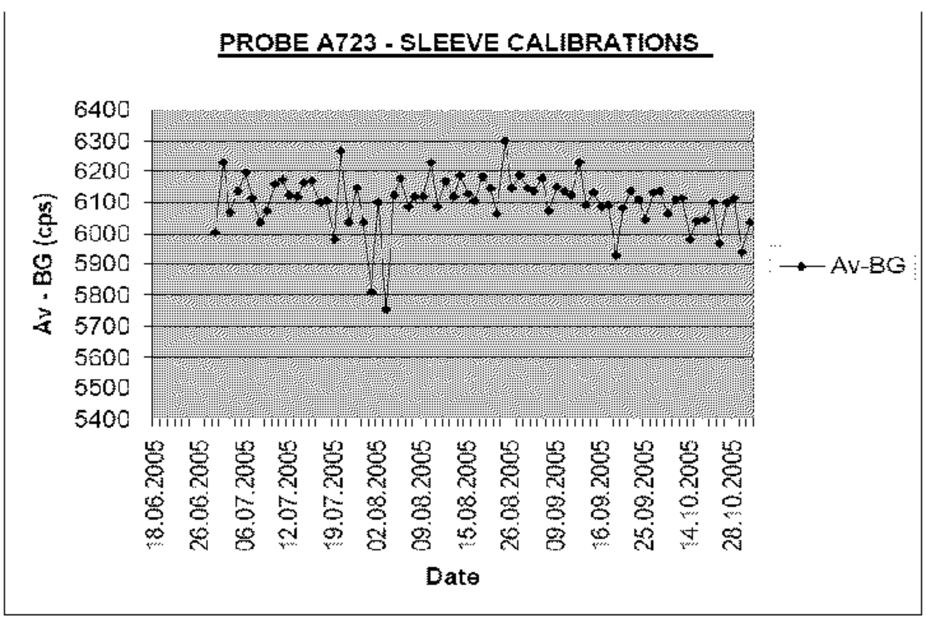


Figure 17: Calibration data of probe A723 for the period from June to November 2005

12.2 Sampling

12.2.1 XRF Assays

A sample was collected through a cyclone for each meter drilled and split through a riffle-splitter by the drill crew on the drill site to give a sample for assays of 3 - 4 kg. This sample was further split to produce two assay samples of about 300 to 500 g each, which were named as "A" and "B" samples. Reject sample was used for geological logging, and then was disposed of in the old test pit area of Gencor's.

Figure 18: The preparation of samples for assay

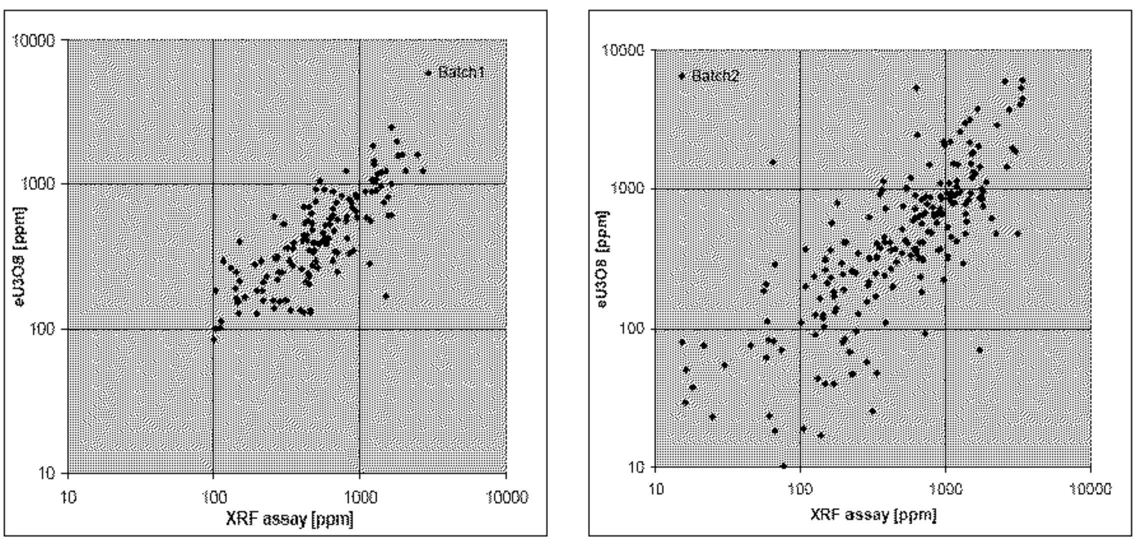
On the basis of the downhole logs, mineralised sections were selected from the boreholes in order to send them for XRF assay to Setpoint Laboratories, Isando, South Africa. The "A" samples were re-packed; and the new sample bags were tagged only with a serial number, one tag inside the bag and one outside the bag. Thus, borehole number and depth of the sample were not identifiable for the laboratory.

Six sample batches totalling 1143 samples (8.8 % of the total meters drilled), were sent for assay. The assay included the pulverisation of the rock chip sample and the assay for U_3O_8 by pressed powder XRF. The total number of samples included 9 % quality samples (103 blanks & duplicates). Blanks were inserted after every 20 normal samples and comprised dune sand collected south of Swakopmund consisting of well sorted quartz with minor iron oxide grain coatings. These dune sands assay below LLD (< 3 ppm U_3O_8). Duplicates were taken from every 20th normal samples. Duplicates were produced by taking "A" and "B" sample material together in order to get 33:33:33 split. The first split "A" was sent to the lab as a normal sample, the second split "B" was inserted at the end of each sample batch as a duplicate of "A". The last split "C" was kept as a reference sample and stored at the storage container.

A comparison between XRF assay and radiometric eU_3O_8 shows that they do not correlate particularly well (*Figure 19*). This problem was already noted during the previous year's drilling programme and discussed in the corresponding drilling report by John Rich. A short summary is given below:

1. The exact cylinder of rock which is ground up and removed for assay is the exact cylinder of rock which the down hole logger doesn't see. Thus, XRF assay and radiometric eU3O8 represent two completely separate samples of material.
2. The RA logger and deconvolution process detects and uses gamma radiation emanating anywhere from an approximate 40cm radius. The geochemical assays can be considered to be "absolute", in other words they define exactly how much U is present in a sample.
3. The mathematics upon which the deconvolution process is based (inaccuracies exist in applying a water factor, calculation for casing factor).
4. Contamination almost certainly occurs due to drilling methodologies, e.g sample contamination in the cyclone, changing of sample bags, cleaning hoses etc.
5. Inaccuracies in defining down hole depth. The RA operator took great care to position the probe so that hole "zero" equalled natural ground level, however the accuracy is probably no better than ± 10cm.

To sum up, the potential difficulties outlined below suggest that on a meter by meter/sample by sample basis, correlation will always be poor, but as sample size increases and errors cancel, average eU_3O_8 will approach chemical U_3O_8.



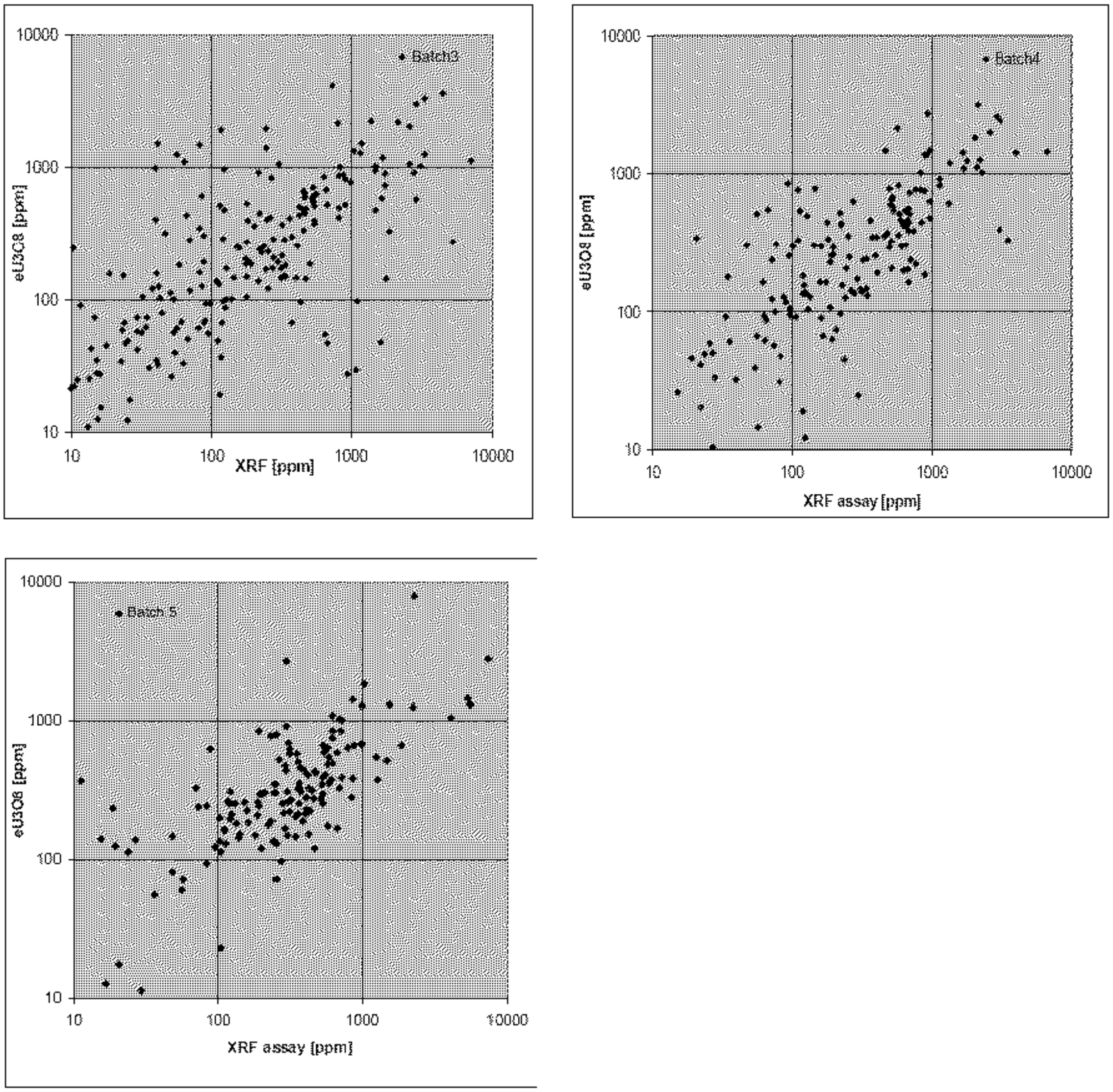


Figure 19: Correlation between XRF assay and radiometric eU3O8 for each sample batch.

12.3 Paladin 2006 Infill drilling

12.3.1 Lithological logging

Drill chip samples for each meter drilled were geologically logged. At the drill site, drill chip samples were checked for their colour, moisture, weight, and HCl reaction. Additionally, samples were measured for total gamma count using spectrometers or scintillometers. After samples were split through a riffle-splitter, RC drill chips were obtained by sieving the reject sample and then lithologically logged. The lithological logging included the description of grain size, sorting, mineralogy and the definition of a lithological code. The drill chips are stored in chip trays at the exploration storage facilities in Detail 3. The logging data were captured using Micromine software.

12.3.2 Downhole logging

Each borehole was radiometrically logged through the RC rods after completion of drilling and prior to the rods being withdrawn from the hole. Some holes were logged open (after the rods were withdrawn) in order to confirm the calculation of the casing factor.

12.3.3 Calibration

As a routine part pf the downhole logging operations at LHU, a calibration sleeve supplied by Auslog is read on the downhole probe every day before commencement of logging. These readings are recorded and provide a quantitative check on the correct functioning of the logger. Any variation in the calibration reading outside of the expected statistical range indicates a possible malfunction in the probe or the logger electronics. The readings also provide a longer term quality control check on the condition of the probe between primary calibrations, which are made at a calibration facility on an annual basis (Barrett 2005).

For the probe A723, which was used during last year's and this year's drilling, the calibration sleeve gave an average reading of 6103 (Barrett 2005) and 6074, respectively. For both count rates, the expected statistical variation at the one standard deviation level would be the square root of 6100 and 6074, respectively, which is 78 cps. Thus, at the 95 % confidence level (2 standard deviations), a random reading with the sleeve should lie within the range of 6074 ± 156 cps.

12.3.4 Graphical Representation of the Data

The daily sleeve calibration readings, corrected for the background, show the change in count rates over the measurement period of about 6 months (*Figure 20*). In general, the count rate was more or less constant over this period, which includes the logging work for the grade control drilling campaign (March – June 2006) as well as the exploration drilling programme (July – September 2006). However, *Figure 20* shows that four of the readings are abnormally high and account for the outliers on the far right hand side in *Figure 21*. These abnormal readings could indicate a possible malfunction After the completion of the drilling programme the probe was sent to Geotron, Potchefstroom, South Africa for repair and then to Pelindaba for re-calibration.

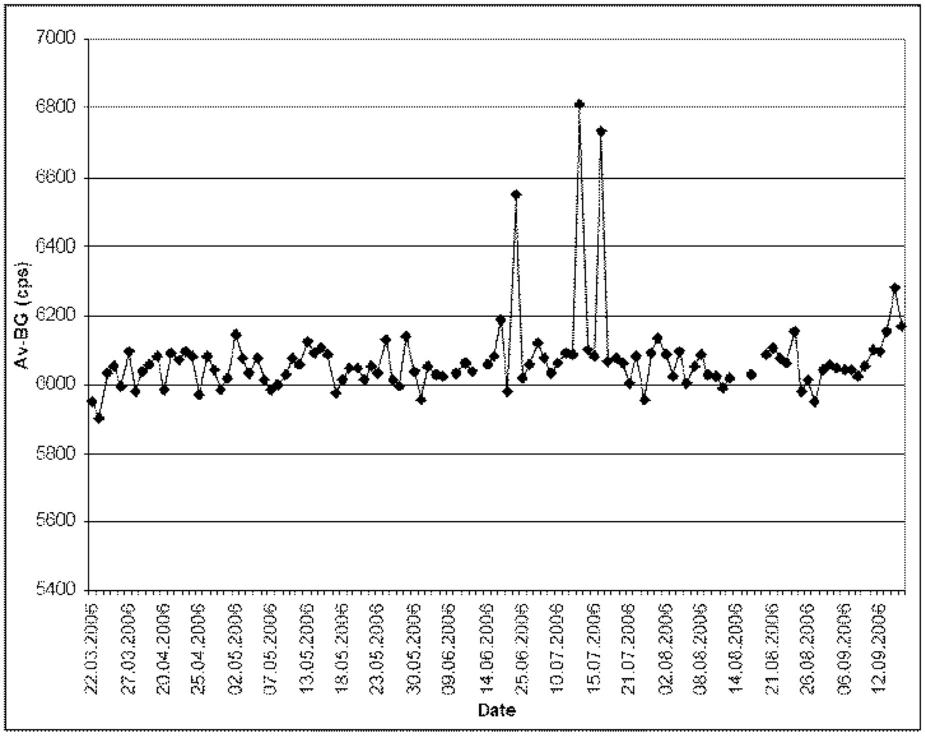


Figure 20: Count rates probe A723.

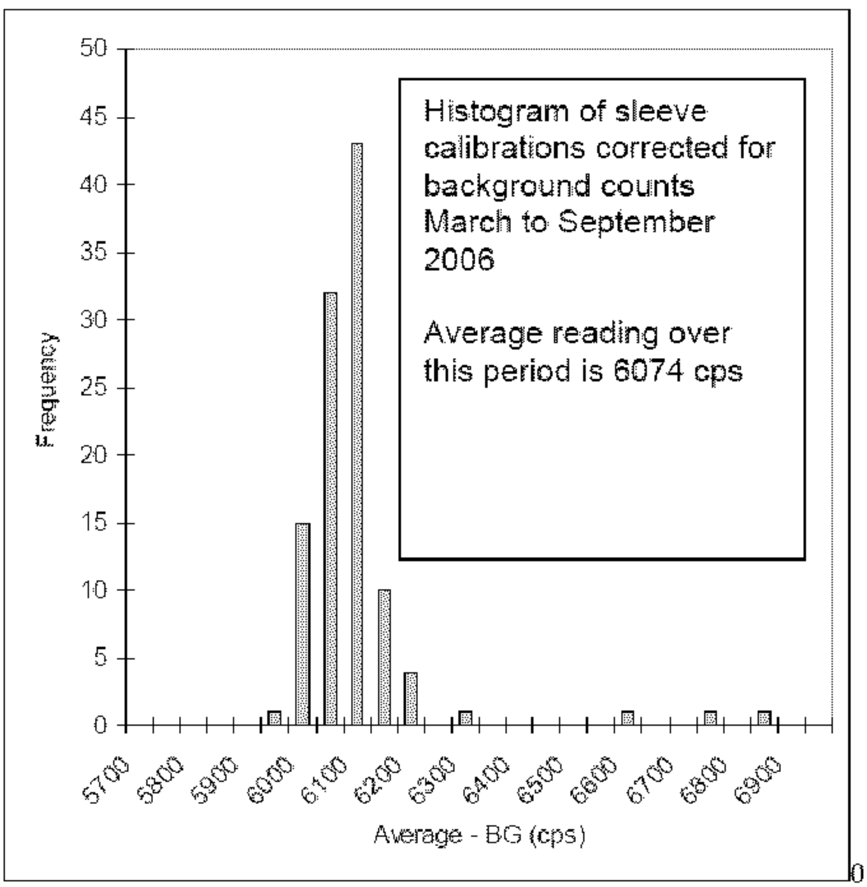


Figure 21: Histogram of sleeve readings probe A723.

12.4 Sampling

12.4.1 XRF Data

In general, the sampling followed the same sampling procedure that was applied during the previous drilling campaign in 2005 (Kärner 2005). Samples were collected through a cyclone for each meter drilled and split through a riffle-splitter by the drill crew on the drill site to give a sample for assay of 3 – 6 kg. This sample was further split to produce two assay samples of about 300 to 500 g each, which were named as "A" and "B" samples. On the basis of the eU3O8 values obtained from the downhole logging, one borehole from each drill line was selected, and its mineralized sections as well as samples from below and above were sampled. These samples were sent for XRF assay to Setpoint Laboratories, Isando, South Africa. The "A" samples were re-packed; and the new sample bags were tagged only with a serial number, one tag inside and another one outside the bag. Thus, borehole number and depth of the sample were not identifiable for the laboratory. Two sample batches totaling 375 samples (5.9 % of the total meters drilled), were sent for assay. The assay included the pulverization of the rock chip sample and the assay for U3O8 by pressed powder XRF.

In order to ensure high-quality results, sets of quality control samples were inserted after each 20th routine sample, including a reference material, a blank, and a duplicate of every 20th routine sample.

The reference sample is the uranium ore standard CUP-1 produced by Canada Centre for Mineral and Energy Technology. The blanks comprise dune sand collected south of Swakopmund consisting of well sorted quartz grains with minor iron oxide coatings. These dune sands assay below the lower detection limit, which is 7 ppm U3O8 according to Setpoint Laboratories. Duplicates were produced by splitting the A sample into a 50:50 split. As noticed during the previous drilling and sampling campaigns, the correlation between the XRF assay and the corresponding radiometric eU3O8 value is reasonable between 100 and 1000 ppm. Outside this range the correlation is rather poor (*Figure 22*). As a result of statistical analysis of the results of the assay process further batches of samples were submitted to Setpoint to allow for more complete mineralised intervals to be used in the comparison between XRF and radiometrically logged intervals, the same was also done with samples from the 2005 drilling campaign.

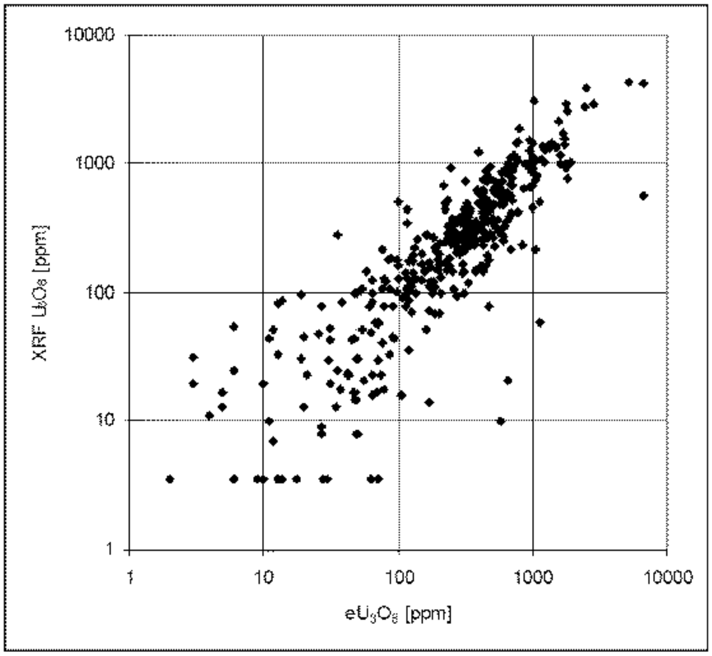


Figure 22: Comparison between XRF and radiometric data.

12.4.2 Disequalibrium Studay

6 composite samples in total were taken along Detail 3, 5 and 6. They were first sent to Setpoint in South Africa for homogenization and XRF assay and then forwarded to ANSTO in Australia for radioanalysis. The results are still outstanding.

13 Sample Preparation, Analysis and Security

13.1 Assay Accuracy

13.1.1 Chemical assays

There is no information available to directly confirm the accuracy of XRF assays of Gencor samples from drill holes and test shafts. The accuracy of those assays can only be inferred by comparison to later Acclaim and LHU drill sampling (Section 13.3.2, below).

Acclaim submitted blanks and certified reference materials with the RC drill samples sent for analysis by SGS Mineral Services. Standards were sourced from Industrial Analytical (Pty) Ltd, Groenkloof, RSA. Twenty-seven blank samples all returned U_3O_8 grades less than detection limit (3ppm). Figure 23 to Figure 30 show run charts for assays of standards. H&S considers the accuracy of SGS's assays satisfactory. Assays for standards 6, 9 and 11 tend to report low across all batches but the errors are not considered significant in terms of resource estimation risk.

Whilst in 2004 Paladin did not submit any standards with the assay batches ALS laboratory included a number of certified reference materials within the analysis batches. These standards were sourced from Canada Centre for Mineral and Energy Technology (Canmet), National Research Centre for Certified Materials, China. One Hundred and thirty four blank samples all returned U_3O_8 grades less than detection limit (3ppm). Figure 31 to Figure 33 show run charts for assays of standards. H&S considers the accuracy of ALS's assays satisfactory; however the results from the BL-1 and NBL42-4 standards show a consistent under reporting of approximately 2% for batches 1-16, these errors are not considered significant in terms of resource estimation risk.

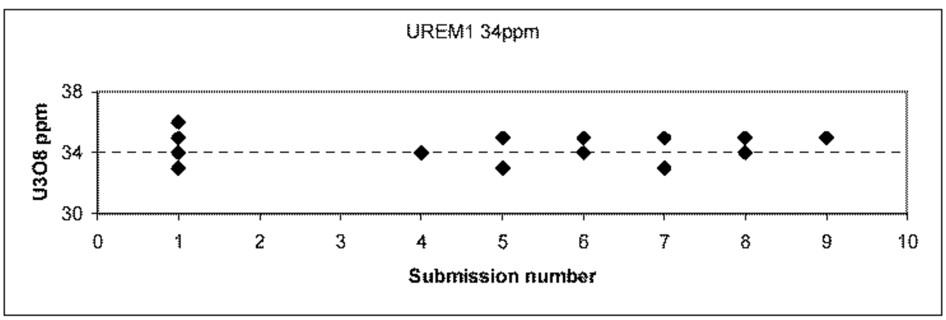


Figure 23: Assays of reference standard UREM1 submitted by Acclaim

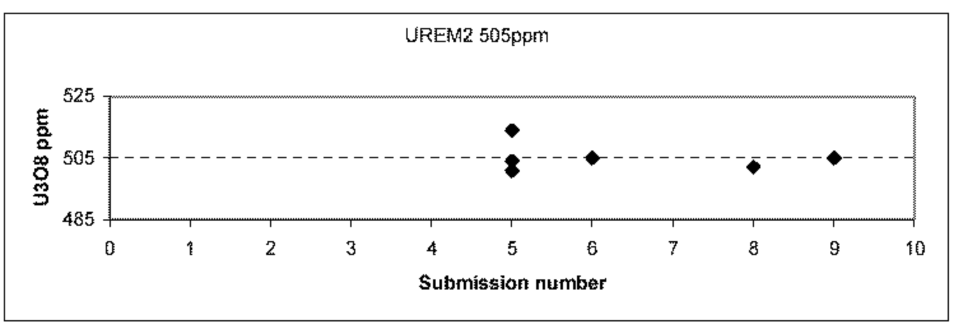


Figure 24: Assays of reference standard UREM2 submitted by Acclaim

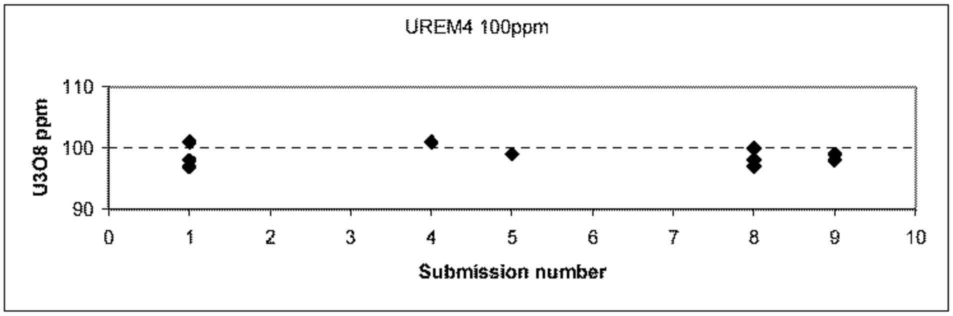


Figure 25: Assays of reference standard UREM4 submitted by Acclaim

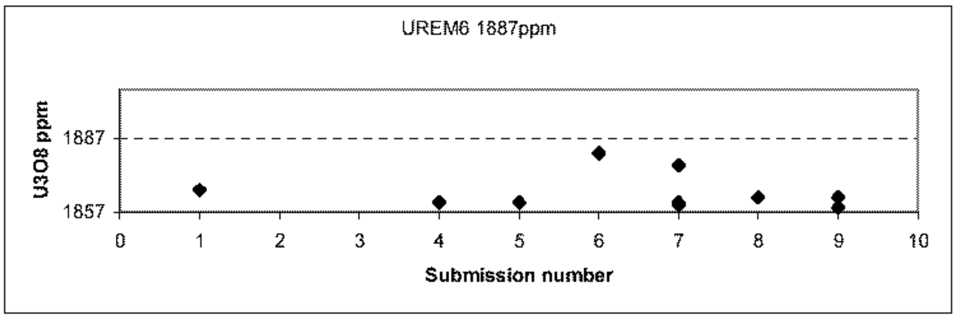


Figure 26: Assays of reference standard UREM6 submitted by Acclaim

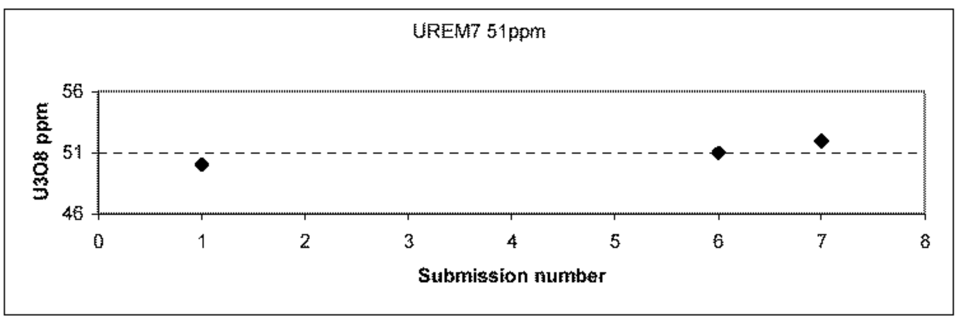


Figure 27: Assays of reference standard UREM7 submitted by Acclaim

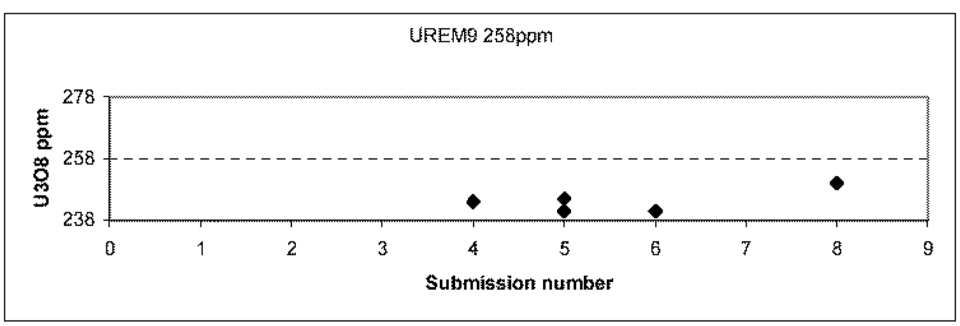


Figure 28: Assays of reference standard UREM9 submitted by Acclaim

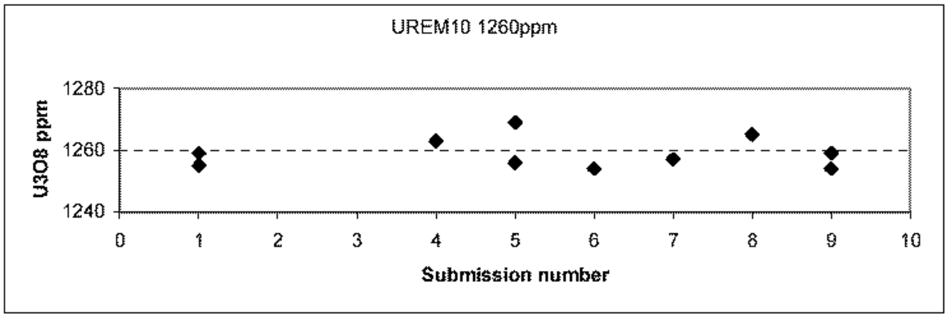


Figure 29: Assays of reference standard UREM10 submitted by Acclaim

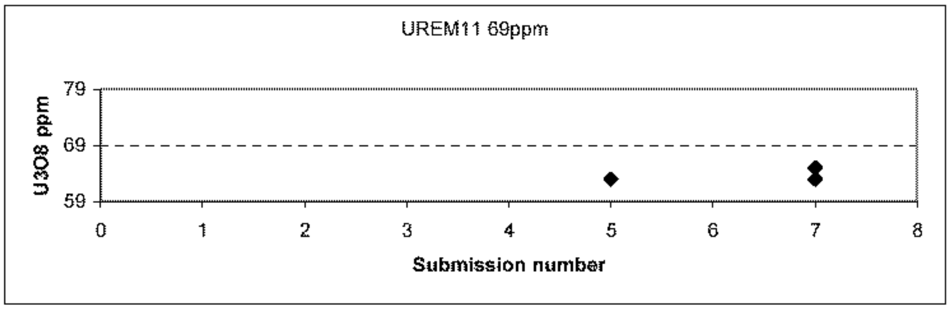


Figure 30: Assays of reference standard UREM11 submitted by Acclaim

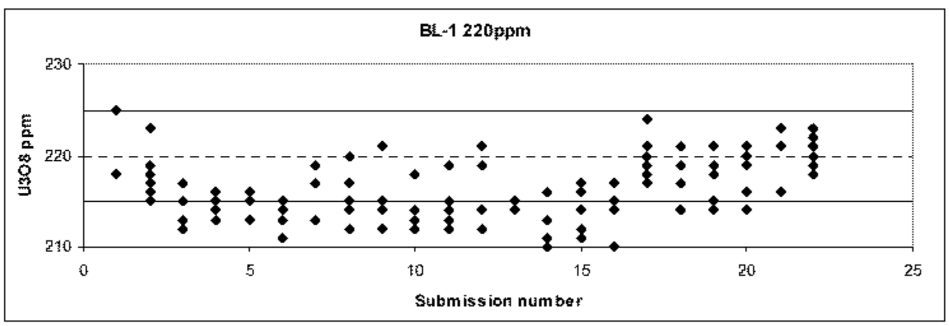


Figure 31: Assays of reference standard BL-1 ALS laboratory

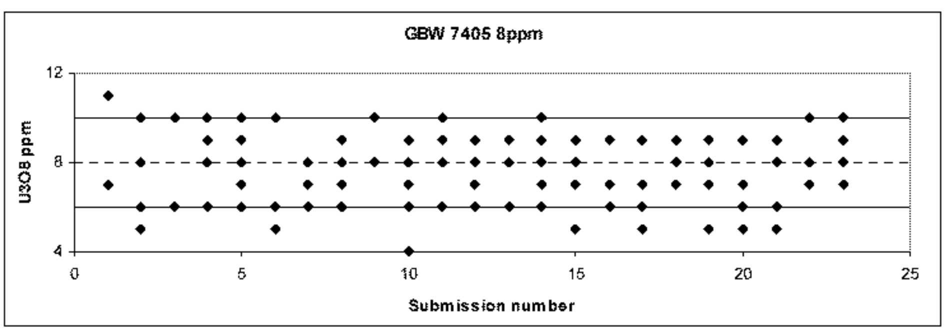


Figure 32: Assays of reference standard GBW 7405 ALS laboratory

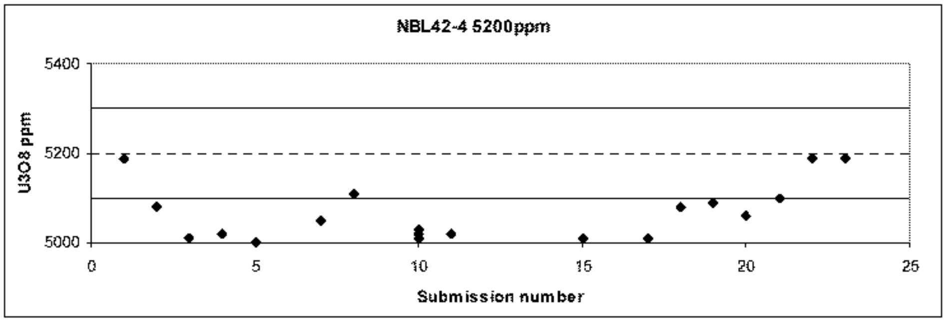


Figure 33: Assays of reference standard NBL42-4 ALS laboratory

During the 2005 and 2006 drilling campaigns international standards were routinely submitted with assay batches. The results from these and the laboratory internal standards are detailed below. In most cases the standards returned values that were 1 to 2 % low, except one batch of UREM10 with was 2.5% high and the very low level NIM G which appeared to suffer from a 3-5ppm background offset. Figure 38 shows the performance of certified standard CUP-1 submitted by Paladin during the 2006 drilling program, this standard appears to be overestimated by approximately 4% although the precision of the analysis appears to be reasonable. It should be noted that SetPoint Laboratories recommend a repeat analysis by fused disc for values above 1200ppm

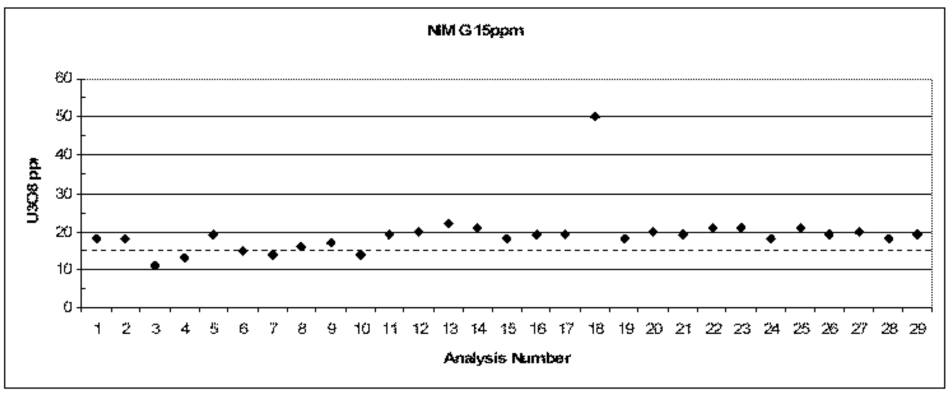


Figure 34: Assays of reference standard NIM G

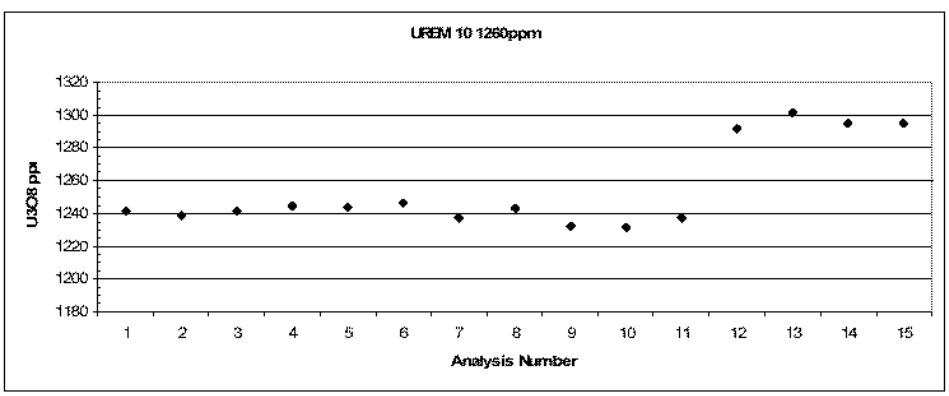


Figure 35: Assays of reference standard UREM10

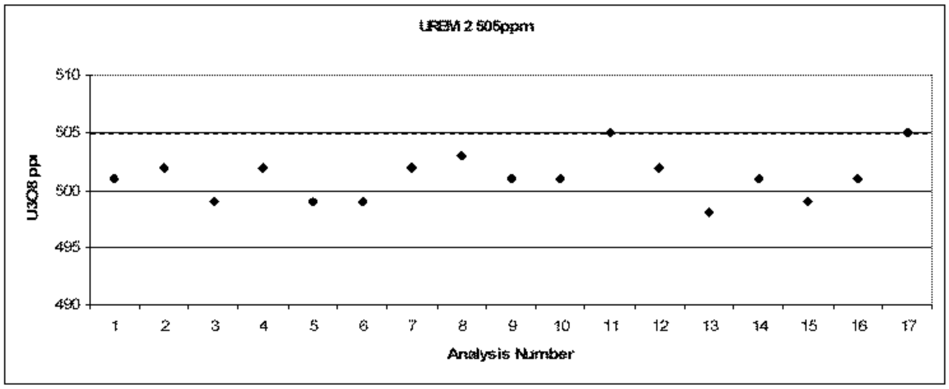


Figure 36: Assays of reference standard UREM2

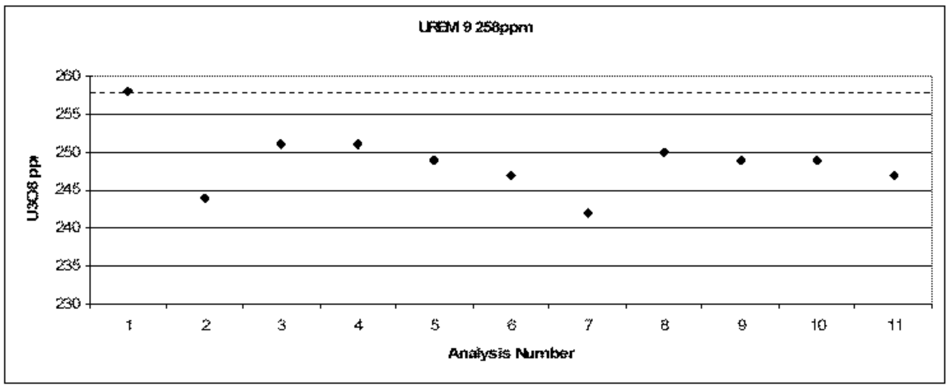


Figure 37: Assays of reference standard UREM9

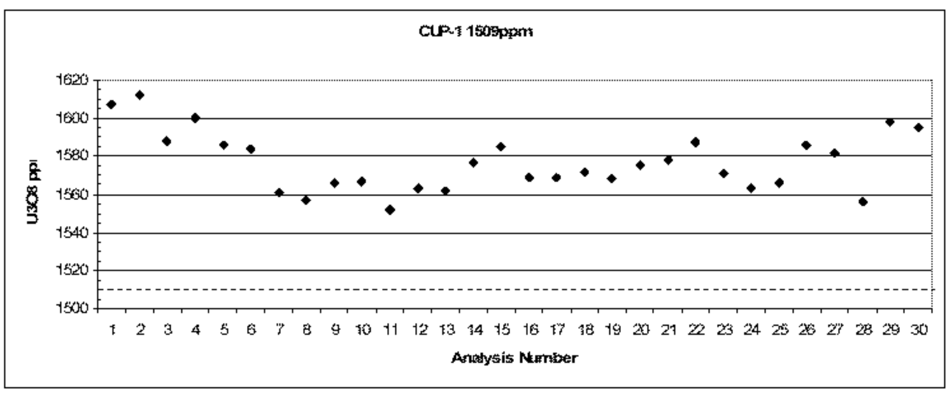


Figure 38: Assays of reference standard CUP-1 Submitted by Paladin

13.1.2 Accuracy of Grades from Radiometric Logging

Acclaim reportedly calibrated scintillometers in a test-pit prior to down-hole logging of RC drill holes. Determination of U_3O_8 grades from scintillometer counts requires a number of corrections including:

- Deconvolution. This is essentially a correction for the difference between the test-pit volume (or linear interval) of mineralisation used to calibrate the instrument and the volume (or linear intervals) of mineralisation comprising the source of down-hole counts.

- Disequilbrium. This correction accounts for the proportion of gamma rays that derive from decay of uranium versus the proportion that derive from decay of daughter products such as thorium, potassium and radon. The correction is most important in deposits in which oxidation state varies. This is almost certainly not the case at Langer Heinrich.

It should be noted that the U_3O_8 grades derived from both Acclaim's and Paladin's down-hole radiometric logs take the above corrections into account. Considering the correlation of radiometric and XRF grades from both Acclaim and Paladins's work (Section 13.4.1, below) it appears likely that any change required to the disequilibrium corrections previously applied will have very little impact on resource estimates.

13.2 Sampling Precision

13.2.1 Gencor Percussion Drill Samples

There are a large number of sample intervals from Gencor percussion drill holes for which XRF analyses are available for multiple sample splits but in many cases one of the samplings represents samples composited over several metres prior to submission for assay. Instances where this was obviously the case were deleted for the sake of this comparison, leaving 142 sample pairs. Comparison of these assays captures the sampling and analytical error arising from the entire chain of sampling and analytical processes.

Figure 39 shows a scatter plot comparing assays of original sample splits and re-splits in single-metre sample intervals. There is reasonable correlation between sample grades. The quantile-quantile plot in Figure 40 compares the marginal histograms of the two sample populations and indicates that there is not significant bias to higher grades in either of the sample splits.

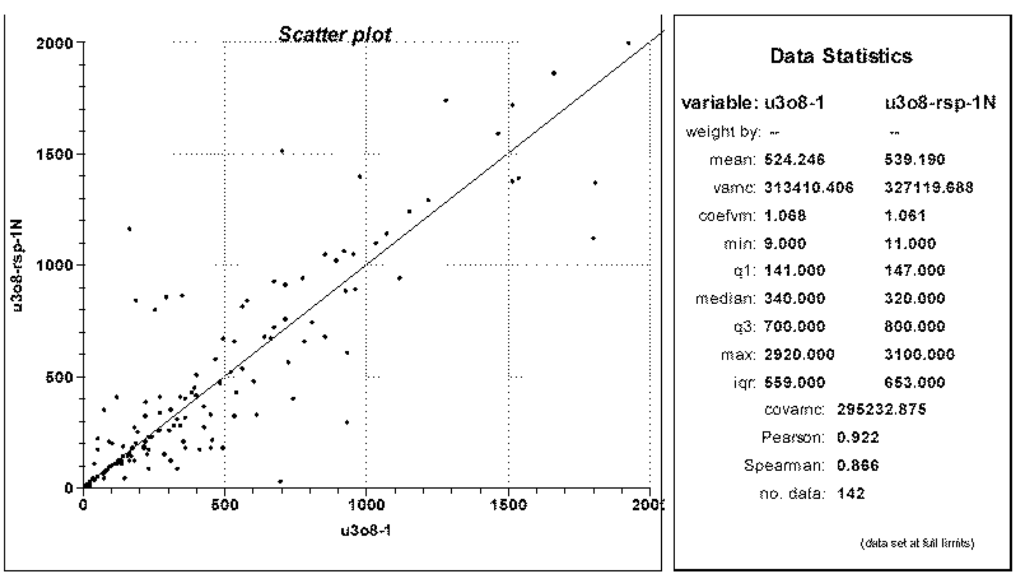


Figure 39: U_3O_8 grades in re-splits of Gencor percussion drill samples

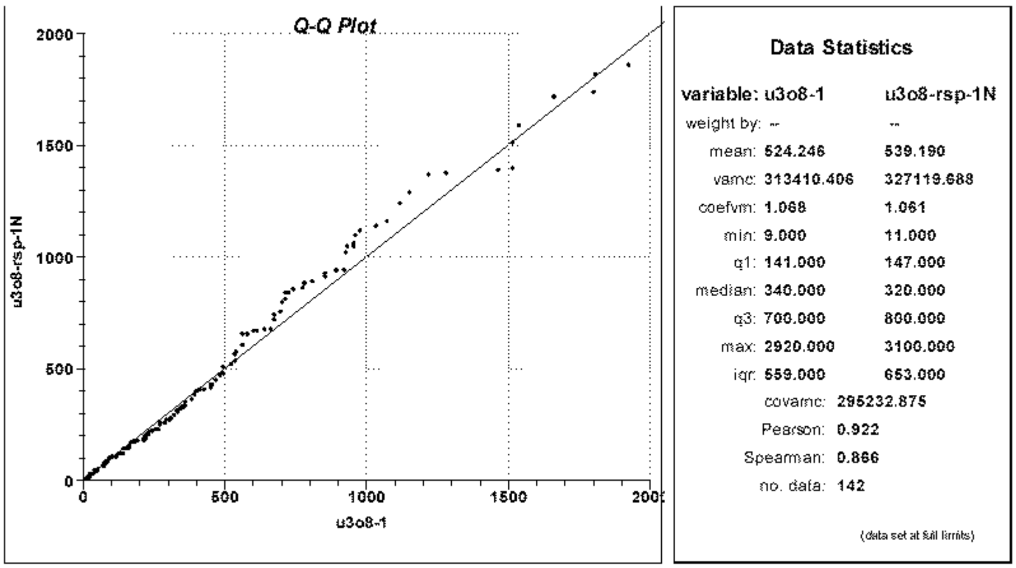


Figure 40: U_3O_8 grades in re-splits of Gencor percussion drill samples

13.2.2 Acclaim RC Drill Samples

There are 103 native sample intervals from Acclaim RC drill holes with XRF analyses for field re-splits of RC samples. Figure 41 compares assays of first and second sample splits. There is excellent agreement between sample pairs and no significant differences between the two sample populations.

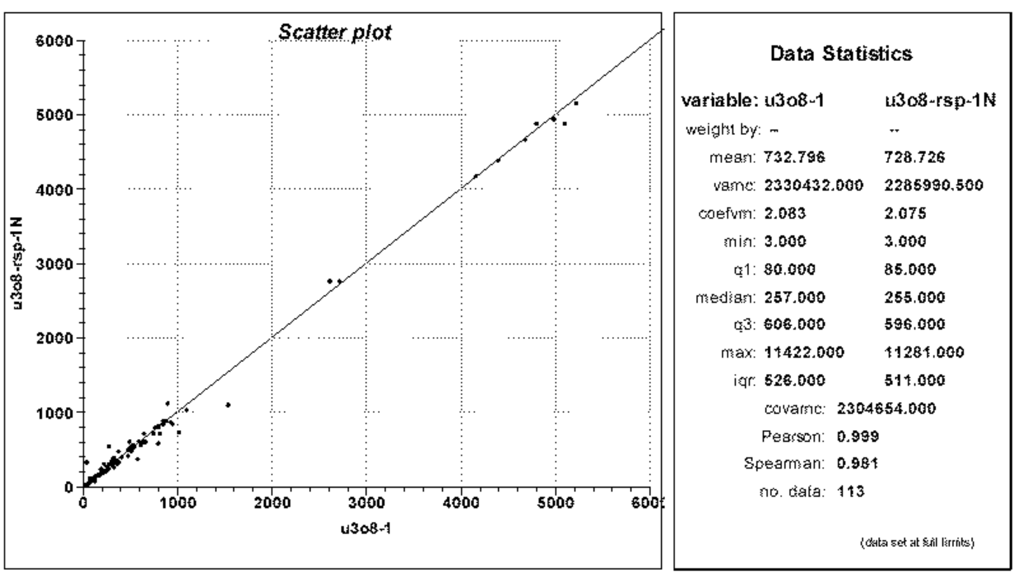


Figure 41: U_3O_8 grades in re-splits of Acclaim RC drill samples

13.2.3 Paladin RC Drill Samples

There are 92 native sample intervals from Paladin 2005 and 2006 RC drill holes with XRF analyses for field re-splits of RC samples. Figure 42 compares assays of first and second sample splits. There is good agreement between sample pairs and no significant differences between the two sample populations.

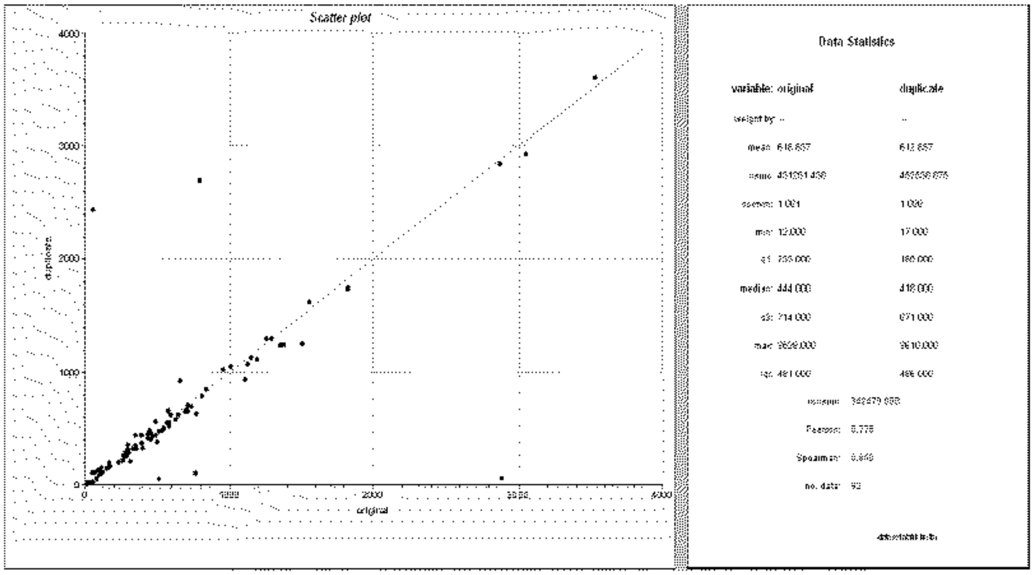


Figure 42: U_3O_8 grades in re-splits of Paladin RC drill samples

13.3 Assay Precision

13.3.1 Gencor Percussion Drill Samples

There are 166 native sample intervals from Gencor percussion drill holes with repeat XRF analyses available (Figure 43). Comparison of these assays captures the sampling and analytical error arising from laboratory sub-sampling and analytical processes. There is reasonable correlation between first and second analyses. The usefulness of the comparison is reduced by the fact that, in some instances, initial analyses relate to composited sample intervals and repeat analyses to individual metre intervals. In such instances the comparison is capturing sampling errors in addition to analytical errors. There are no significant differences between the two sample populations (Figure 44).

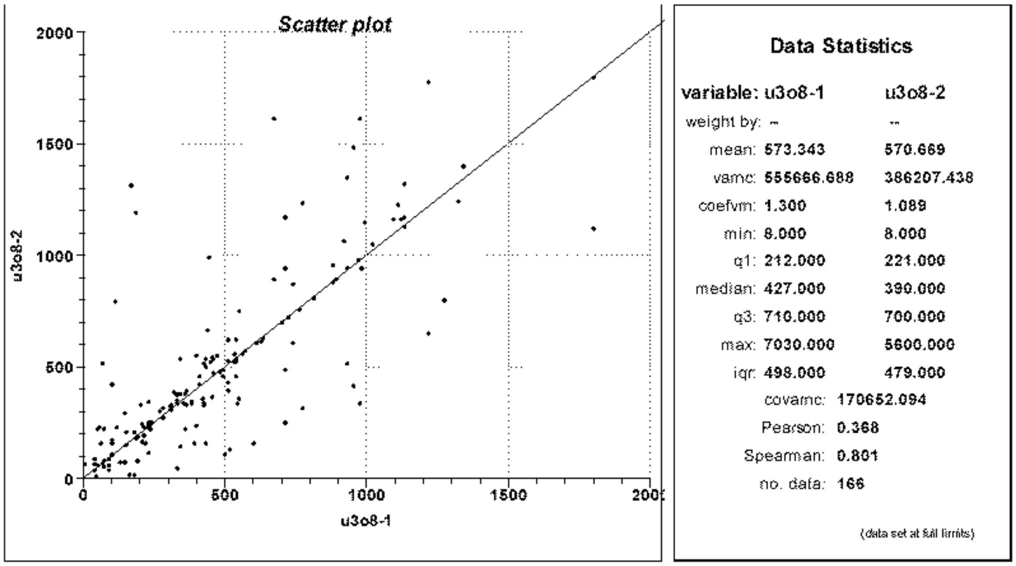


Figure 43: U_3O_8 grades in repeat analyses of Gencor percussion drill samples

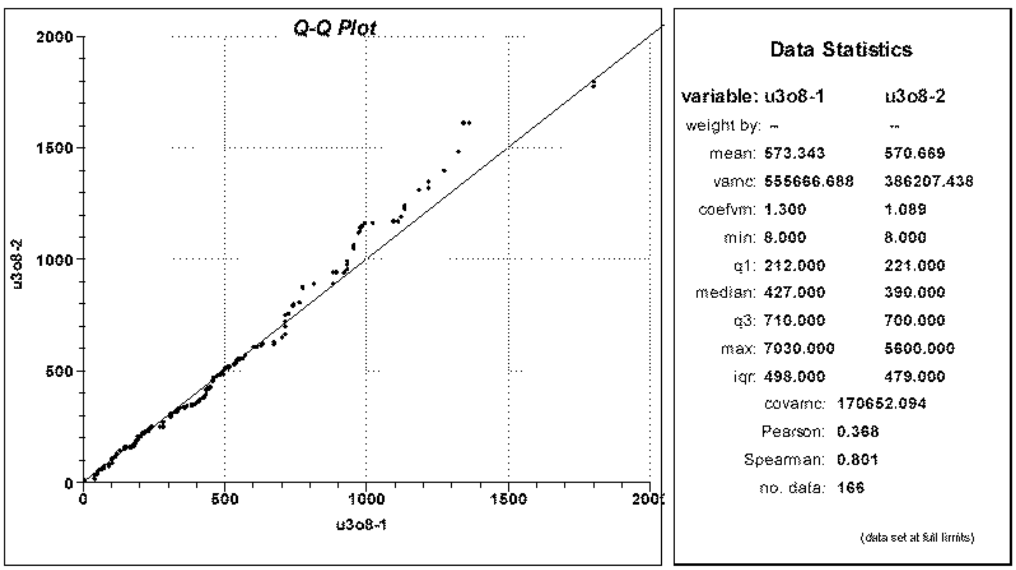


Figure 44: U_3O_8 grades in repeat analyses of Gencor percussion drill samples

13.3.2 Acclaim RC Drill Samples

There are 203 sample intervals in Acclaim RC drill holes with repeat XRF assays. Figure 45 compares U_3O_8 grades in duplicate analyses. Assay precision is very high.

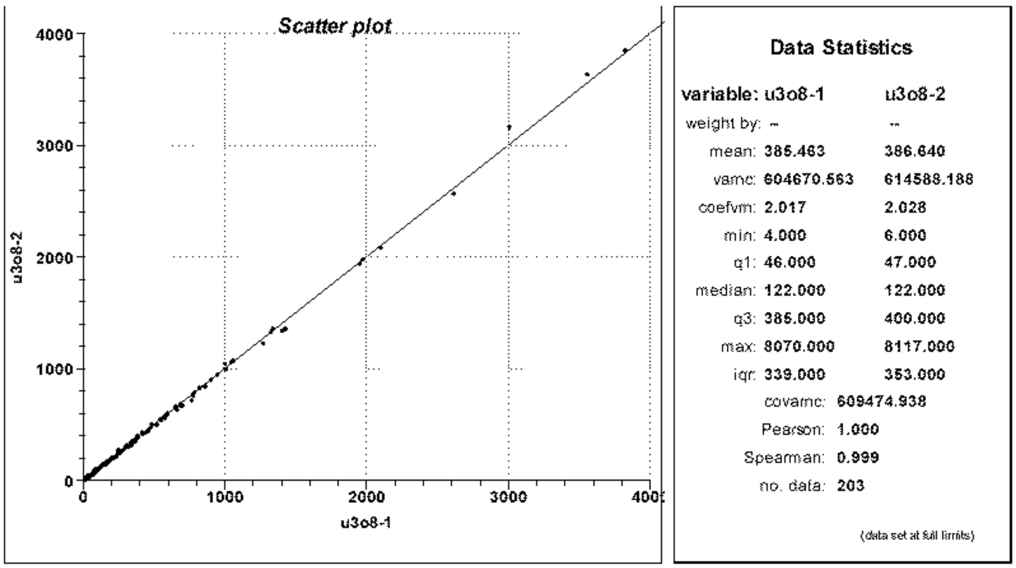


Figure 45: U_3O_8 grades in repeat analyses of Acclaim RC drill samples

13.3.3 Paladin RC Drill Samples

There are 92 sample intervals in the Paladin 2005 and 2006 RC drill holes with repeat XRF assays. Figure 46 compares U_3O_8 grades in duplicate analyses. Assay precision is very high.

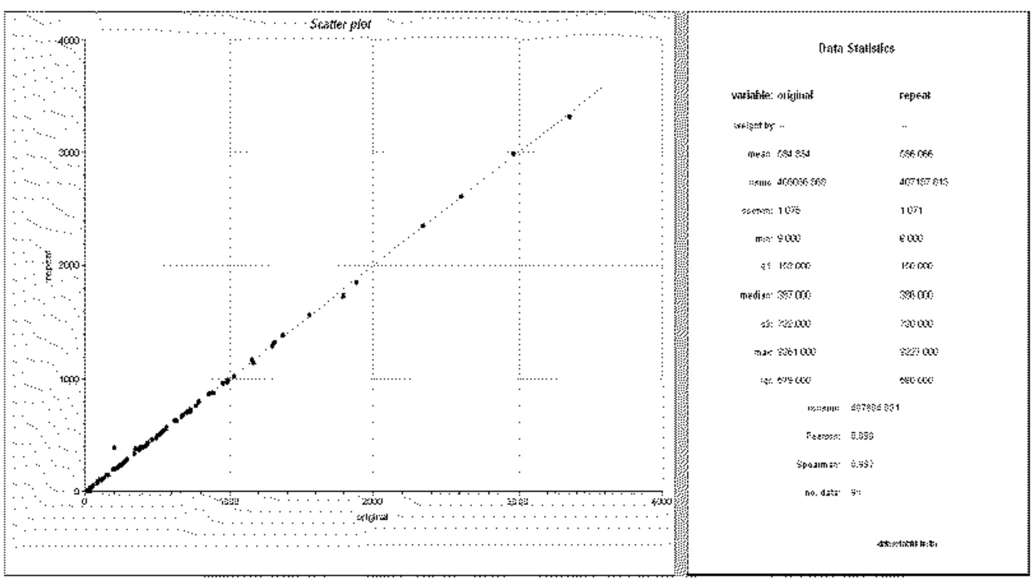


Figure 46: U_3O_8 grades in repeat analyses of Paladin RC drill samples

13.4 Compatibility of XRF Assays and Radiometric Data

13.4.1 XRF Assays and Radiometric Grades in Acclaim RC Drill Holes

There are 1727 native sample intervals from Acclaim RC drill holes with U_3O_8 determinations by both XRF analysis (above detection limit) and down-hole radiometric logging (Figure 47). Grades of individual sample intervals vary considerably, as might be expected: the XRF analyses represent the grade of material extracted from holes whereas the radiometric logs represent the grade of material comprising the walls of drill holes. Figure 48 compares the marginal histograms of grades arising from the two samplings. In high-grade mineralisation there is a tendency for radiometric logging to report higher U_3O_8 grades but differences between the two grade populations are otherwise negligible. The differences at high U_3O_8 grades may reflect a tendency, as indicated by assays of reference standards, for XRF assays to report slightly low.

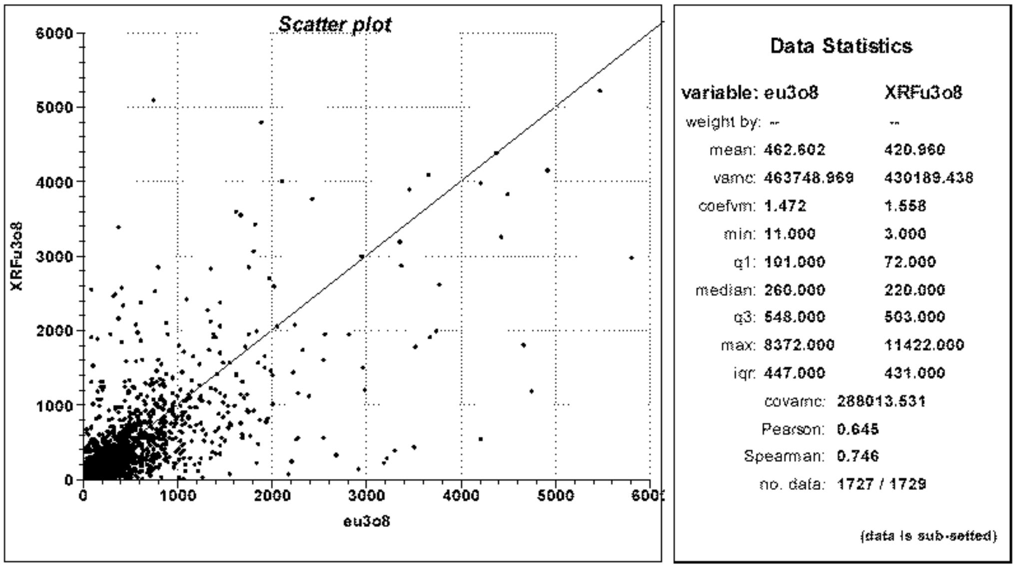


Figure 47: U_3O_8 grades by XRF and by down-hole logging, Acclaim RC drill holes

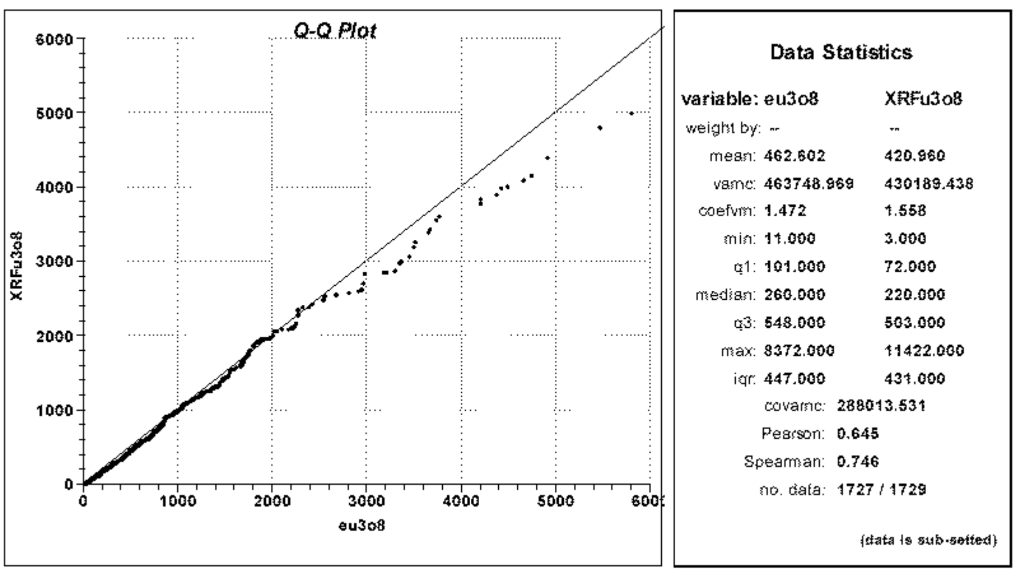


Figure 48: U_3O_8 grades by XRF and by down-hole logging, Acclaim RC drill holes

13.4.2 XRF Assays and Radiometric Grades in Paladin RC Drill Holes

There are 318 Paladin RC drill holes with U_3O_8 determinations by both XRF analysis and down-hole radiometric logging (Figure 49). Grades of individual sample intervals vary considerably, as might be expected: the XRF analyses represent the grade of material extracted from holes whereas the radiometric logs represent the grade of material comprising the walls of drill holes, as a consequence of this individual samples intervals have been composited to complete drill holes to minimise this local variability. Figure 50 compares the marginal histograms of grades arising from the two samplings. In low-grade mineralisation there is a tendency for radiometric logging to report higher U_3O_8 grades. The differences at high U_3O_8 grades may reflect a tendency for XRF assays to report slightly low.

Barrett Geophysical were engaged by Paladin to assess the relationship between the XRF values and those obtained by down hole radiometric logging. It was their conclusion that the radiometric values contain an offset of 35ppm (most likely as a result of an elevated background) and appear, following regression analysis, to overstate the U_3O_8 values by 5%. This effect may be due to insufficient disequilibrium calibration factors being applied to the original radiometric values.

As a consequence all Paladin radiometric U_3O_8 values, after application of appropriate casing and water factors, used in the resource estimation were factored to 96% of their resultant value. This in effect provides an allowance for formation and disequilibrium factors.

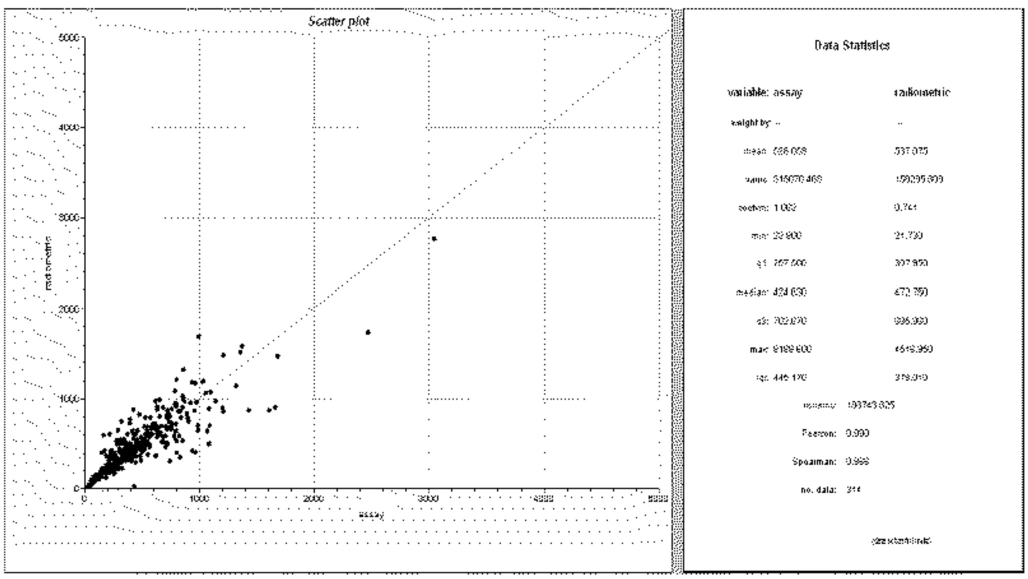


Figure 49: U_3O_8 grades by XRF and by down-hole logging, Paladin RC drill holes

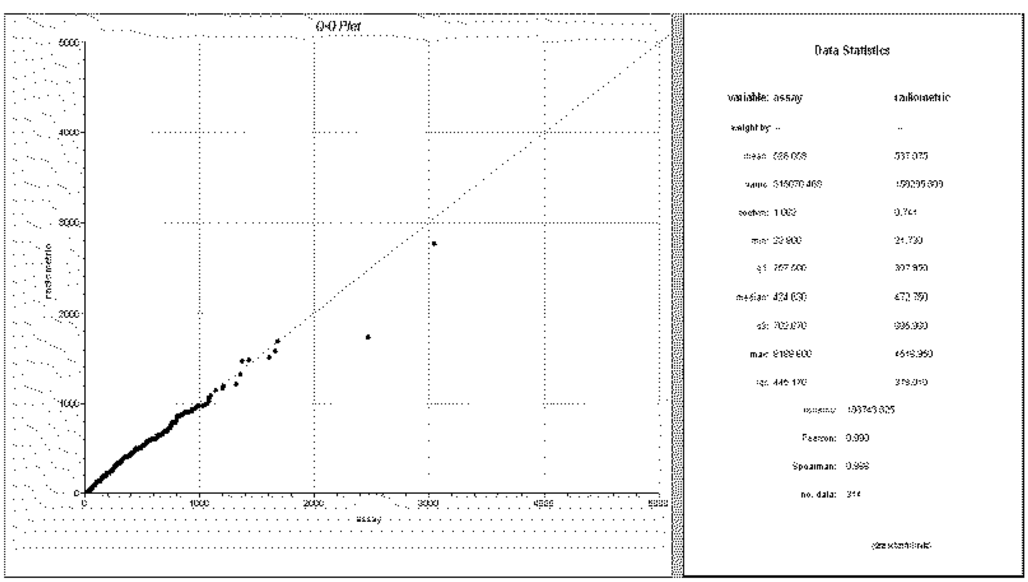


Figure 50: U_3O_8 grades by XRF and by down-hole logging, Paladin RC drill holes

13.4.3 XRF Assays and Radiometric Grades in Gencor Percussion Drill Holes

There are only sixteen sample intervals available with both chemical analyses and radiometric U_3O_8 grades.

13.4.4 XRF Assays on Percussion Drill Samples and Radiometric Grades in Nearby RC Drill Holes

Samples from Gencor percussion drill holes in Detail 1 assayed by XRF were compared with U_3O_8 grades from radiometric logging of nearby Acclaim RC drill holes. The comparison was restricted to:

- Gencor percussion drill sample composites from holes other than close-spaced holes in the trial mine area, for which XRF assays are available, excluding samples from below the water table and those in basement rock

- Radiometric U_3O_8 grades for one-metre down-hole intervals above the water table in Acclaim drill holes.

A search was conducted to find Acclaim samples lying within radii of 25mE x 25mN x 0.49mRL of each Gencor sample. The search located 527 nearest neighbour pairs separated by, on average, 22 metres that can be regarded as two independent samplings of approximately the same volume of mineralisation. Figure 51 shows a quantile-quantile plot comparing the two sample grade populations. In contrast to the comparison of XRF assays and radiometric logging in Acclaim RC drill holes, there is a tendency for Gencor's XRF analyses of percussion drill samples in higher-grade mineralisation to return higher U_3O_8 grades than radiometric logs of nearby Acclaim RC drill holes. This may indicate a tendency for a bias to higher grades in percussion drill samples.

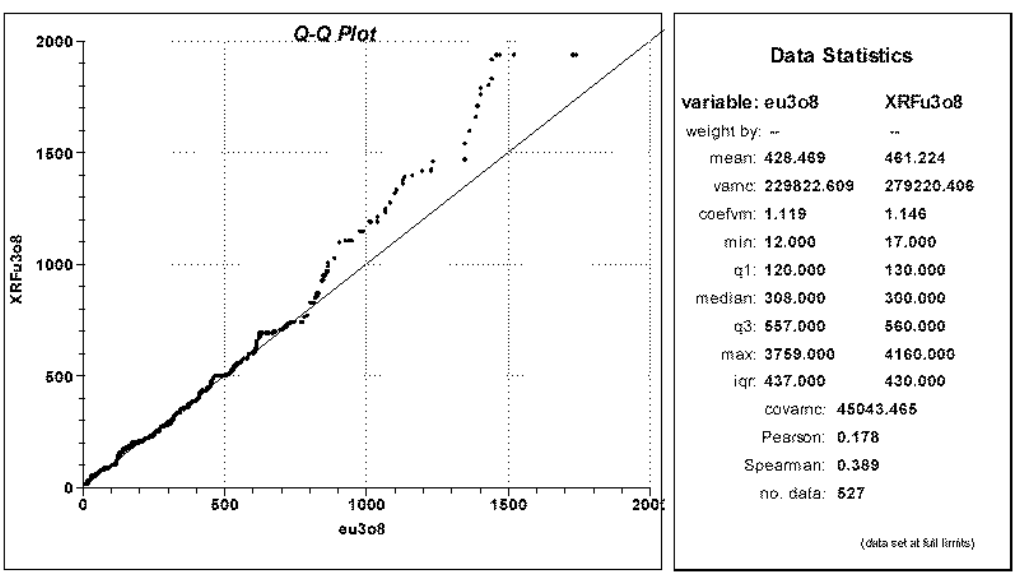


Figure 51: Gencor XRF assays versus nearest-neighbour Acclaim radiometric grades

13.5 Reliability of Gencor Dry Percussion Drill Samples

13.5.1 Gencor Percussion Drill Samples and Diamond Core Twins

A search was undertaken to find sample composites from Gencor percussion drill holes lying within radii of 25mE x 25mN x 0.49mRL of each Gencor diamond drill sample using:

- Gencor percussion drill sample composites from all areas other than close-spaced holes in the trial mine area, for which XRF assays are available, excluding samples from below the water table and those in basement rock
- Sample composites from Gencor diamond drill holes in all areas, for which XRF assays are available, excluding samples from below the water table and those in basement rock.

The search located 358 nearest neighbour pairs separated by, on average, 2.7 metres. Figure 52 shows a quantile-quantile plot that compares the marginal histograms of the two grade populations. Percussion drill samples show consistently higher grades than do nearby diamond core samples: either the percussion drill samples are biased high or the core samples are biased low.

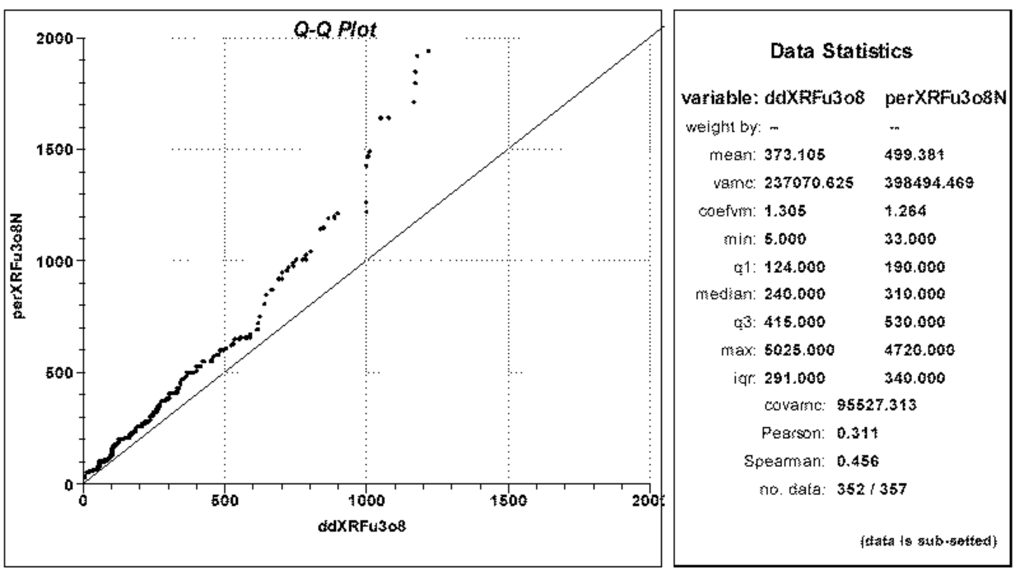


Figure 52: U_3O_8 grades in Gencor percussion holes and twin diamond core samples

13.5.2 Gencor Percussion Drill Samples and Test Shafts

There are 32 test shafts that were excavated after a percussion drill hole had been drilled at the centre of each. Figure 53 shows a scatter plot comparing paired one-metre samples in drill holes with bulk samples from the shafts, both populations being limited to samples from above the water table. The scatter on the plot gives an indication of the short-scale spatial variation in U_3O_8 grades. Figure 54 shows a quantile-quantile plot for the same data. As expected, considering the differences in volumes of the two sample types, percussion drill samples tend to be lower grades in low-grade mineralisation and higher in higher-grade mineralisation. The variance of the population of grades from the test shaft samples is lower, an expression of the volume-variance effect. Despite this, the means and medians of the two sample populations are very nearly equal, indicating that the percussion drill samples reliably represent the grade of mineralisation.

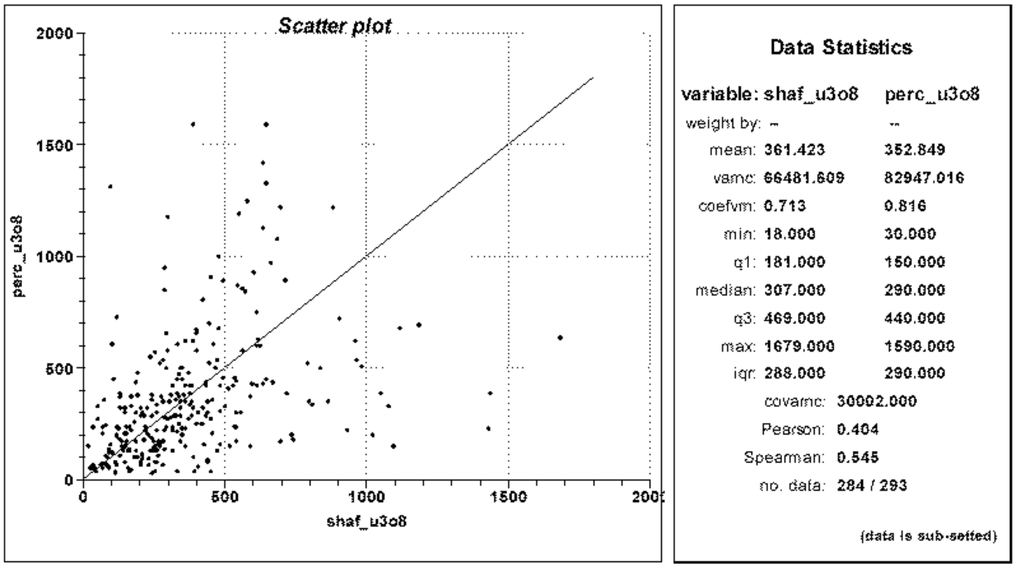


Figure 53: U_3O_8 grades in Gencor percussion drill samples and co-located test shafts

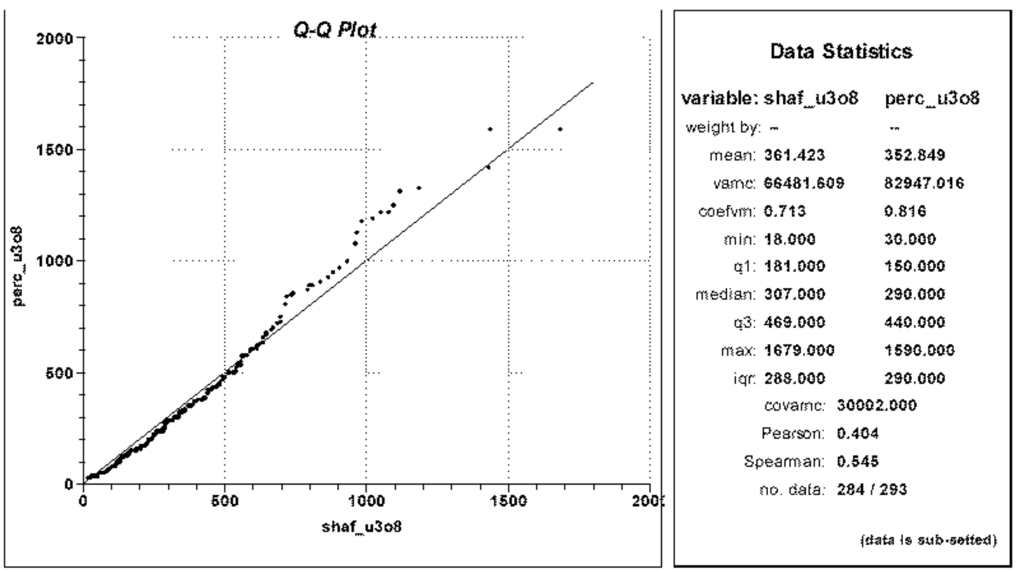


Figure 54: U_3O_8 grades in Gencor percussion drill samples and co-located test shafts

13.5.3 Gencor Diamond Core Samples and Test Shafts

There are four diamond core holes over which test shafts were subsequently excavated. Pairing of samples grading over 15ppm U_3O_8 yields only 29 samples for comparison but the bulk samples from shafts report significantly higher U_3O_8 grades than do core samples. Considering the comparisons of U_3O_8 grades in percussion drill samples and test shafts and twinned percussion and diamond core holes, above, it appears likely that grades in diamond core samples are biased low.

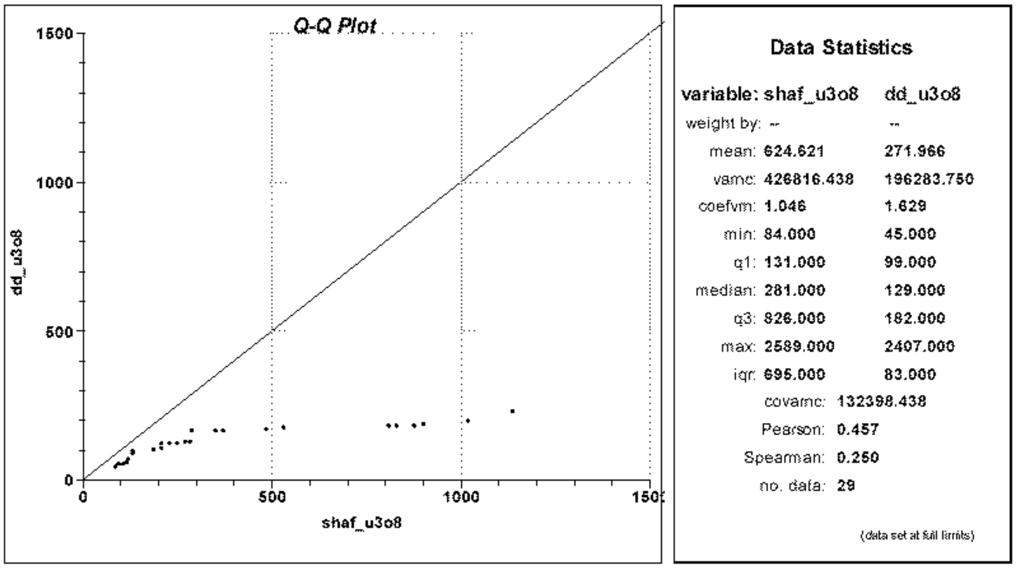


Figure 55: U_3O_8 grades in Gencor diamond core samples and co-located test shafts

13.6 Reliability of Gencor Wet Percussion Drill Samples

The reliability of Gencor percussion drill samples that were probably drilled wet can be gauged by comparison with several other sources of sample grades: nearby diamond drill core samples, nearby test shafts and radiometric logs of nearby Paladin RC drill holes. Only 16 shaft samples are available below the interpreted water table level so no useful comparison can be had for wet percussion drill samples. Of principal concern is the potential for down-hole sample contamination in percussion drill samples from below the water table.

13.6.1 Wet Percussion Drill Samples and Radiometric Grades in Nearby Paladin RC Drill Holes

A search was undertaken for nearest neighbour sample composites from:

- Radiometric U_3O_8 grades for one-metre down-hole intervals below the water table and above basement in Paladin drill holes within Detail 1.

- Gencor percussion drill sample composites from holes in Detail 1 other than close-spaced holes in the trial mine area, for which XRF assays are available, excluding samples from above the water table and those in basement rock

The search located 451 pairs of sample composites within search radii 50mE x 50mN x 1.0mRL, separated by, on average, 35 metres. In low-grade mineralisation, wet percussion drill samples return U_3O_8 grades slightly higher than those from down-hole logging of nearby Paladin drill holes. In high-grade mineralisation the trend is reversed.

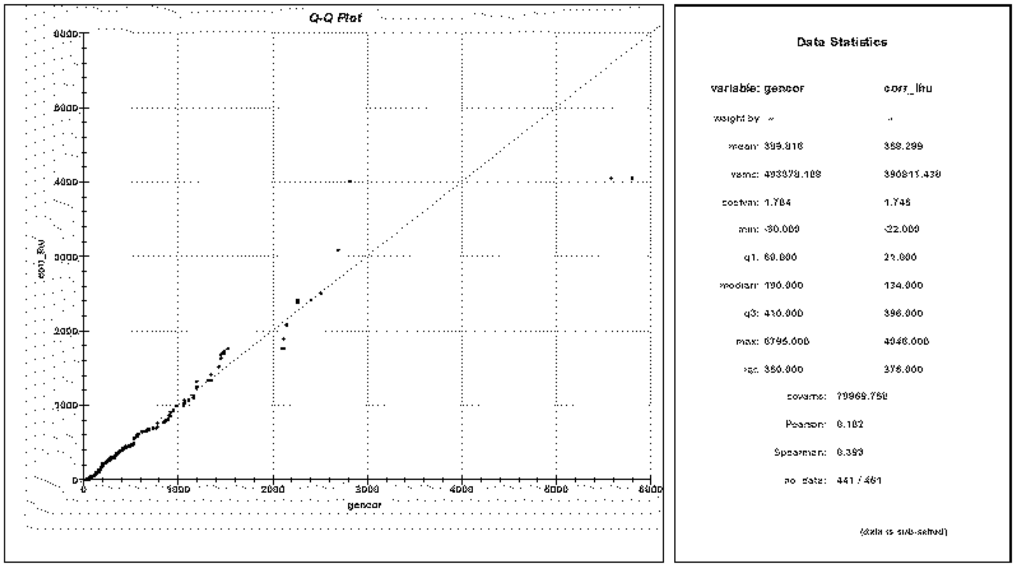


Figure 56: U_3O_8 grades in wet Gencor percussion drill samples and nearby Paladin gamma logs

13.6.2 Wet Percussion Drill Samples and Diamond Core Samples

A search was undertaken to find sample composites from Gencor percussion drill holes lying within radii of 25mE x 25mN x 0.49mRL of each Gencor diamond drill sample using:

- Gencor percussion drill sample composites from all areas other than close-spaced holes in the trial mine area, for which XRF assays are available, excluding samples from above the water table and those in basement rock

- Sample composites from Gencor diamond drill holes in all areas, for which XRF assays are available, excluding samples from above the water table and those in basement rock.

The search located 172 nearest neighbour pairs separated by, on average, 2.3 metres. Figure 57 shows a quantile-quantile plot comparing the histograms of grades in the two sample populations. Sample pairs are limited to relatively low-grade mineralisation and U_3O_8 grades in wet percussion drill samples are consistently higher than those in diamond core samples. Considering that U_3O_8 grades from diamond core samples are probably biased low, it may be concluded that the wet percussion drill samples have not been grossly upgraded due to sample contamination.

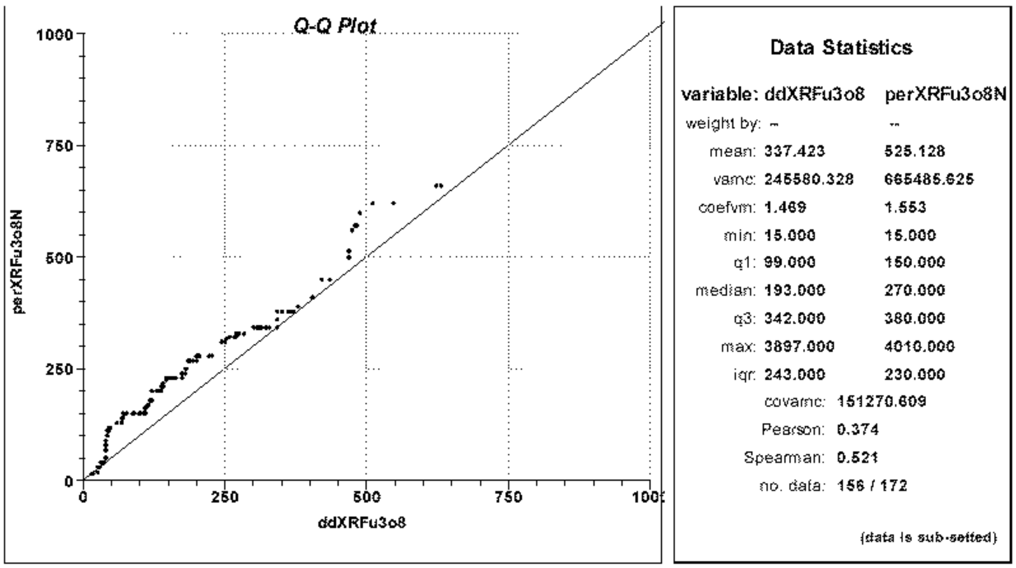


Figure 57: U_3O_8 grades in wet Gencor percussion drill samples and twin diamond cores

13.7 Conclusions

XRF assays of Acclaim and Paladin RC drill samples are regarded as accurate and precise and they compare very closely to grades derived from down-hole radiometric logging of RC holes. In high-grade mineralisation there is a tendency for radiometric logging to return slightly higher U_3O_8 grades. The two types of grade determinations are considered compatible for the purposes of resource estimation.

U_3O_8 grades in Gencor percussion drill samples from above the water table compare closely to grades of bulk samples from test shafts. A small volume-variance effect is evident and differences in grades between individual paired samples from the same depth interval give an indication of the short-scale spatial continuity of U_3O_8 grades. They also closely match grades derived from down-hole radiometric logging of nearby Acclaim and Paladin RC drill holes.

Comparisons of U_3O_8 grades in Gencor percussion drill samples from below the water table with grades in nearby Paladin RC drill holes, and with twin diamond core holes, indicate that the wet samples may have suffered from some down-hole sample contamination. There is, however, is no definitive evidence of significant upgrading of U_3O_8 grades in the percussion drill samples.

Comparisons of U_3O_8 grades in twinned Gencor percussion and diamond core holes, and twinned core holes and test shafts, indicate that grades in diamond core samples are probably biased low. This possibly relates to loss of matrix material, the main host of mineralisation, during drilling and possibly during subsequent sub-sampling of core.

Langer Heinrich Uranium employees were responsible for the collection of samples from the drill rig and for the splitting of samples prior to dispatch to SetPoint Laboratories, once despatched from site all sample preparation and analysis was handled by SetPoint.

The sampling methods, chain of custody procedures, sample preparation procedures and analytical techniques are all considered appropriate and are compatible with accepted industry standards.

14 Data Verification

14.1 Data Sources and Database Compilation

14.1.1 General

The Gencor/Acclaim database acquired by Paladin consists of two parts: a large hard copy database containing numerous reports, paper maps and files, and a digital database which includes drill hole data, geochemical assays, downhole logging data, topographic contours, meteorological data and a photographic archive. All materials located in the Swakopmund office and Acclaim's Perth offices were transferred to Paladin's office in Perth.

Paladin reviewed every document and compiled a bibliography listing title and tabulated category, author, language, date and number of pages of each record. The data were subdivided into nine principal categories: geology, geophysics, drilling, metallurgy, engineering/mining, environment, feasibility/planning summaries, corporate and photographs. The completed bibliography contains 897 separate reports, memoranda, etc. in 52 lever arch files.

14.1.2 Digital Database Compilation

Gencor

The first digital database of the percussion drilling was compiled by Gencor in 1976. When Acclaim acquired the project this database was available only as printouts; the electronic version could not be found.

Acclaim

Acclaim endeavoured to digitise all available drill hole data. Drill hole collars were entered from the Gencor computer printout and other survey sheets. The drill hole survey commissioned by Acclaim in July 1999 had confirmed that Gencor drill holes had been correctly reported to within 0.5m. The new survey data were entered for comparative purposes and the 1999 survey co-ordinates, where available, were used in preference to Gencor's data. A total of 1889 drill holes, including Acclaim's drilling, were entered into this database.

Assay data were compiled in sequence of data confidence with audit trail flags in the database to confirm validity. The assay-certificated data were captured first with date and report number also recorded. The Gencor printout assays that had no certificate numbering had to be confirmed as valid prior to inclusion into the new database. The printout consisted of grades for regular one-metre down-hole intervals for only the percussion drilling. Most of the diamond drill hole assay data utilised irregular intervals and have traceable assay certificates. The Gencor computer printout of one-metre intervals was validated by Acclaim by cross-referencing the respective assay certificates and repeat assay certificates.

A limited database of scintillometer readings from the collected drill samples and a very limited repeat assay database were also compiled and merged with the original assay database. Sample data from infill percussion drilling in the trial mining areas, and U_3O_8

grades for blast holes drilled in the mega-trench, were also captured, with assays cross-referenced to assay certificates.

Paladin

Paladin, in conjunction with H&S, found that the database collated by Acclaim contained numerous inconsistencies and inaccuracies that had significantly affected their resource estimates. Some of these inconsistencies and inaccuracies resulted from Gencor's use of local grids, each with differing drill hole name and numbering systems along the 15 kilometre strike length of the deposit. When Gencor digitised the drill hole data into a uniform global grid in 1976 some of the local grid drill hole names were retained, while some had to be changed because of the limitations of computer programs in use at that time. This resulted in some drill holes having two different drill hole names. Further, in many instances, Gencor did not differentiate between twinned holes, both often having the same identifier name although one may have been a diamond and the other a percussion hole. Only some of these "double" holes were removed by Acclaim when compiling their database. Also most holes without intersections greater than 100ppm U_3O_8 were not recorded in the Gencor database and this practice was followed by Acclaim. Both Gencor and Acclaim failed to enter < (less than) values as such in their respective databases, recording some values as 0 (zero) and others as 10 or 30ppm depending on the detection limit. Also, drill hole intervals that had obviously been assayed as composite samples were listed by Acclaim as consecutive one-metre sample intervals with the same U_3O_8 grade.

Corrections and validation by Paladin have included:

- Plotting drill holes on stacked sections to locate obvious errors and duplications;
- Checks of drill hole numbers in relation to their locations to ensure that transposition from Gencor local grid co-ordinates to global grid co-ordinates was correct;
- Checks of drill hole location against hardcopy Gencor maps and cross-sections, identifying missing drill holes;
- Checking easting, northing and elevation entries against Gencor computer printouts;
- Cross-referencing updated hole names to previous Gencor hole names where consolidation of the database had necessitated changes;
- Checks of hardcopy files and re-entry of correct down-hole sample intervals and U_3O_8 assays, cross-referenced to assay certificates where available;
- Suffixing of diamond drill holes (D), twin percussion drill holes (T) and exploration shafts (P); and
- Entry of lithological logs for the 396 holes for which logging is available.

The work resulted in the removal of 181 duplicated drill holes from the database. Location co-ordinates for over 80 drill holes were corrected and 135 missing drill holes were added. Paladin have incorporated all the recent drilling into this database which now includes 2,234, percussion exploration drill holes and 72 diamond drill holes for a total of 2,306 exploration drill holes. All 128 corner holes drilled to establish the exploration shafts were found to be mis-located in relation to the centre holes and these were corrected. Data for the 32 exploration shafts and 253 infill and blast holes in the trial mining area were added to the main drill hole collar file.

14.1.3 Drill Hole Collar Locations

Locations of drill hole collars and test shafts were checked by independent licensed surveyors in 1999 and are believed to be reliable (Section 14.1.2, above). In 2004 H&S geologist David Princep walked a substantial number of Gencor drill traverses and located about eight per cent of drill hole collars, with another eight per cent of hole locations marked by pegs but with no visible remnant collars. Collars of several holes not presently incorporated into the digital database were also located. H&S considers that the uncaptured drill holes are unlikely to significantly affect the resource estimates. About 60 per cent of Acclaim RC drill hole collars were located in the field by H&S.

14.1.4 Sample and Assay Information

Sample and assay data were provided in digital form by Paladin. H&S has cross-checked about 25 per cent of Gencor sample intervals and 60 per cent of Acclaim sample intervals in the digital database to hardcopy assay certificates.

Paladin has used a series of alphabetical codes in the drill hole collar and assay files to track drill hole and sample type. H&S added an equivalent series of numeric codes (Table 6) so that sample types could be tracked through the down-hole compositing process.

Similarly, sample intervals in the Paladin database contain codes describing the source of U_3O_8 grades. A series of numeric codes was added (Table 7) in order that this information could also be carried through the compositing process.

Sample type	Alpha code	Numeric code
Gencor regional percussion drill sample	HPREG	1
Gencor percussion drill sample	HPEXP	2
Gencor percussion drill hole at centre of a test shaft	HPPCE	3
Gencor percussion drill hole at corner of a test shaft	HPPCO	4
Gencor percussion drill hole in "geostatistical star", Detail 2 only	HPSTAR	5
Gencor percussion drill hole, no samples assayed by XRF	HPNOT	6
Gencor infill percussion drill hole in trial mining area	HPMINE	11
Gencor infill percussion drill hole in trial mining area, no samples assayed by XRF	HPMINENOT	12
Gencor percussion drill hole in mega-trench area	HPTRENCH	13
Gencor diamond drill hole	HD	21
Gencor diamond drill hole that twins a percussion hole	HDD	22
Gencor percussion drill hole at centre of a test shaft	HDPCE	23
Gencor diamond drill hole, no samples assayed by XRF	HDNOT	24
Gencor bulk samples from test shaft	SHAFT	31
Gencor blasthole samples from trial pit	BLASTHOLE	41
Acclaim RC drill samples	NEWRAD	51
Paladin RC drill samples	PALADIN	61

Table 6: Sample type codes and H&S numeric equivalents

Assay type	Alpha code	Numeric code
Gencor XRF	XRF	1
Gencor printout	PRINTOUT	2
Gencor Radiometric count	e U_3O_8	3
Gencor sample assumed below detection level		4
Assumed average over core loss interval in Gencor dd		5
Assumed missing from Gencor hole		6
Acclaim down-hole radiometric log	e U_3O_8	11
Acclaim XRF assay	XRF	12
Acclaim sample interval with no grade available		13
Paladin down-hole radiometric log	e U_3O_8	14

Table 7: Assay type codes and H&S numeric equivalents

14.1.5 Geological Mapping and Logs

A good geological base map compiled at 1:40,000 scale by Gencor is available in hardcopy form. The map shows bedrock lithologies, the extents of valley-fill sediments, calcrete terraces and recent alluvium. The mapped limits of valley-fill sediments were used by Paladin geologists to aid interpretation of the basement-sediment interface throughout the resource area.

Drill hole logs are available for 857 holes. Paladin has simplified lithological descriptions into a series of concatenated codes as shown in Table 8.

Rock type code	Description
CZ	Recent alluvium / colluvium
T	Undifferentiated valley fill
TC	Undifferentiated calcrete-cemented sediments
TCC	Calcareous conglomerate
TCG	Calcareous grit
TCS	Calcareous silt/clay
TYC	Clayey conglomerate
TYG	Clayey grit
TYS	Clayey sand
TYY	Clay/silt
P	Proterozoic bedrock
N	No log available

Table 8: Lithological logging codes

With relatively few geological logs available it is not possible to create a model of rock types to, for example, separate indurated calcrete material from poorly cemented valley-fill sediments.

14.1.6 Topography

A triangulated topographic surface was supplied in dxf format by Paladin Resources. That surface has been constructed using photogrammetry based on a low-level survey flown in 2004. Ground checks by a licensed surveyor have indicated that the original work is reliable (Section 14.1.2, above)

14.1.7 Valley Sediments Limits

Paladin geologists provided a series of strings, digitised on cross-sections, representing the base and edge limits of valley-fill sediments. Their interpretation was based on drill hole geological logs and on Gencor geological mapping. H&S formed the strings into a triangulated surface, then combined that surface with the topographic DTM for areas of exposed basement to form a surface that represents palaeotopography, or the basement interface.

14.1.8 Water Table

There are observations of standing water levels available for 829 drill holes: 268 Gencor holes, 100 Acclaim holes and 461 Paladin holes. A triangulated surface was constructed using the point data and extended north and south to cover areas peripheral to the area within resources were to be estimated. There are no observations available for holes in Detail 7. The western end of the triangulated surface was extended horizontally to cover that area.

During the trenching programme, several small perched pockets of water were encountered. The maximum water flow was 2 cubic metres per hour at 606.5m AMSL, indicating that the rocks have low permeability. The mega-trench, which at the base measured 200 metres by x 12 metres, was excavated to 3m below the water table into uncemented ore. Water levels in the excavation rose at a rate of 4cm/day during the first 8 weeks following excavation.

Figure 58 shows the water table surface along with the basement interface surface.

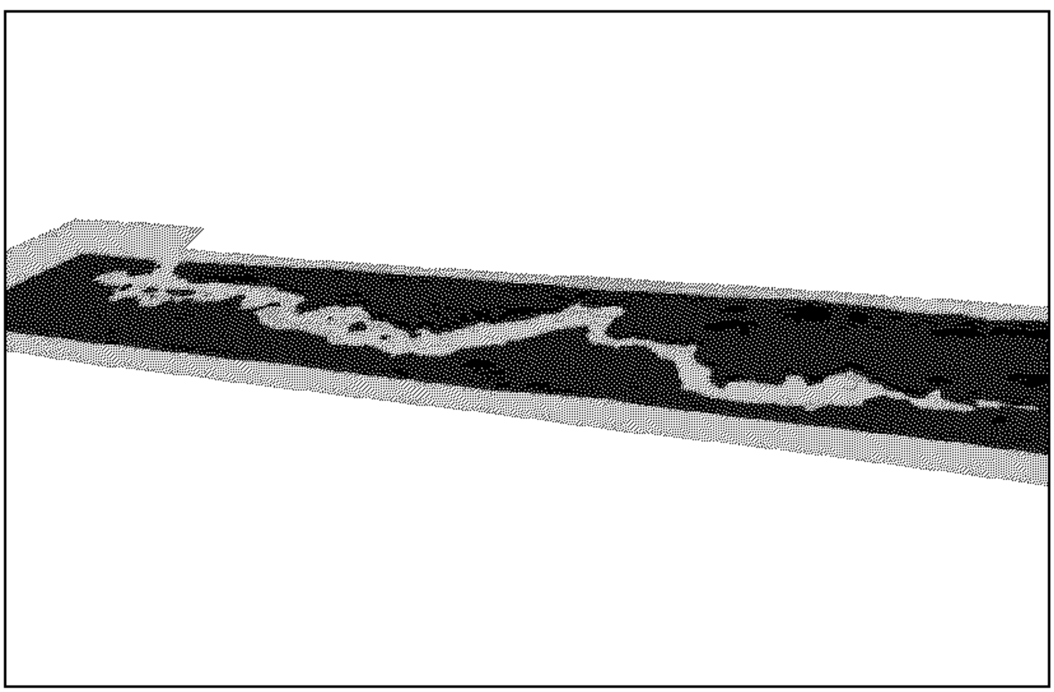


Figure 58: Perspective view of water table and basement interface surfaces, looking NW

14.1.9 Bulk Density

During mining of the mega-trench and trial pit Gencor weighed a substantial proportion of trucks. The surveyed volumes of the excavations, truck counts and truck weights yielded an average bulk density estimate of 2.1g/cc, varying from 1.95g/cc for clay through to 2.25g/cc for carbonate-cemented sediments.

During subsequent screening and pilot plant operations Gencor determined densities on 16 finely ground samples, resulting in an average wet density of 2.56g/cc. Densities were also measured on nine lump rock samples, yielding an average dry density of 2.38g/cc.

Acclaim attempted to improve the bulk density database by down-hole logging of 10 RC drill holes using a gamma-gamma probe with a Ce134 active isotope source to log density. The system reads "formation densities" (i.e., pore space is seen as pore space) and, in dry conditions, normally yields results that are directly comparable to true rock bulk density. The logging yielded average bulk densities of 2.28g/cc for material above the water table and 2.51 below the water table. Using only sections with U_3O_8 grades of less than 100ppm (higher uranium concentrations can interfere with the method), average density was 2.4g/cc above the water table and 2.5 below.

H&S regards the results of Gencor's trial mining as the more reliable available estimate. It is possibly conservative. As indicated by Gencor's mining records, density is likely to vary by between 5 and 10 per cent across different material types but there is insufficient information available to create a reliable density model. The effect on the reliability of resource estimates is considered inconsequential for the purposes of the present work.

15 Adjacent Properties

See NI43-101, Langer Heinrich, Namibia, Independent Technical Report, Resource and Reserve Estimation, 7th June 2005 .

16 Mineral Processing and Metallurgical Testing

See NI43-101, Langer Heinrich, Namibia, Independent Technical Report, Resource and Reserve Estimation, 7th June 2005.

17 Mineral Resource Estimates

17.1 Indicator Kriging for Resource Estimation

The MIK method was developed in the early 1980's with a view toward addressing some of the problems associated with estimation of resources in mineral deposits. These problems arise where sample grades show the property of extreme variation and consequently where estimates of grade show extreme sensitivity to a small number of very high grades. These characteristics are typical of many metal deposits where the component of interest comprises a very small proportion of the rock mass, for example lode gold deposits, and where the coefficient of variation in samples is commonly 1.5 or higher 2. MIK is one of a number of methods that can be used to provide better estimates than the more traditional methods such as ordinary kriging and inverse distance weighting.

It is fundamental to the estimation of resources that the estimation error is inversely related to the size of the volume being estimated. To take the extreme case, the estimate of the average grade of a deposit generated from a weighted average grade of the entire sample data set is much more reliable than the estimate of the average grade of a small block of material within the deposit generated from a local neighbourhood of data.

Another fundamental notion relevant to the optimisation of resources to develop an open pit mine and schedule is that the optimisation algorithm does not require the resource be defined on extremely small blocks relative to data spacing.

The basic unit of an MIK block model is a panel that normally has the dimensions of the average drill hole spacing in the horizontal plane. The panel should be large enough to contain a reasonable number of blocks, or Selective Mining Units (SMUs; about 15). The SMU is the smallest volume of rock that can be mined separately as ore or waste and is usually defined by a minimum mining width. At Langer Heinrich, the dimensions of this block are assumed to be in the order of 5mE x 5mN x 2mRL.

The goal of MIK is to estimate the tonnage and grade of ore that would be recovered from each panel if the panel were mined using the SMU as the minimum selection criteria to distinguish between ore and waste. To achieve this goal, the following steps are performed:

1. Estimate the proportion of each domain within each panel. This estimation can be achieved by kriging of indicators of domain classifications of sample data points or by using wireframes. In Details 1, 2 and 5 of the Langer Heinrich model, the proportions of each domain in each panel were estimated by indicator kriging. The proportions of each panel above and below the water table, i.e., the proportions of subdomain 1 and subdomain 2, in each panel were estimated by cutting the panels with the triangulated water table surface.

2. Estimate the histogram of grades of sample-sized units within each domain within each panel using MIK. MIK actually estimates the probability of the grade within each panel being less than a series of indicator threshold grades. These probabilities are interpreted as panel proportions.

3. For each domain, and for each panel that receives an estimated grade greater than 0 ppm U_3O_8, implement a block support correction (variance adjustment) on the estimated histogram of sample grades in order to achieve a histogram of grades for SMU-sized blocks. This step incorporates an explicit adjustment for Information Effect.

4. Calculate the proportion of each panel estimated to exceed a set of selected cut-off grades, and the grades of those proportions.

5. Apply to each panel, or portion of a panel below surface, a bulk density to achieve estimates of recoverable tonnages and grades for each panel.

Apart from considerations of resource confidence classification, Step 5 completes construction of the resource model. The estimates of resources for each panel may be combined to provide an estimate of global resources for the deposit.

Panel Model Extents			
	East	North	Elevation
Detail 1			
Panel origin (centroid)	32400	-90675	574
Panel dimensions	50	50	4
No. of panels	63	28	30
Panel discretisation	4	4	4
Detail 2			
Panel origin (centroid)	30425	-90425	562
Panel dimensions	50	50	4
No. of panels	50	22	16
Panel discretisation	4	4	4
Detail 3			
Panel origin (centroid)	37725	-90275	650
Panel dimensions	50	50	4
No. of panels	24	26	15
Panel discretisation	4	4	4
Detail 4			
Panel origin (centroid)	40225	-91225	698
Panel dimensions	50	50	4
No. of panels	54	20	11
Panel discretisation	4	4	4
Detail 5			
Panel origin (centroid)	35225	-90925	602
Panel dimensions	50	50	4
No. of panels	50	29	23
Panel discretisation	4	4	4
Detail 6			
Panel origin (centroid)	38925	-90975	678
Panel dimensions	50	50	4
No. of panels	24	20	12
Panel discretisation	4	4	4
Detail 7			
Panel origin (centroid)	28825	-89475	554
Panel Dimensions	50	50	4

No. of panels	28	20	14
Panel Discretisation	4	4	4
Kriging Search Parameters (all areas)*			
Criteria	Measured	Indicated	Inferred
Min no. of data	16	16	8
Max no. of data per octant	6	6	6
Min no. of octants with data	4	4	2
X (east) search radius (metres)	55	71.5	100
Y (north) search radius (metres)	55	71.5	100
Z (rl) search radius (metres)	4	5.2	5.2
Search rotations			
	Rotation axis		Rotation
All domains	nil		nil

Table 9: Resource model panel extents

17.2 Indicator Kriging Parameters

The input parameters to Indicator Kriging of the Langer Heinrich mineralisation include:

- Indicator variogram models describing the spatial continuity of indicator variables within each domain at each indicator threshold.
- Variograms describing the spatial continuity of U_3O_8 grades within each domain.
- Mean U_3O_8 grades of each of the indicator classes within each domain.

Figure 135 to Figure 143 list the indicator variogram models applied in each of the geological domains. The last variogram model listed in each table is the variogram model of U_3O_8 grades, used for calculation of variance adjustments.

Table 18 to Table 26 list the conditional statistics of sample data in each of the modelling domains. The statistics in Detail 1 domain 1 were calculated on data that exclude the close-spaced drilling in the trial mining area but those samples were included in the data that inform the indicator kriging estimates.

Table 9 shows the grid framework and kriging search parameters used in the indicator kriging models. Within each Detail, the boundaries between domains and between subdomains were treated as soft boundaries in the kriging process.

17.3 Derivation of Preferred U_3O_8 Assays

For Gencor drill holes and test shafts the following scheme was adopted:

1. Where U3O8-1 (XRF assay) ≠ blank and is >0, accept U3O8-1 (9981 records)
2. Where U3O8-1 = -30 adopt 15ppm, where = -20 adopt 10ppm, where = -10 adopt 5ppm, where = -3 adopt 1ppm, where = -1 adopt 0ppm.
3. Where U3O8-1 = blank and print-U3O8 ≠ blank and is >0, accept print-U3O8 (2817 records)

4. Where U3O8-1 = blank and print-U3O8 ≠ blank and is = 0, adopt 15ppm (726 records)
5. Where U3O8-1 = blank and print-U3O8 = blank and eU3O8 ≠ and is >-100, accept eU3O8 (528 records). Replace -10 with 5ppm and 0 with 15ppm.
6. Where U3O8-1 = blank and print-U3O8 = blank and eU3O8 = blank and U3O8-rsp-1 ≠ blank, accept U3O8-rsp-1.
7. In diamond drill holes where no assay or eU3O8 is available and sample length ≥ 1 metre, adopt grade of 5ppm on the basis that core was scanned with a scintillometer and deliberately not assayed. Affected intervals are normally at the tops and bottoms of core holes. Exceptions to this procedure were applied to hole K7-D 8.87-10.08m and M4-D 7.37-8.48m which probably represent intervals of core loss. These two intervals were treated as for shorter core loss intervals, below.
8. In diamond core holes where no assay or eU3O8 is available and sample length is < 1 metre, at tops and bottoms of holes adopt 5ppm. At other down-hole depths assume core loss and adopt a length-weighted average of grades from sample intervals immediately above and below the affected interval. Affected intervals are mainly less than 5cm in length.
9. Where type = HPEXP, HPMINE, MPPCE, HPPCO or HPTRENCH and U3O8-1 = blank and print-U3O8 = blank and eU3O8 = blank, adopt 15ppm on the basis that all percussion drill samples were checked by scintillometer and those with low counts were not submitted for XRF assay. (1330 HPEXP records, 24 HPPCE records, 36 HPPCO records, 0 HPMINE records, 15 HPTRENCH records).
10. Where type = HPSTAR and no U3O8 grade is available, adopt -99999. These holes were selectively assayed and missing data should not be treated as low grades.
11. Where type = SHAFT and no U3O8 grade is available, adopt 15ppm in near-surface intervals not assayed and -99999 in lower intervals not assayed.
12. Where type = HPREG and U3O8-1 = blank, adopt 0ppm. These holes were assayed by low-level technique with lower detection limit of 1ppm.
13. Where type = HPNOT or HDNOT and no U3O8 grade is available, adopt 15ppm assuming all samples were read by scintillometer and deliberately not assayed because of low counts. Subsequent viewing of data section-by-section demonstrated that this was not sensible in all cases and intervals in holes SL72, SL75, SL81, SL94, SL95, SL96, SL97, SL98, SL99 (mega trench holes) and CE2, CF4, Z3, R1, H5, N7, L2, G3 and 28W1S were allocated -99999.
14. Where type = HPMINENOT and no U3O8 grade is available, adopt -99999.

In Acclaim RC drill holes:

15. Where eU3O8 ≠ blank accept eU3O8
16. Where eU3O8 = blank and u3O8-1 ≠ blank, accept the XRF assay at U3O8-1. Note that Paladin moved samples below 15 metres depth up-hole by one metre based on metre-by-metre comparisons against U3O8 grades from down-hole radiometric logging.
17. Otherwise set U3O8 to -99999.

In Paladin RC drill holes:

18. Where eU3O8 ≠ blank accept eU3O8

19. Otherwise set U3O8 to -99999.

17.4 Compositing

Figure 59 to Figure 65 show the native sample intervals employed by Gencor in drill holes in each of the Details. The majority of intervals for which U_3O_8 assays are available are one-metre lengths in all areas.

After derivation of preferred U_3O_8 sample grades, weighted average grades were calculated for uniform one-metre down-hole composite intervals in all drill holes and test shafts. Residuals less than 0.5 metres length were discarded. Composited intervals receiving negative U_3O_8 grade that had been affected by unsampled intervals were also discarded. Numeric codes for sample type and assay method were carried across in the compositing process. Composites were then assigned, by their mid-point locations, as being above or below the water table and above or below the interpreted basement surface. Summaries of the numbers of resulting data by area and sample type are shown in Table 10to Table 16.

Figure 66 to Figure 72 show data in each of the details. In the figures, data are sorted prior to display such that highest-grade samples are plotted last. This is a useful way of highlighting trends in U_3O_8 grades.

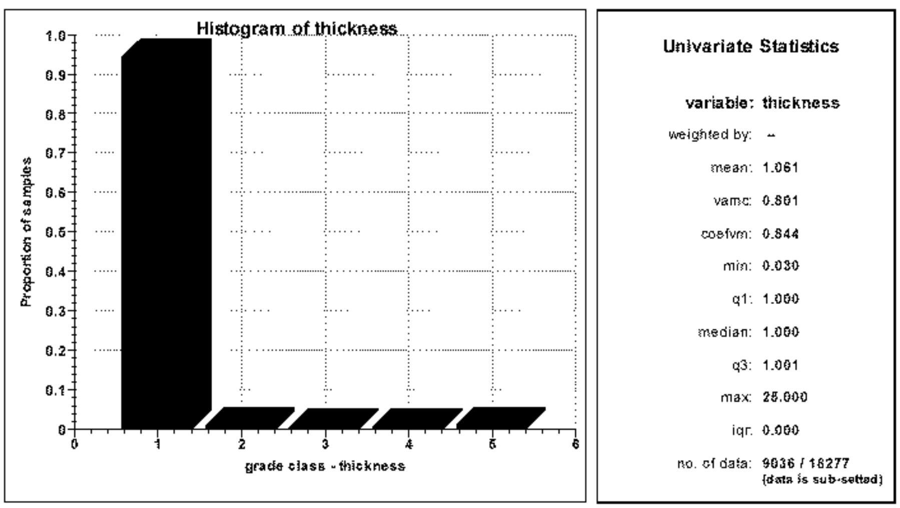


Figure 59: Lengths of Gencor native sample intervals with U_3O_8 assays, Detail 1

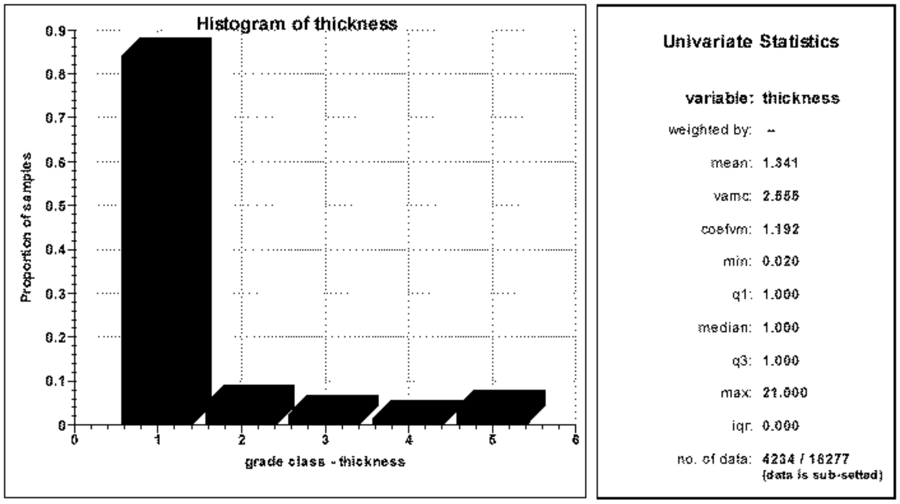


Figure 60: Lengths of Gencor native sample intervals with U_3O_8 assays, Detail 2

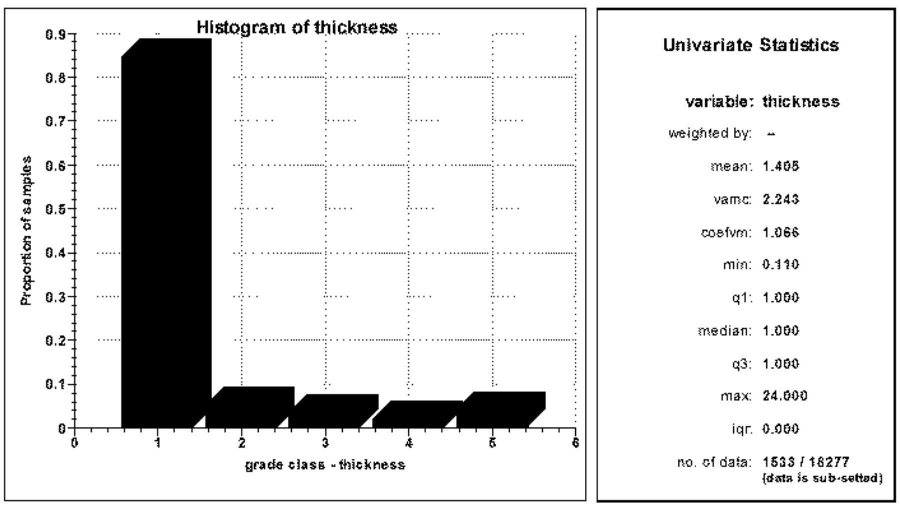


Figure 61: Lengths of Gencor native sample intervals with U_3O_8 assays, Detail 3

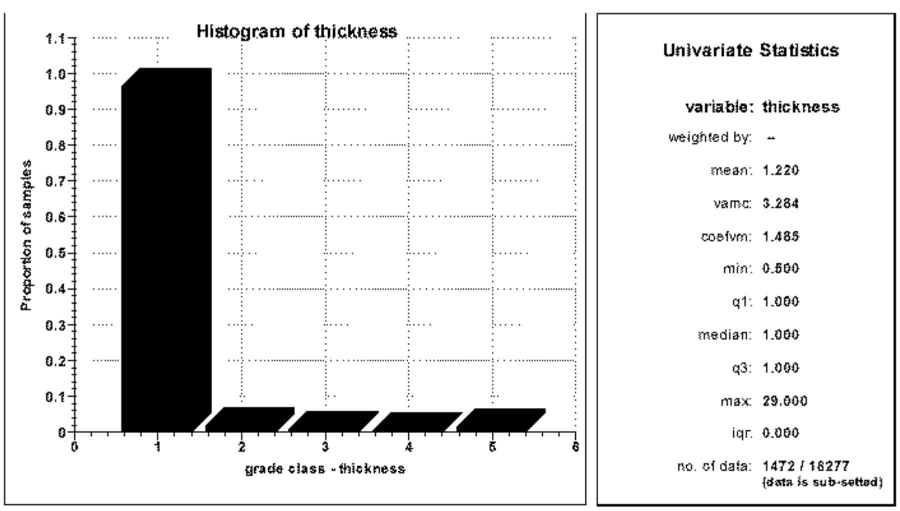


Figure 62: Lengths of Gencor native sample intervals with U_3O_8 assays, Detail 4

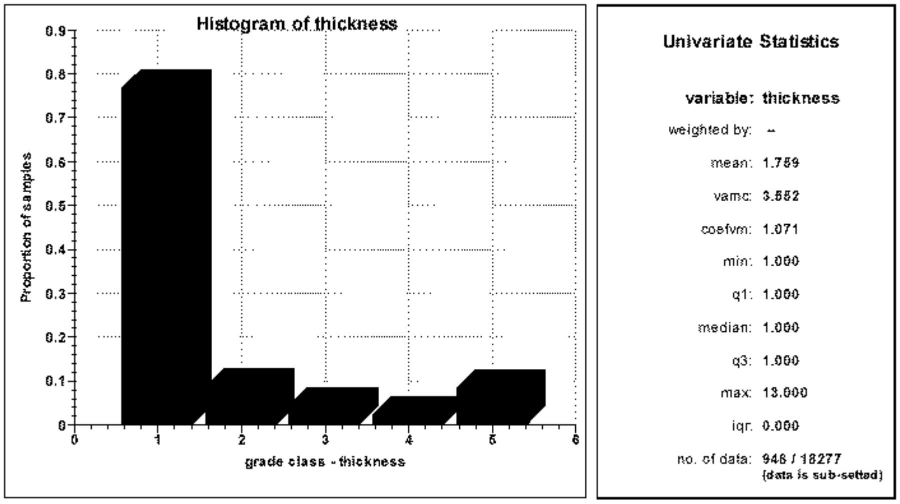


Figure 63: Lengths of Gencor native sample intervals with U_3O_8 assays, Detail 5

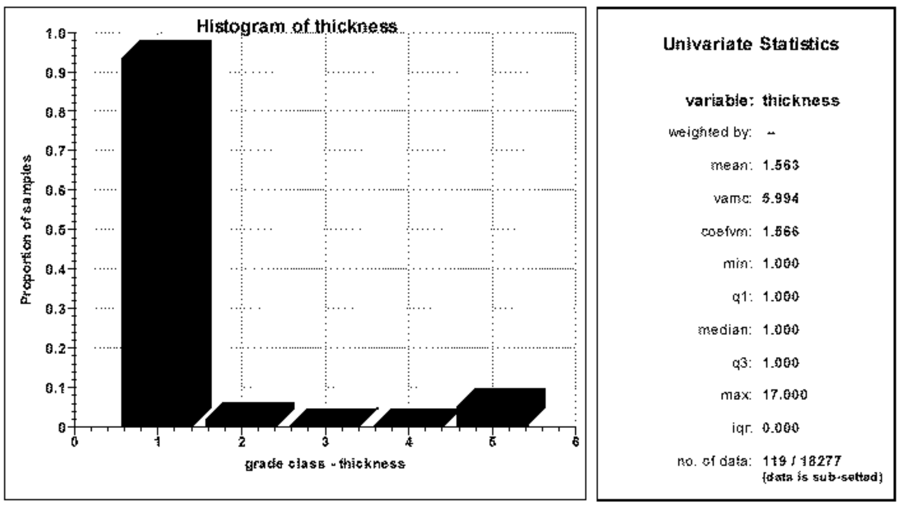


Figure 64: Lengths of Gencor native sample intervals with U_3O_8 assays, Detail 6

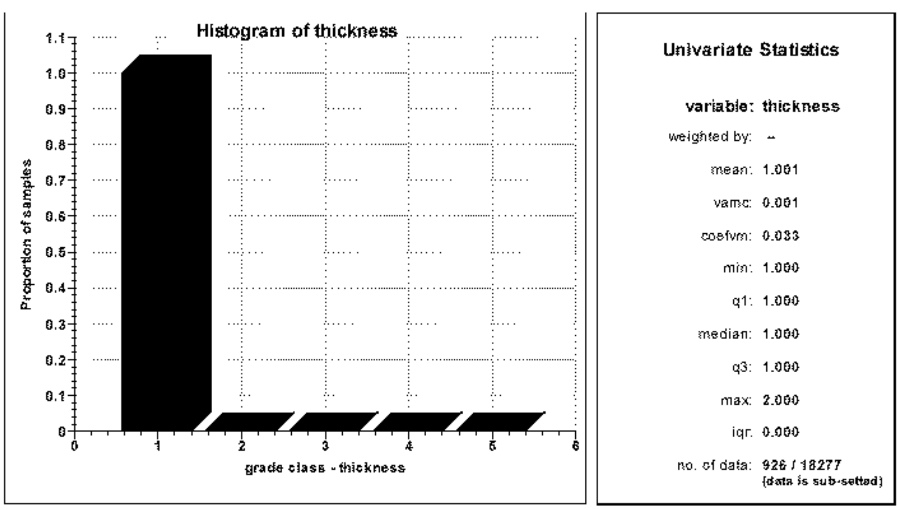


Figure 65: Lengths of Gencor native sample intervals with U_3O_8 assays, Detail 7

Sample type	XRF	Printout	e U_3O_8	Assumed < detection	Above water	Below water	Below basement
HPREG	65	0	0	29	91	3	0
HPEXP	1183	1890	6	1461	3927	613	41
HPPCE	56	56	2	15	117	12	0
HPPCO	380	16	0	101	457	40	0
HPSTAR	0	0	0	0	0	0	0
HPNOT	0	0	0	546	499	47	5
HPMINE	711	0	0	0	640	71	0
HPMINENOT	18	0	0	0	18	0	0
HPTRENCH	479	0	0	17	484	12	3
HD	92	27	14	158	142	150	0
HDD	215	53	20	78	202	164	6
HDPCE	0	0	0	0	0	0	0
HDNOT	6	0	0	33	23	16	0
SHAFT	95	0	0	17	105	7	0
BLASTHOLE	1264	0	0	0	1264	0	0
NEWRAD	158	-	2818	0	1553	1423	119
PALADIN	-	-	5325	0	1646	3519	160

Table 10: Composites in Detail 1 by sample type and assay type

Sample type	XRF	Printout	e U_3O_8	Assumed < detection	Above water	Below water	Below basement
HPREG	0	0	0	0	0	0	0
HPEXP	1480	2441	14	811	3868	878	99
HPPCE	54	130	0	26	176	35	26
HPPCO	567	0	0	58	557	68	37
HPSTAR	490	0	0	0	340	150	13
HPNOT	0	0	0	1200	1058	142	70
HPMINE	-	-	-	-	-	-	-
HPMINENOT	-	-	-	-	-	-	-
HPTRENCH	-	-	-	-	-	-	-
HD	73	10	10	30	88	35	0
HDD	100	28	10	36	136	38	7
HDPCE	32	6	3	16	47	10	3
HDNOT	0	0	0	0	0	0	0
SHAFT	143	0	0	0	133	10	4
BLASTHOLE	-	-	-	-	-	-	-
NEWRAD	4	-	121	0	87	38	3
PALADIN	-	-	5697	-	1576	3923	198

Table 11: Composites in Detail 2 by sample type and assay type

Sample type	XRF	Printout	e U_3O_8	Assumed < detection	Above water	Below water	Below basement
HPREG	0	0	0	0	0	0	0
HPEXP	1447	304	3	935	2424	265	54
HPPCE	57	3	0	3	58	5	0
HPPCO	214	0	0	1	204	11	0
HPSTAR	-	-	-	-	-	-	-
HPNOT	0	0	0	228	199	29	0
HPMINE	-	-	-	-	-	-	-
HPMINENOT	-	-	-	-	-	-	-
HPTRENCH	-	-	-	-	-	-	-
HD	0	0	0	0	0	0	0
HDD	75	3	3	17	68	30	0
HDPCE	0	0	0	0	0	0	0
HDNOT	0	0	0	0	0	0	0
SHAFT	49	0	0	0	47	2	0
BLASTHOLE	-	-	-	-	-	-	-
NEWRAD	-	-	-	-	-	-	-
PALADIN	-	-	1752	-	340	1357	55

Table 12: Composites in Detail 3 by sample type and assay type

Sample type	XRF	Printout	e U_3O_8	Assumed < detection	Above water	Below water	Below basement
HPREG	0	0	0	0	0	0	0
HPEXP	161	817	0	903	1693	188	74
HPPCE	37	88	0	28	144	9	5
HPPCO	591	0	0	1	555	37	47
HPSTAR	-	-	-	-	-	-	-
HPNOT	0	0	0	20	16	4	6
HPMINE	-	-	-	-	-	-	-
HPMINENOT	-	-	-	-	-	-	-
HPTRENCH	-	-	-	-	-	-	-
HD	0	0	0	0	0	0	0
HDD	0	0	0	0	0	0	0
HDPCE	0	0	0	0	0	0	0
HDNOT	0	0	0	0	0	0	0
SHAFT	102	0	0	21	123	0	0
BLASTHOLE	-	-	-	-	-	-	-
NEWRAD	-	-	-	-	-	-	-
PALADIN	-	-	127	-	32	80	15

Table 13: Composites in Detail 4 by sample type and assay type

Sample type	XRF	Printout	e U_3O_8	Assumed < detection	Above water	Below water	Below basement
HPREG	0	0	0	0	0	0	0
HPEXP	1242	237	1	1639	2551	568	50
HPPCE	0	0	0	0	0	0	0
HPPCO	0	0	0	0	0	0	0
HPSTAR	-	-	-	-	-	-	-
HPNOT	0	0	0	539	534	5	60
HPMINE	-	-	-	-	-	-	-
HPMINENOT	-	-	-	-	-	-	-
HPTRENCH	-	-	-	-	-	-	-
HD	103	40	0	165	179	129	0
HDD	31	14	0	55	49	51	10
HDPCE	0	0	0	0	0	0	0
HDNOT	0	0	0	0	0	0	0
SHAFT	0	0	0	0	0	0	0
BLASTHOLE	-	-	-	-	-	-	-
NEWRAD	-	-	-	-	-	-	-
PALADIN	-	-	2753	-	346	2298	109

Table 14: Composites in Detail 5 by sample type and assay type

Sample type	XRF	Printout	e U_3O_8	Assumed < detection	Above water	Below water	Below basement
HPREG	0	0	0	0	0	0	0
HPEXP	77	109	0	150	288	48	7
HPPCE	0	0	0	0	0	0	0
HPPCO	0	0	0	0	0	0	0
HPSTAR	-	-	-	-	-	-	-
HPNOT	0	0	0	350	285	65	0
HPMINE	-	-	-	-	-	-	-
HPMINENOT	-	-	-	-	-	-	-
HPTRENCH	-	-	-	-	-	-	-
HD	0	0	0	0	0	0	0
HDD	0	0	0	0	0	0	0
HDPCE	0	0	0	0	0	0	0
HDNOT	0	0	0	0	0	0	0
SHAFT	0	0	0	0	0	0	0
BLASTHOLE	-	-	-	-	-	-	-
NEWRAD	-	-	-	-	-	-	-
PALADIN	-	-	874	-	346	502	26

Table 15: Composites in Detail 6 by sample type and assay type

Sample type	XRF	Printout	e U_3O_8	Assumed < detection	Above water	Below water	Below basement
HPREG	0	0	0	0	0	0	0
HPEXP	399	0	528	1673	1501	1099	0
HPPCE	0	0	0	0	0	0	0
HPPCO	0	0	0	0	0	0	0
HPSTAR	-	-	-	-	-	-	-
HPNOT	0	0	0	1780	1761	19	0
HPMINE	-	-	-	-	-	-	0
HPMINENOT	-	-	-	-	-	-	0
HPTRENCH	-	-	-	-	-	-	0
HD	0	0	0	0	0	0	0
HDD	0	0	0	0	0	0	0
HDPCE	0	0	0	0	0	0	0
HDNOT	0	0	0	0	0	0	0
SHAFT	0	0	0	0	0	0	0
BLASTHOLE	-	-	-	-	-	-	-
NEWRAD	-	-	-	-	-	-	-
PALADIN	-	-	6681	-	1451	5049	181

Table 16: Composites in Detail 7 by sample type and assay type

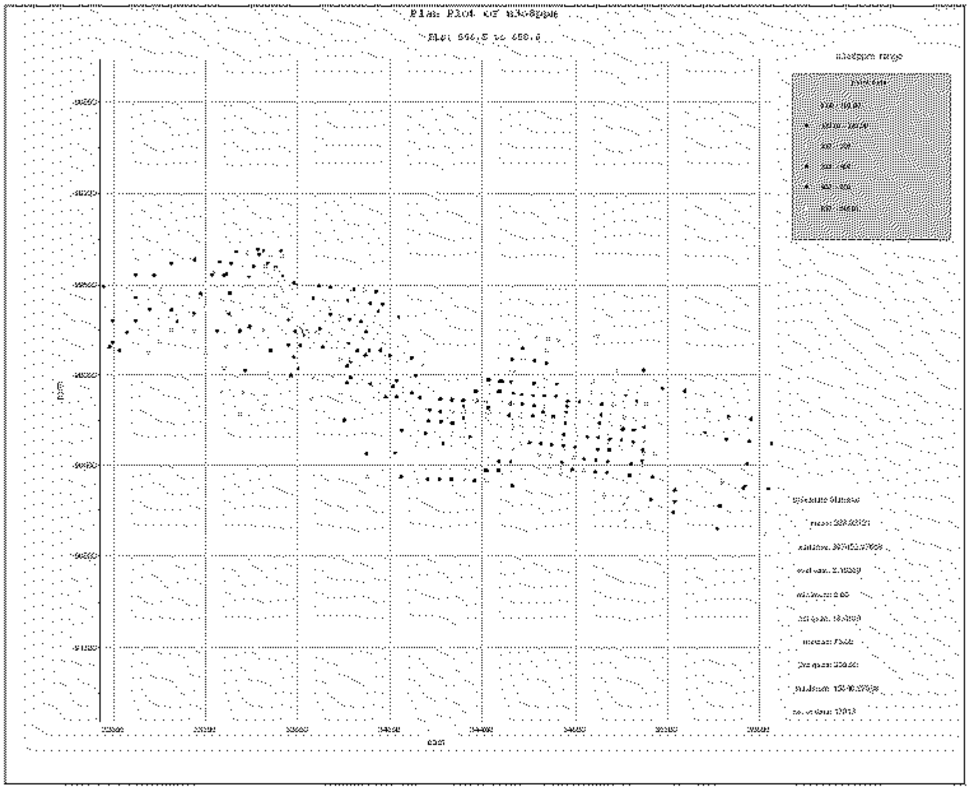


Figure 66: Sample composites in Detail 1

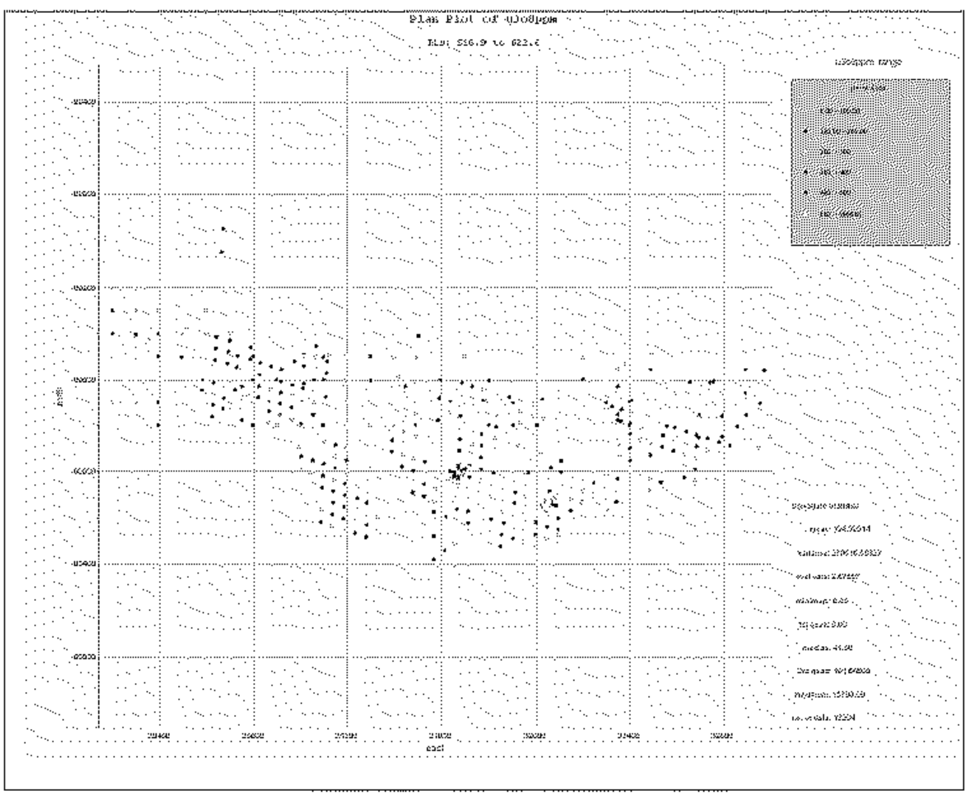


Figure 67: Sample composites in Detail 2

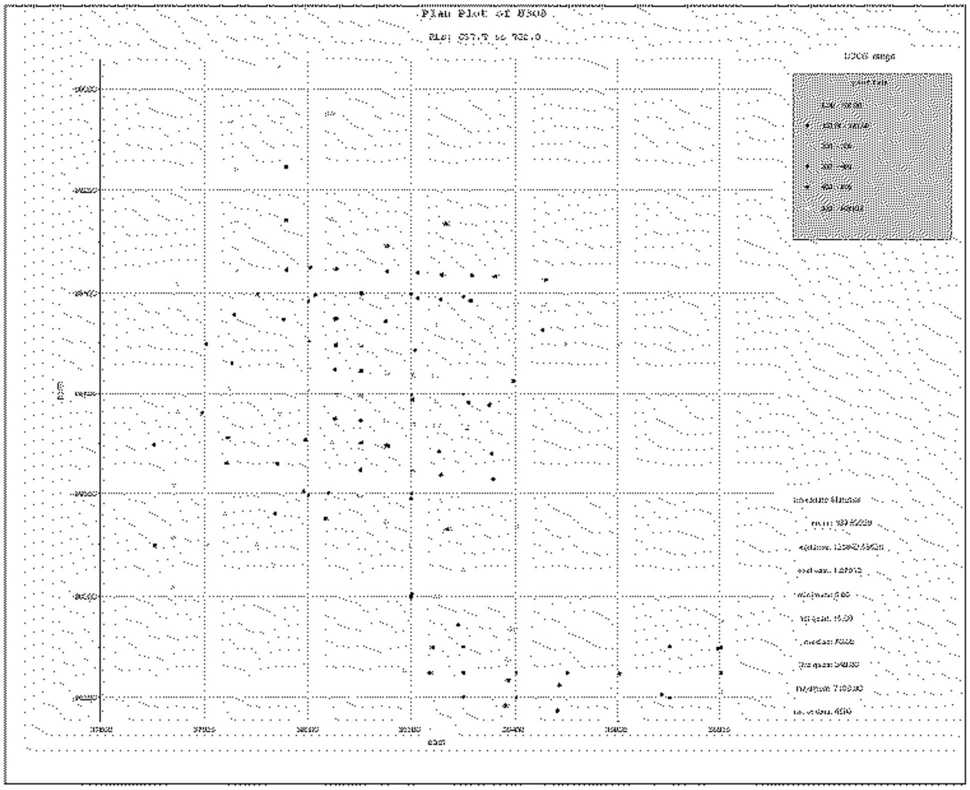


Figure 68: Sample composites in Detail 3

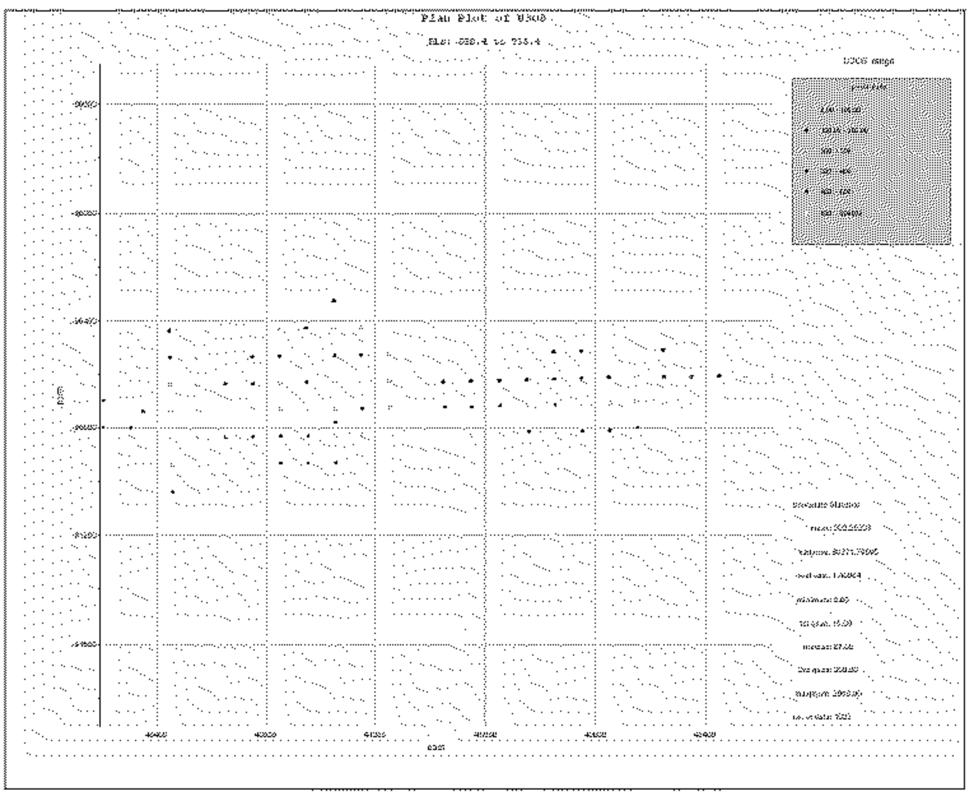


Figure 69: Sample composites in Detail 4

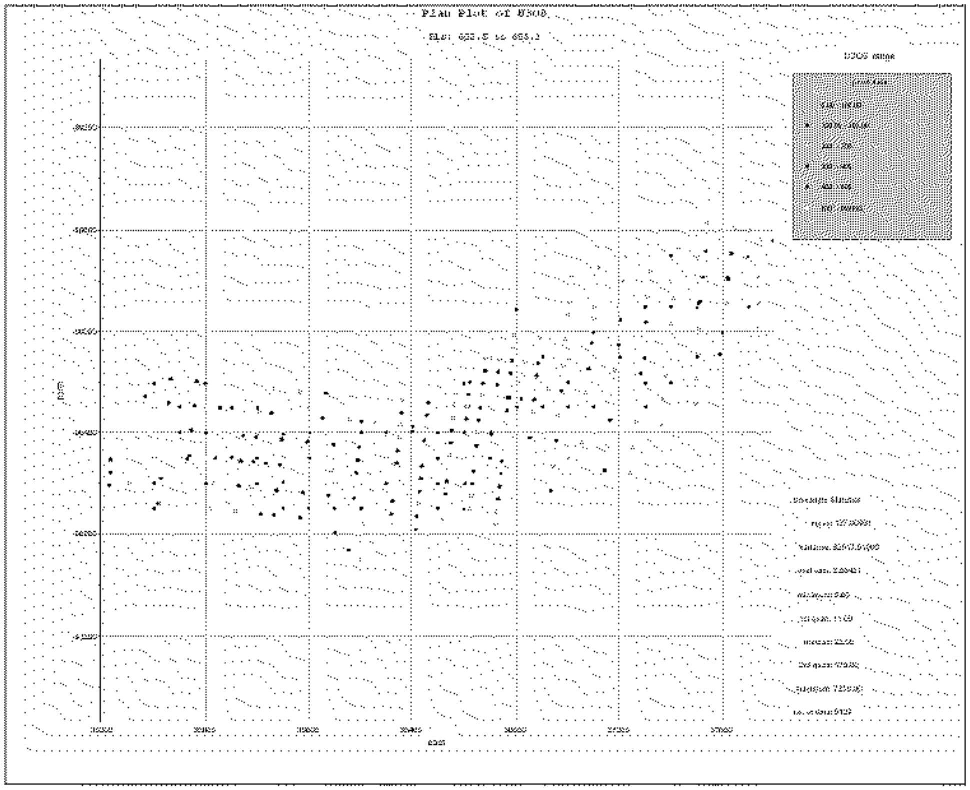


Figure 70: Sample composites in Detail 5

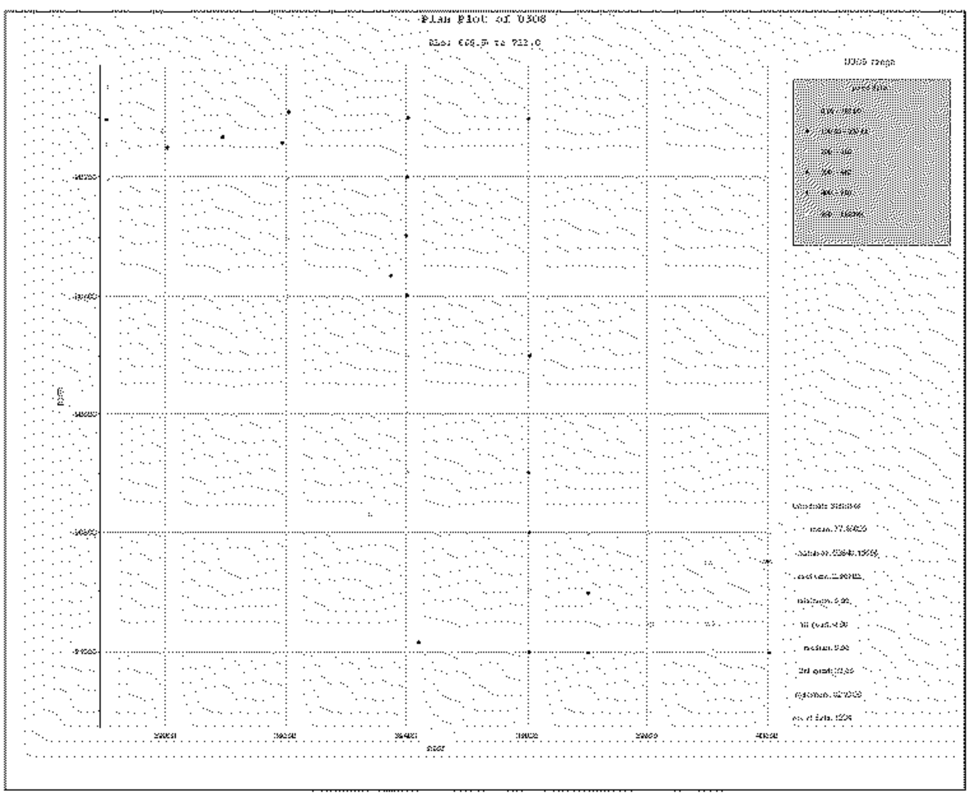


Figure 71: Sample composites in Detail 6

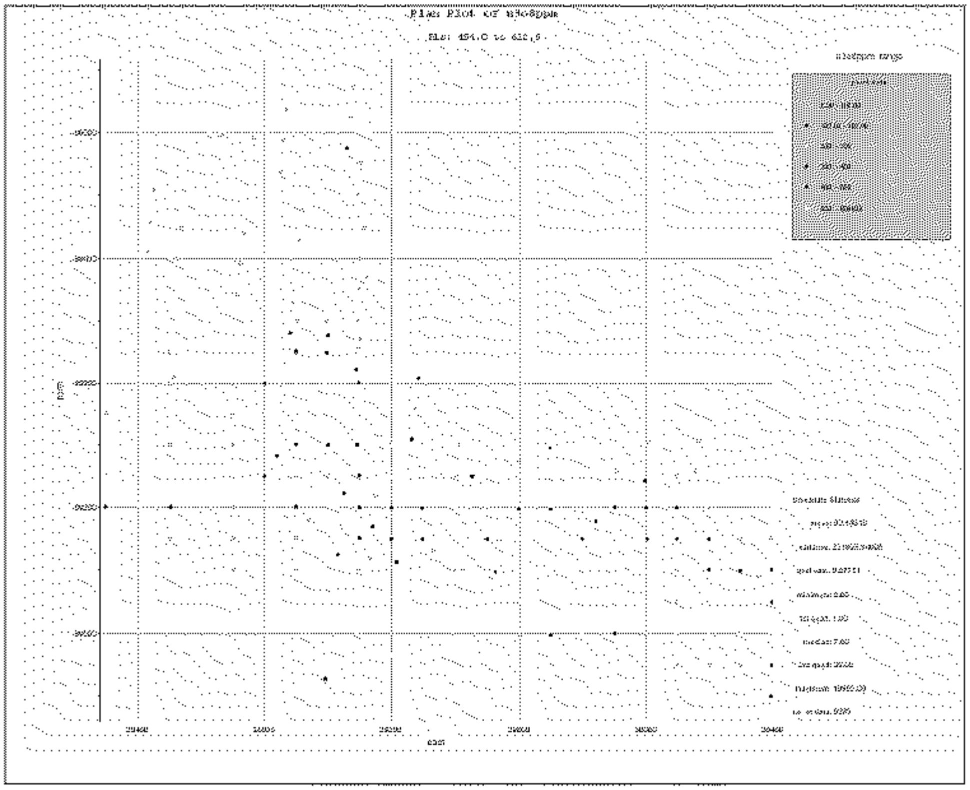


Figure 72: Sample composites in Detail 7

17.5 Domaining

Data in each of the Details were selected and samples from below the basement contact discarded. Each area was examined and, in Details 2 and 5, data were allocated to primary domains to separate areas of different directional trends or different general tenor of mineralisation. Figure 73 and Figure 74 show data in each of the affected Details, coloured by primary domain code.

To avoid under-representing the potential tonnage of mineralisation beneath the water table it was considered prudent to allow wet samples to inform estimates. However, doubts about the reliability of wet percussion drill samples makes it desirable to limit their influence on estimates of resources above the water table. To achieve the dual aims, samples above the water table were allocated to subdomain 1 and those below to subdomain 2 in each of the Details. Figure 75 shows an example cross-section through Detail 1 with composites coloured by secondary domain code.

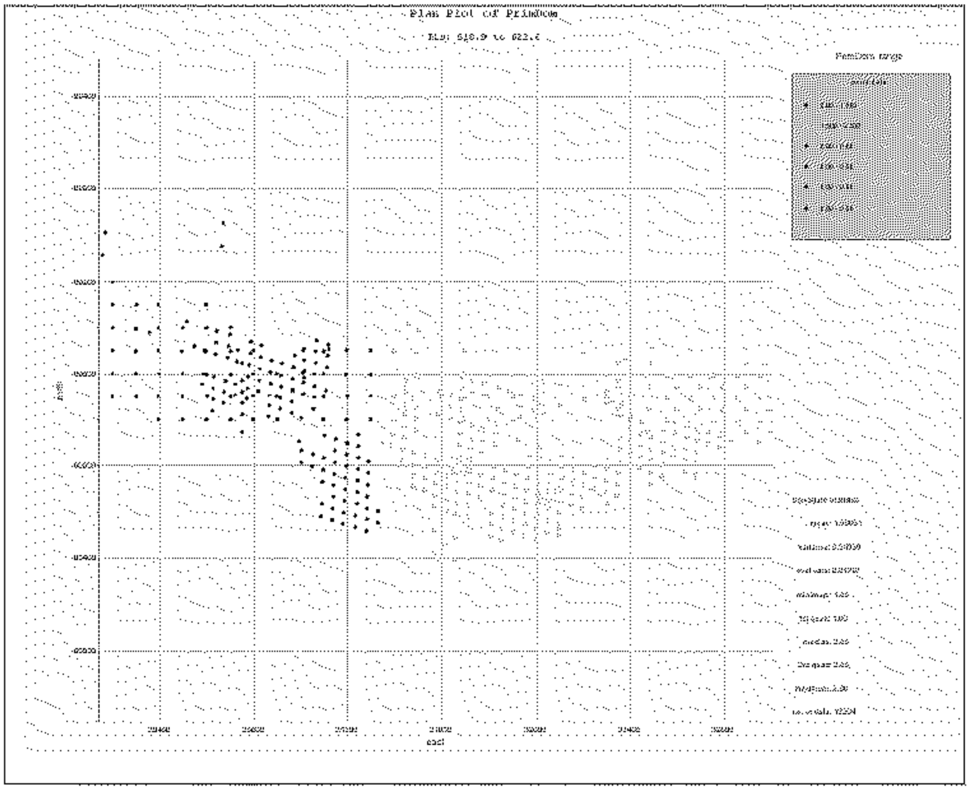


Figure 73: Sample composites in Detail 2 coloured by primary domain code

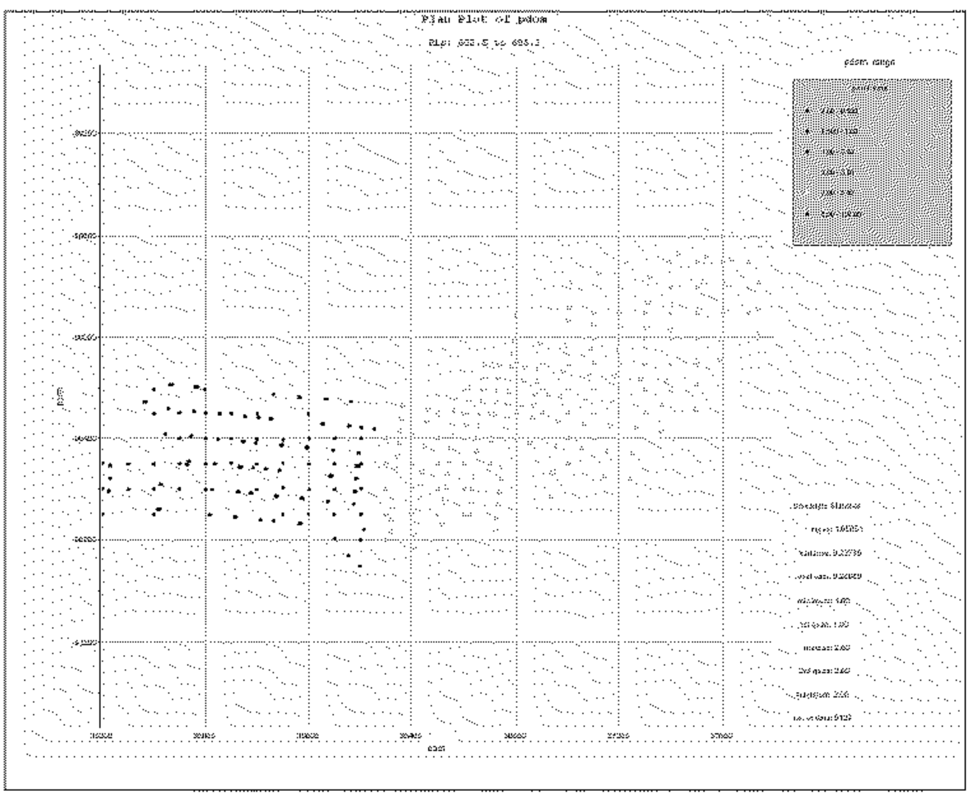


Figure 74: Sample composites in Detail 5 coloured by primary domain code

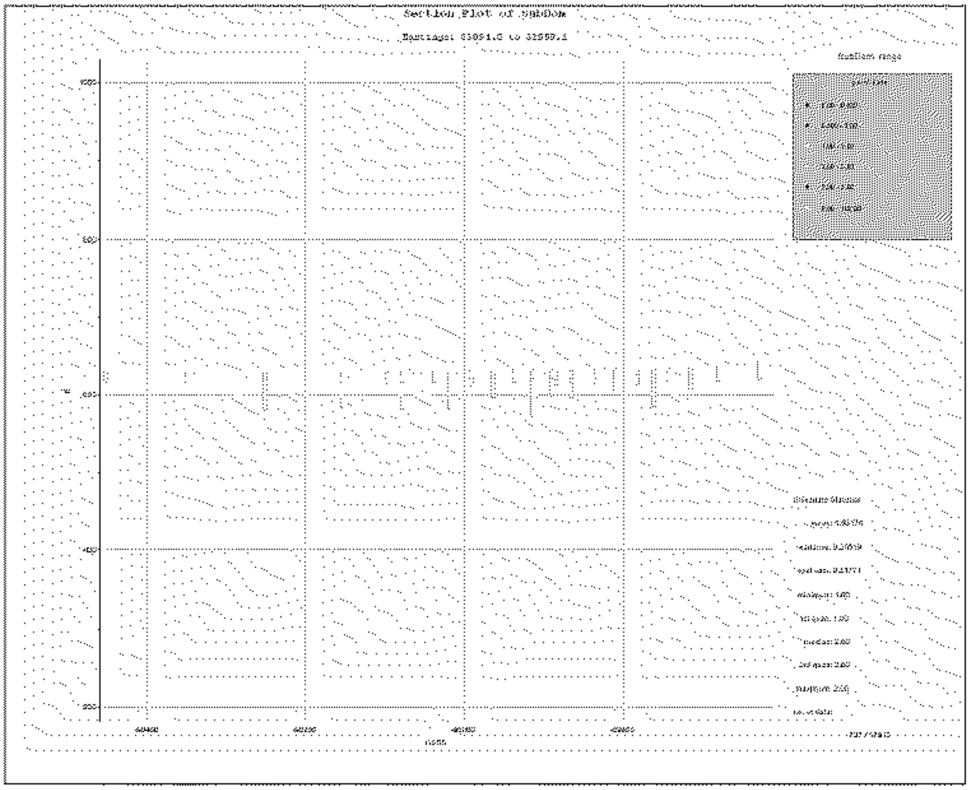


Figure 75: Cross-section through Detail 1, composites coloured by secondary domain code

17.6 Univariate Statistics

Figure 76 to Figure 95 show histograms of U_3O_8 grades in each of the Details. In some instances, data representing a Detail include samples from the first one or two drill traverses in adjacent Details where it appeared useful to allow those data to inform estimates in the Detail. The number of samples in the Detail 1 data subset, for example, portrayed in histograms may not equal the count that derives from sub-setting of Detail 1 data from the total set of composited data.

Sample composites are subdivided into those above the water table and those below and in Details 1, 2 and 5 the data are also subdivided by primary domains. Also, in Detail 1, histograms and summary statistics are shown for data sets including and excluding samples from close-spaced drilling in the mega-trench and trial pit areas. The trial mining areas are located in an area of high-grade mineralisation and their inclusion in data that inform a global average grade, or conditional mean grades, that influence resource estimates will almost certainly lead to over-estimation of resource tonnages and grades.

Each of the areas contains large proportions of samples that were deliberately not assayed and have been allocated grades of 15ppm U_3O_8 (Section 16.1, above). Composites grading less than 20ppm have been excluded from the data shown in the histograms and summary statistics, as have samples from below the basement interface. The histograms thus indicate the distribution of mineralised samples above a low U_3O_8 grade threshold. Proportions of the sample data in each area excluded by this treatment are shown in Table 17.

Detail	Number of composites above basement	Proportion of composites < 20ppm U_3O_8	Proportion of comps allocated below detection limit grades
1	17913	26.6%	9.70%
1, excluding mine data	15144	31.5%	11.5%
2	13204	38.9%	20.8%
3	4479	35.4%	11.7%
4	1837	31.9%	18.7%
5	5900	46.7%	18.0%
6	1171	69.2%	6.0%
7	9383	63.8%	63.4%

Table 17: Proportions of sample composites allocated below detection limit grades

Comparison of the summary statistics in Figure 76 and Figure 78 demonstrates the effect of the close-spaced mine area drilling on the histogram of sample grades in the western portion of Detail 1. The mean grade of samples above the water table falls from 330ppm U_3O_8 to 315ppm when those data are excluded. There is not such a great effect on the population of samples below the water table as the drilling in the original pit area had only limited penetration below the water table.

In both domains 1 and 2 of Detail 1 sample populations from below the water table, many of which derive from radiometric logging of Acclaim and Paladin drill holes, show many more low-grade samples but also a slightly higher-proportion of high-grades. The net effect is a similar average grade of samples below the water table and a population with higher variance and higher coefficient of variation.

In Detail 2, domain 2 the trend is similar but not as pronounced. The mean grade of samples above the water table in that area is skewed upward by a maximum sample grade of 15789 ppm.

In Detail 3, samples from above and below the water table show more similar variances and mean grades. The histogram of grades in samples from above the water table is notably less skewed than those in Details 1 and 2.

Samples in Details 4 and 5 do not show any trend to higher or more erratic grades beneath the water table. There are an unusually low proportion of low-grade samples from above the water table in domain 1 of Detail 5. There is no obvious explanation for this feature; the data are not obviously clustered in high-grade mineralisation. It is possible that some higher threshold, perhaps about 100ppm U_3O_8, was applied in deciding which drill samples were to be sent for XRF assay within the Gencor sampling although this seems unlikely.

There are only 102 sample composites flagged as lying below the water table in Detail 6. Of samples from above the water table, only 259 grade over 20ppm U_3O_8. Those samples describe a positively skewed population with a significant proportion of high-grade samples affecting the mean, variance and coefficient of variation statistics.

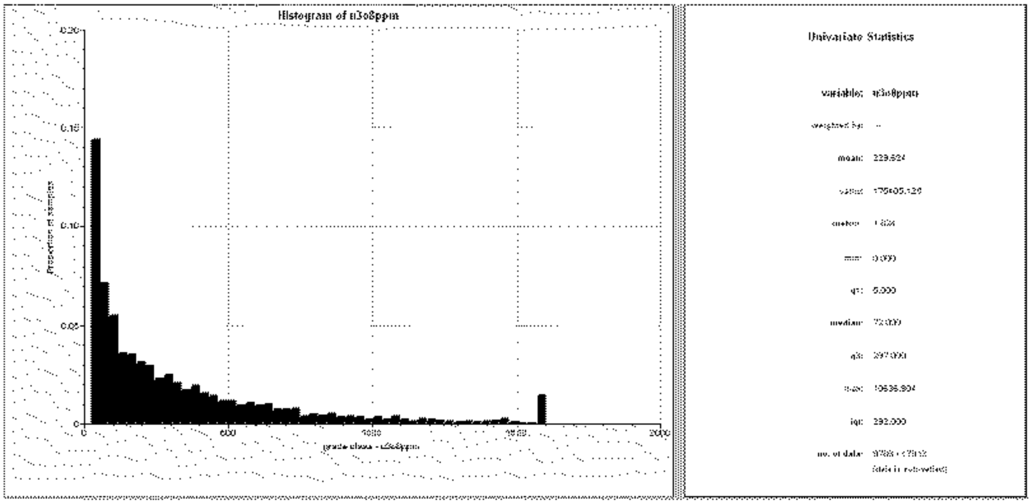


Figure 76: Histogram of grades in Detail 1, domain 1, above the water table, including clustered data

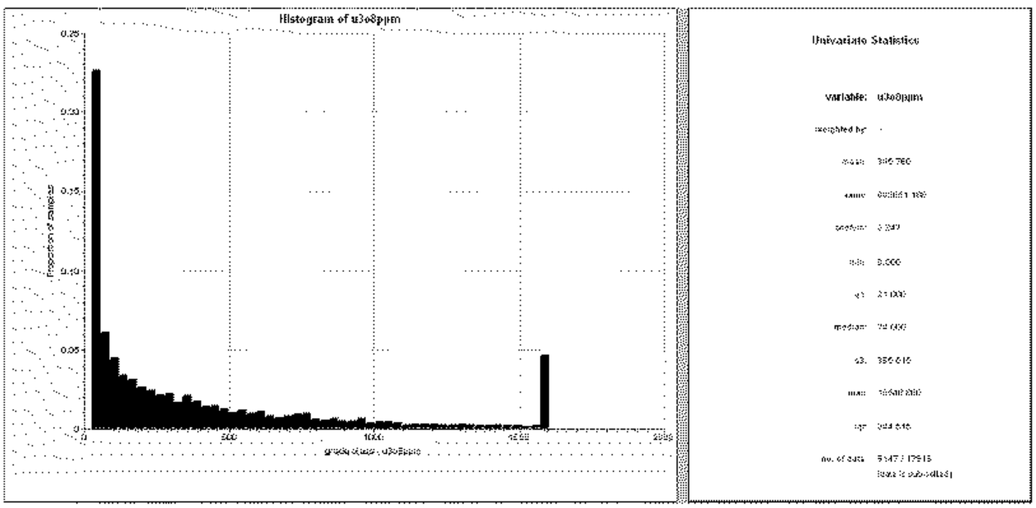


Figure 77: Histogram of grades in Detail 1, domain 1, below the water table, including clustered data

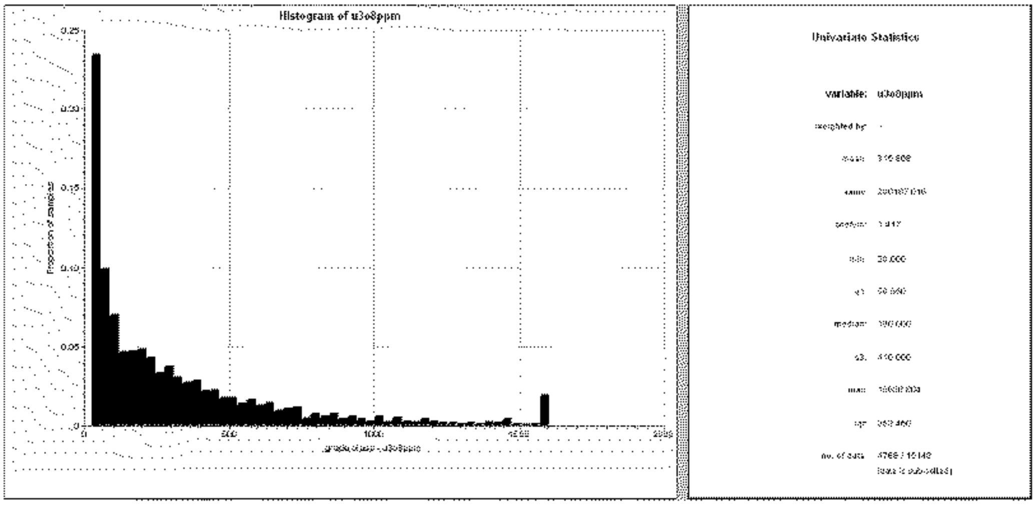


Figure 78: Histogram of grades in Detail 1, above the water table, excluding clustered data

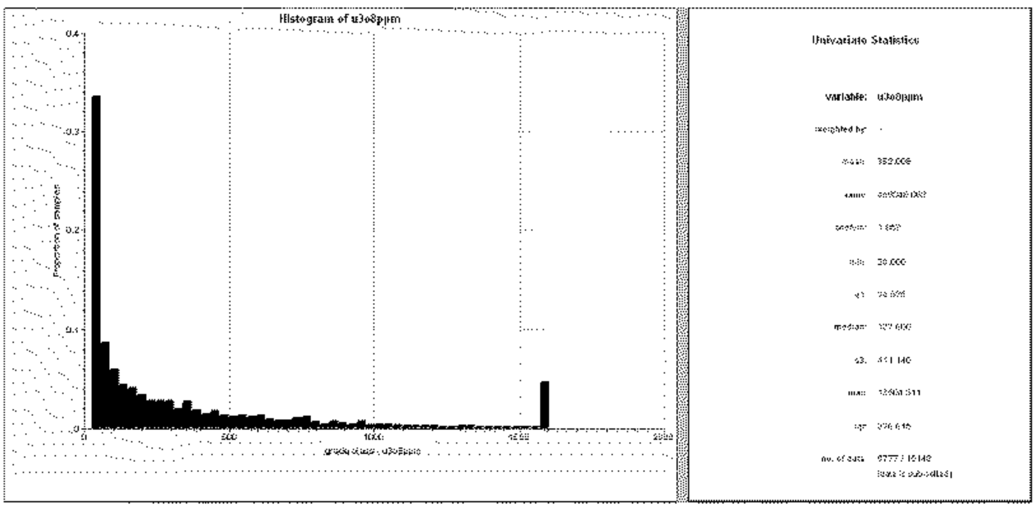


Figure 79: Histogram of grades in Detail 1, below the water table, excluding clustered data

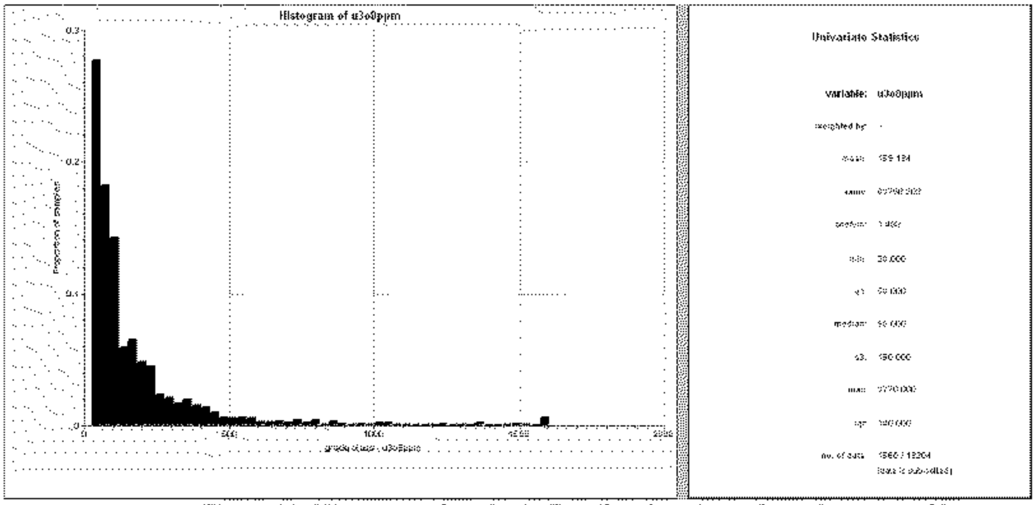


Figure 80: Histogram of grades in Detail 2, domain 1, above the water table

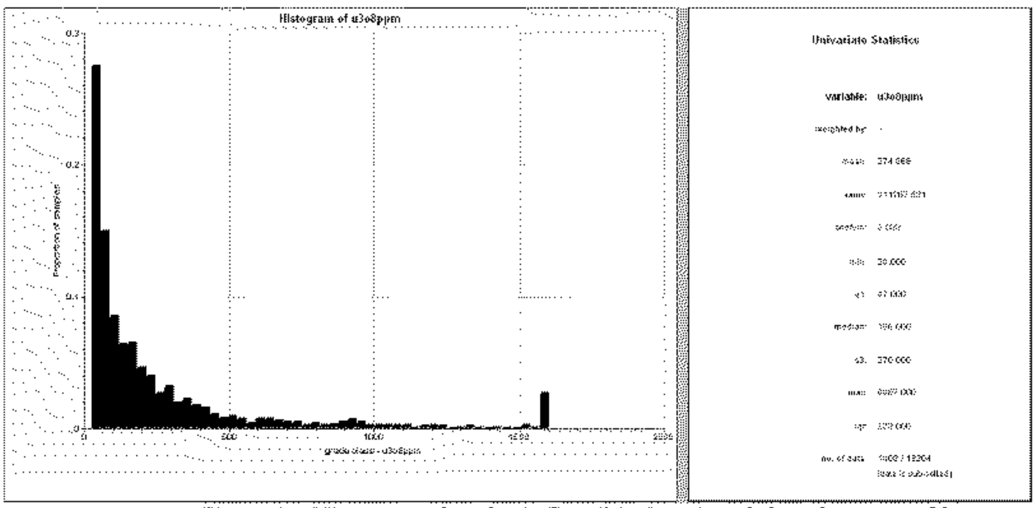


Figure 81: Histogram of grades in Detail 2, domain 1, below the water table

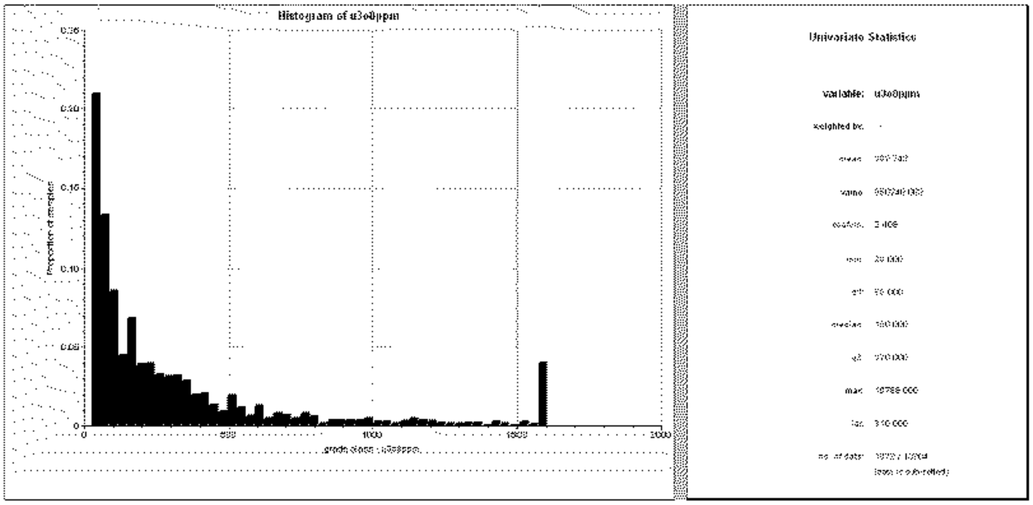


Figure 82: Histogram of grades in Detail 2, domain 2, above the water table

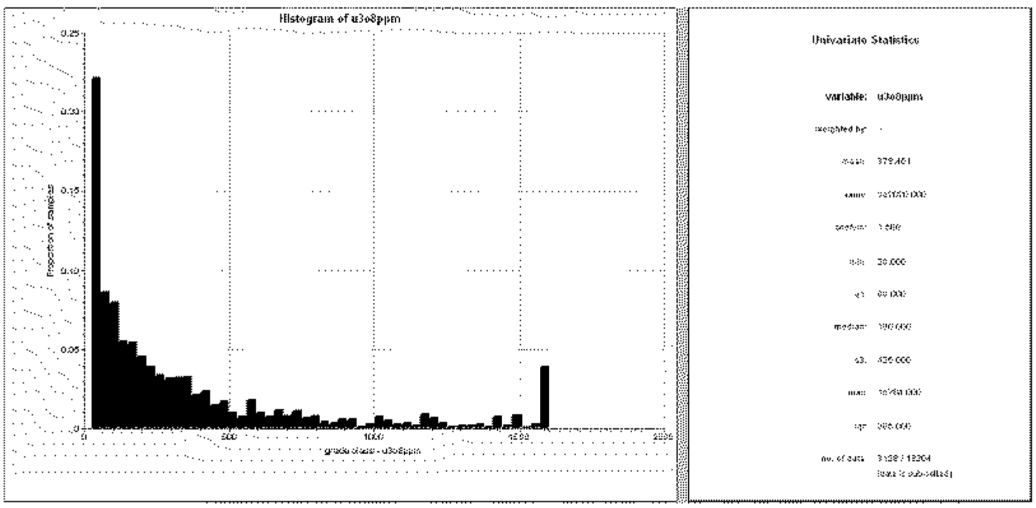


Figure 83: Histogram of grades in Detail 2, domain 2, below the water table

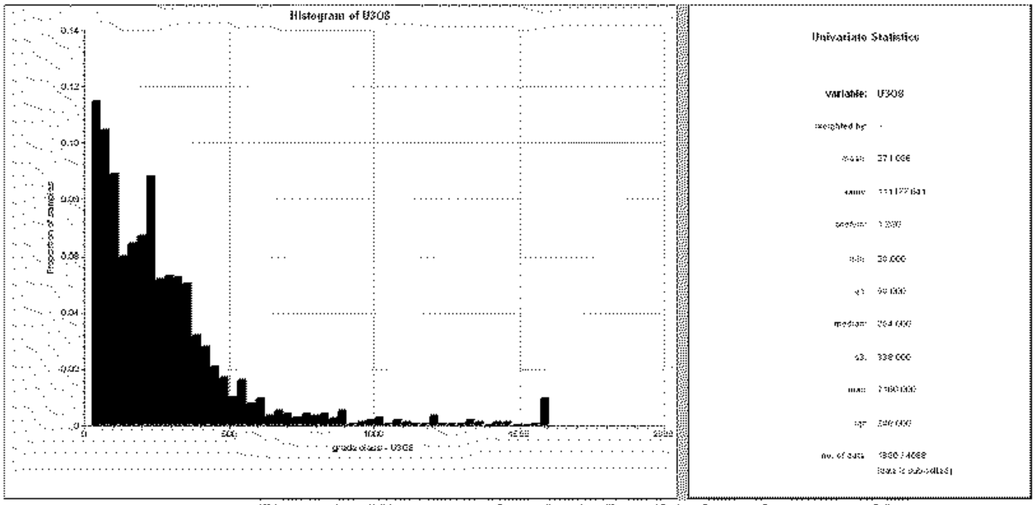


Figure 84: Histogram of grades in Detail 3 above the water table

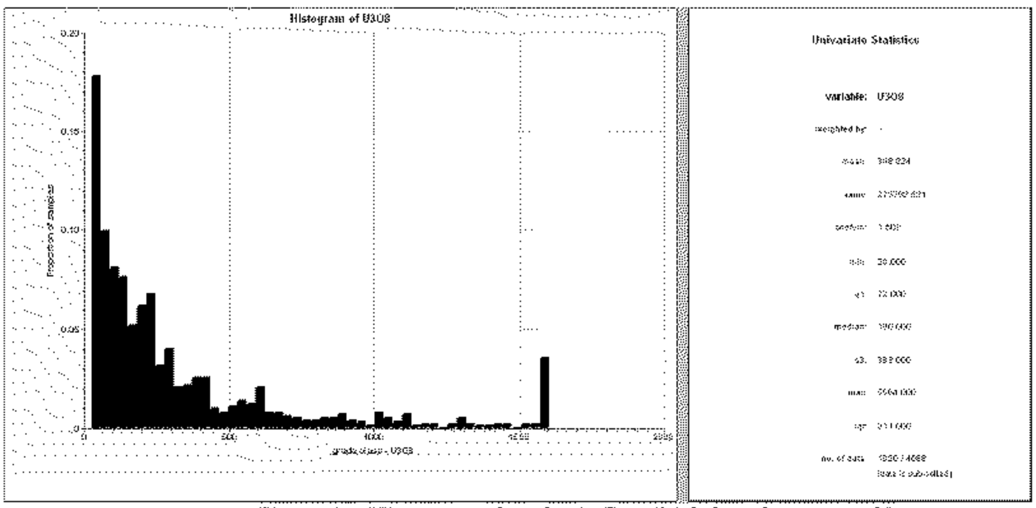


Figure 85: Histogram of grades in Detail 3 below the water table

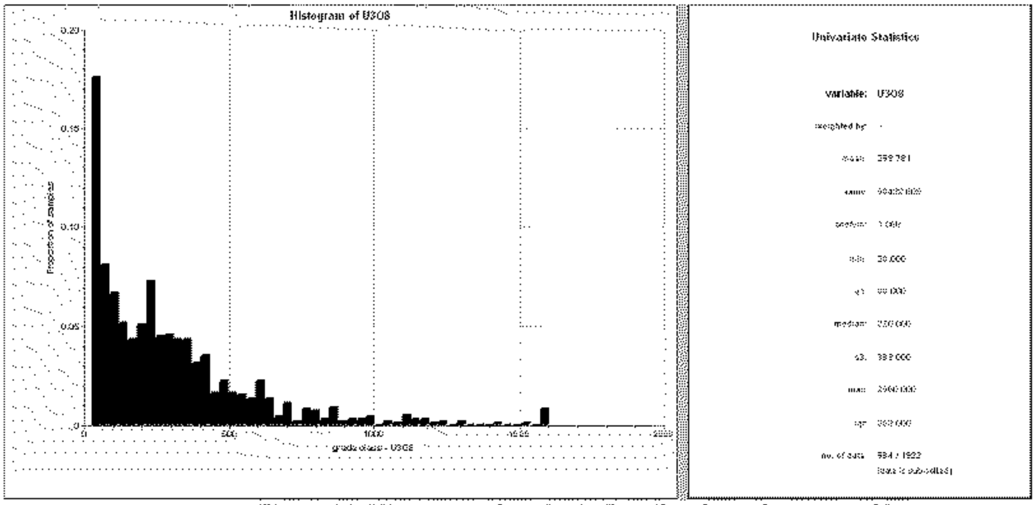


Figure 86: Histogram of grades in Detail 4 above the water table

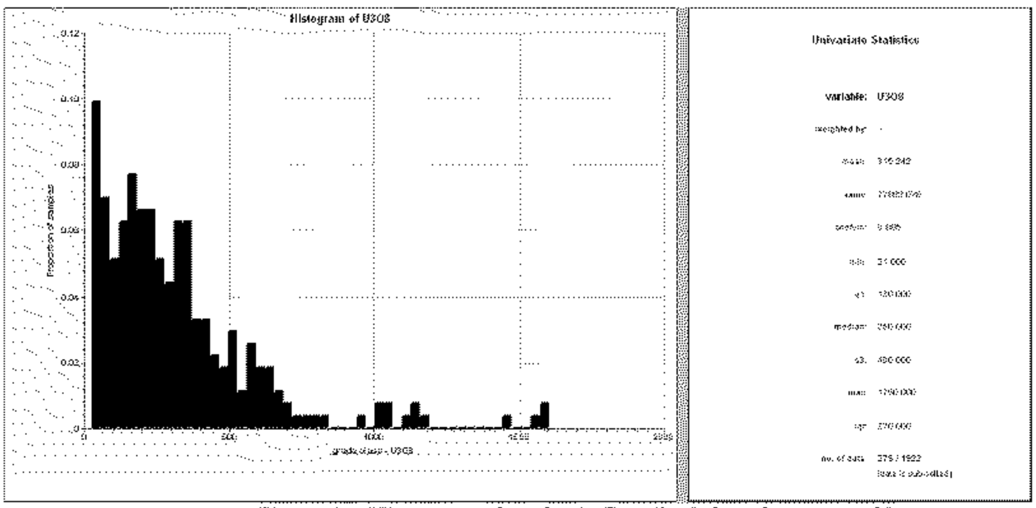


Figure 87: Histogram of grades in Detail 4 below the water table

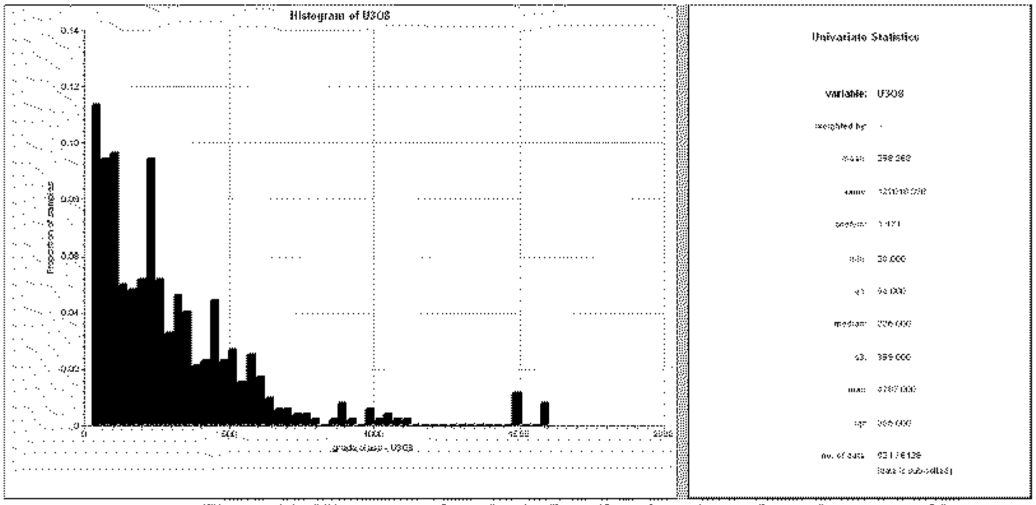


Figure 88: Histogram of grades in Detail 5, domain 1, above the water table

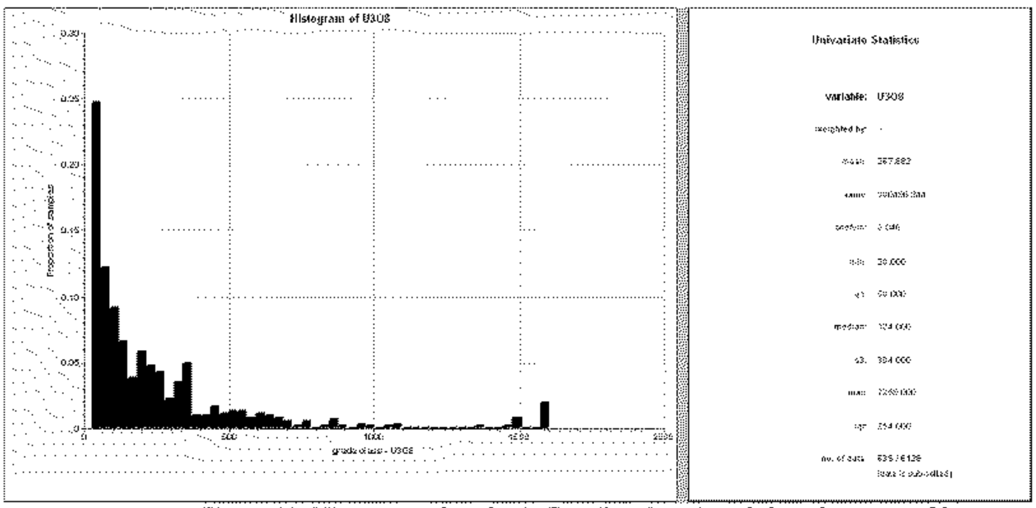


Figure 89: Histogram of grades in Detail 5, domain 1, below the water table

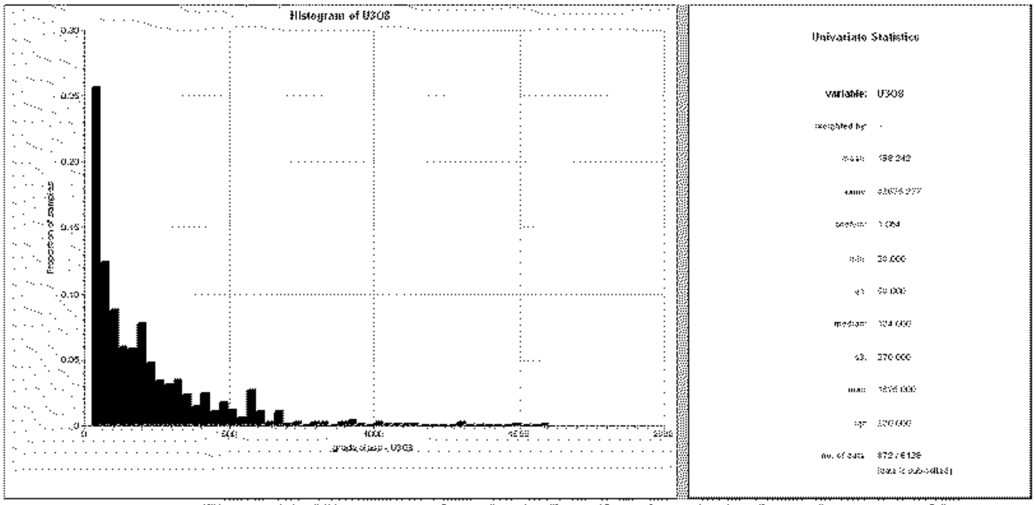


Figure 90: Histogram of grades in Detail 5, domain 2, above the water table

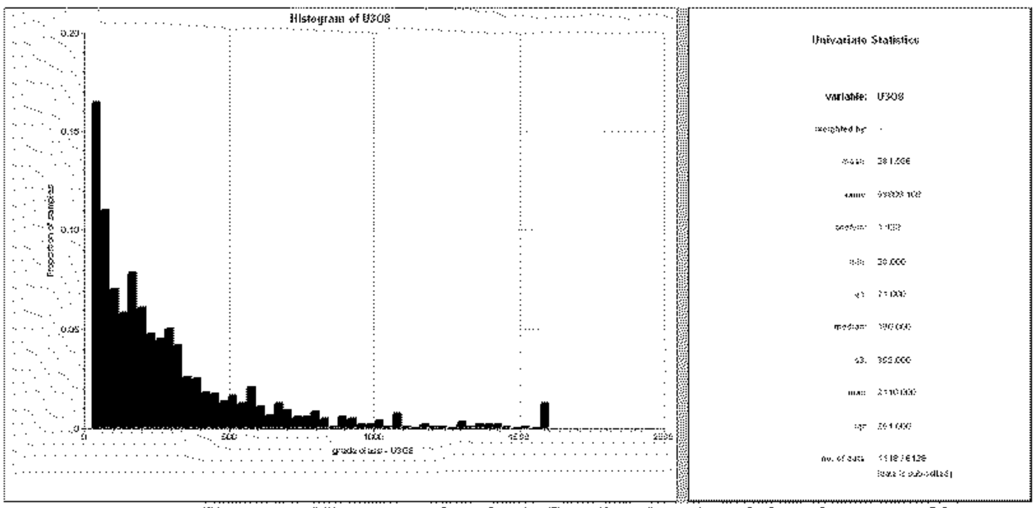


Figure 91: Histogram of grades in Detail 5, domain 2, below the water table

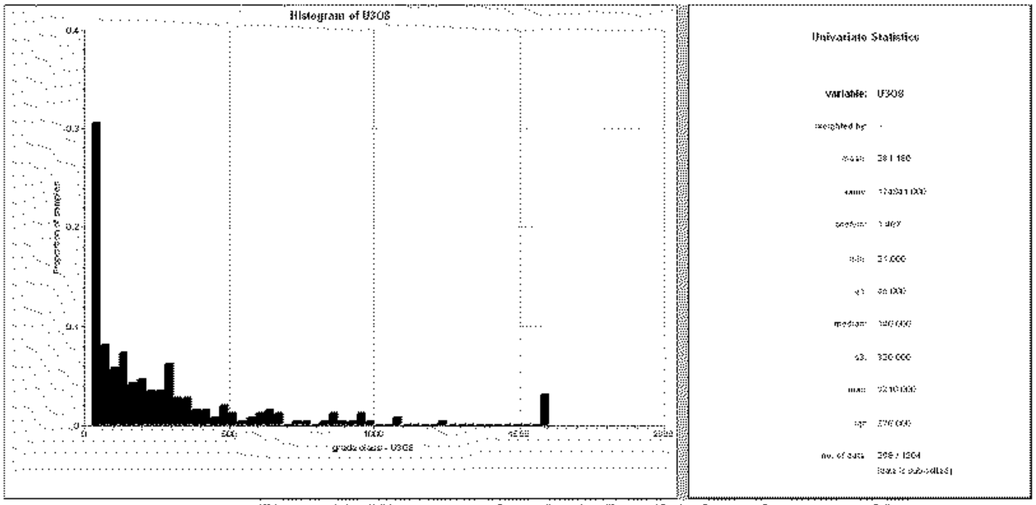


Figure 92: Histogram of grades in Detail 6 above the water table

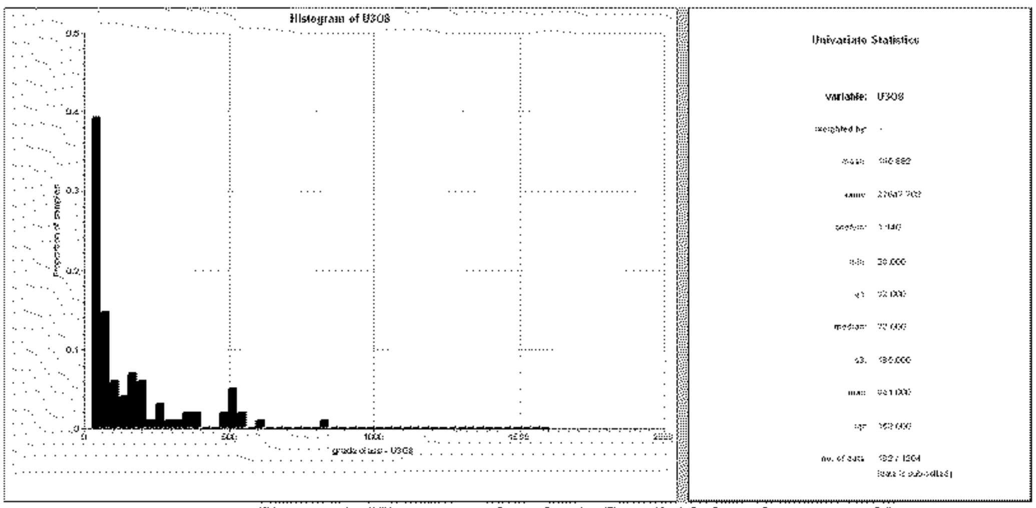


Figure 93: Histogram of grades in Detail 6 below the water table

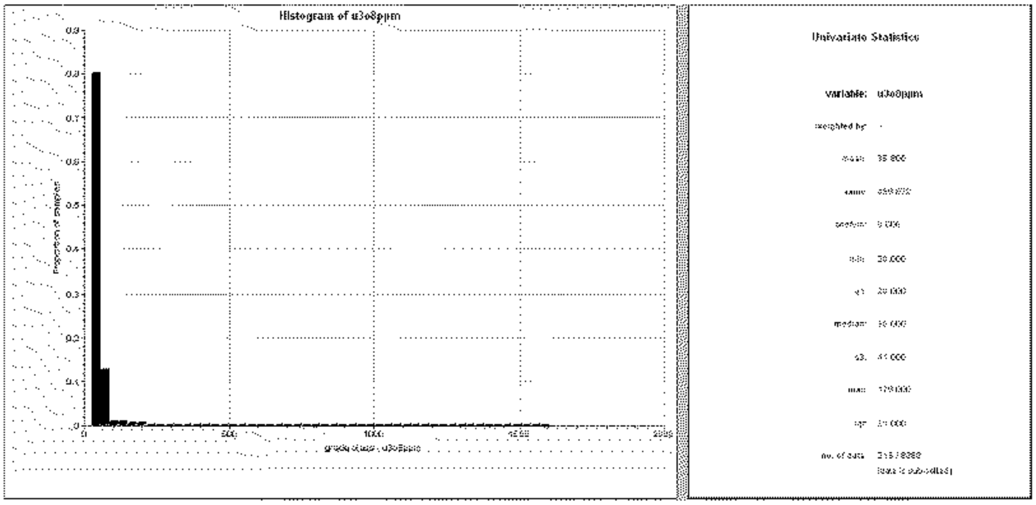


Figure 94: Histogram of grades in Detail 7 above the water table

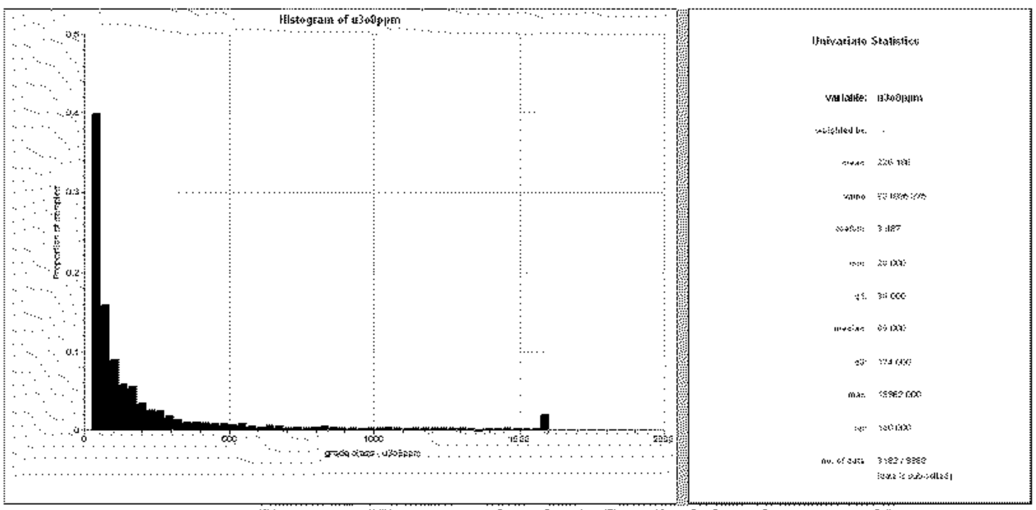


Figure 95: Histogram of grades in Detail 7 below the water table

17.7 Variograms of U_3O_8 Grades

Considering the uncertainty about the reliability of percussion drill samples from beneath the water table, and the small number of those data in some Details, calculation of experimental U_3O_8 and indicator variograms was based solely on samples from above the water table. An exception was Detail 7 where all significant mineralisation lies beneath the water table. The models fitted to those variograms were then applied to estimation of resources in panels both above and below the water table, i.e., in both subdomains.

Figure 96 to Figure 103 show variogram maps of U_3O_8 grades in each of the resource areas, calculated using only samples above the water table. No useful variogram map could be calculated for Detail 4 because of the short north-south drill coverage.

In all instances the directional trends evident in the variogram maps are evident to some extent in plan views of the sample data. They normally conform to the expected direction of groundwater flow given the geometry of the palaeovalley at each location. In some areas the maps are affected by the extent or pattern of drill coverage. Ratios of anisotropy are mainly less than 2. The variogram maps, in conjunction with plots of the sample data, were used to guide variogram model rotations.

For each of Details, and for each domain in Details 1, 2 and 5, experimental variograms of U_3O_8 grades were calculated and modelled (Figure 104 to Figure 130). The azimuths referred to in the titles of the diagrams conform to the trigonometric convention in which azimuth zero is grid east and azimuth 90 is grid north. The variograms in Detail 1, domain 1 were calculated after exclusion of samples from the close-spaced drilling in the trial mining area but the spatial continuity of U_3O_8 grades in those data were examined separately to confirm short-scale structures. The x and y directions of the two sets of experimental variogroms could not be exactly matched because of drill hole spacings. Figure 131 to Figure 133 show the experimental variograms deriving from the close-spaced drilling with the variogram model from the broader-spaced data superimposed. The variogram in the vertical direction confirms the choice of model nugget and the variograms in plan view directions generally confirm the model parameters. The plan view experimental variograms are affected by the change in drill hole spacing from 6m x 6m in the mega-trench to 12.5m x 12.5m in the trial pit. It should also be recalled that the close-spaced drilling covers an area of high-grade mineralisation where U_3O_8 grades might be expected to be more erratic.

As expected, variogram model ranges in the vertical direction are short. The majority of variograms display reasonable structure, with anisotropies reflecting those observed in the variogram maps. In Detail 7 the sill is reached at less than 100 metres separation, the drill holes spacing.

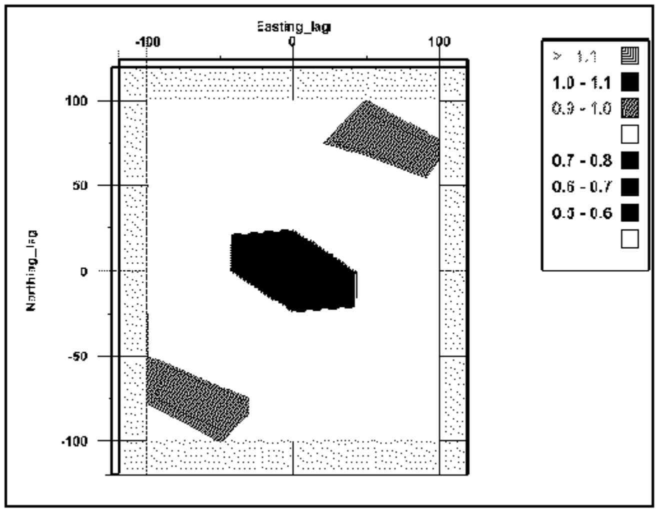


Figure 96: Plan view variogram map of Detail 1, domain 1

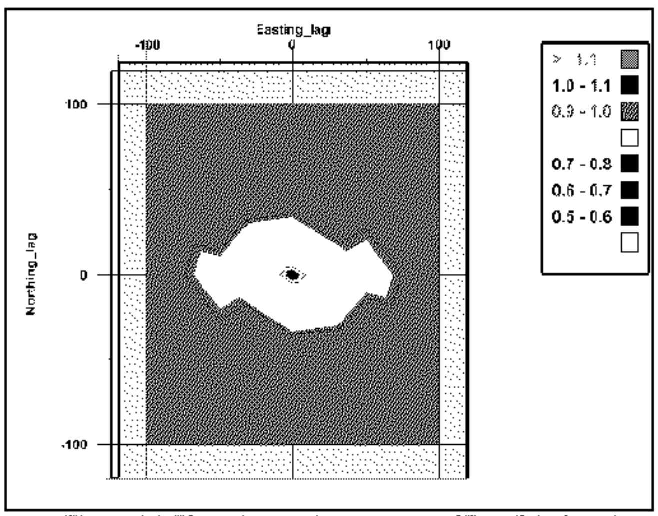


Figure 97: Plan view variogram map of Detail 2, domain 1

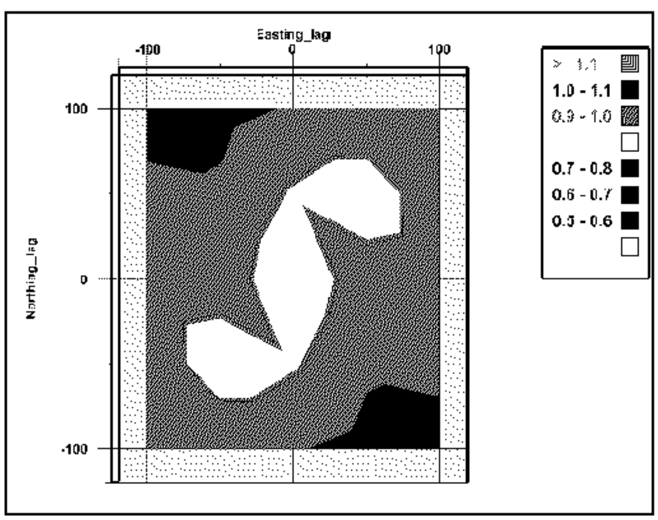


Figure 98: Plan view variogram map of Detail 2, domain 2

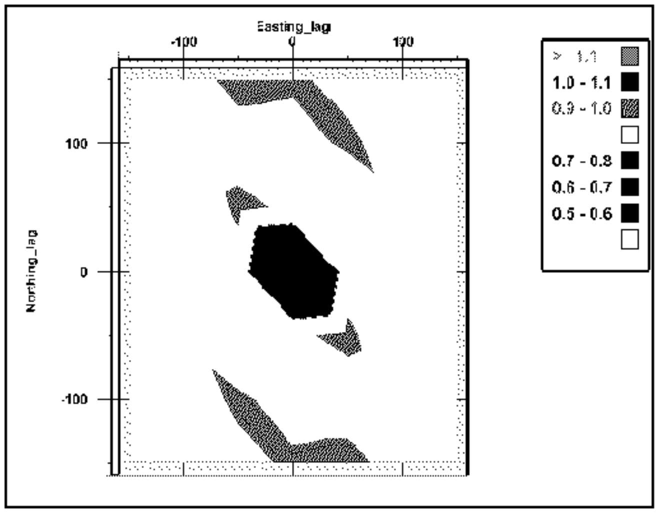


Figure 99: Plan view variogram map of Detail 3

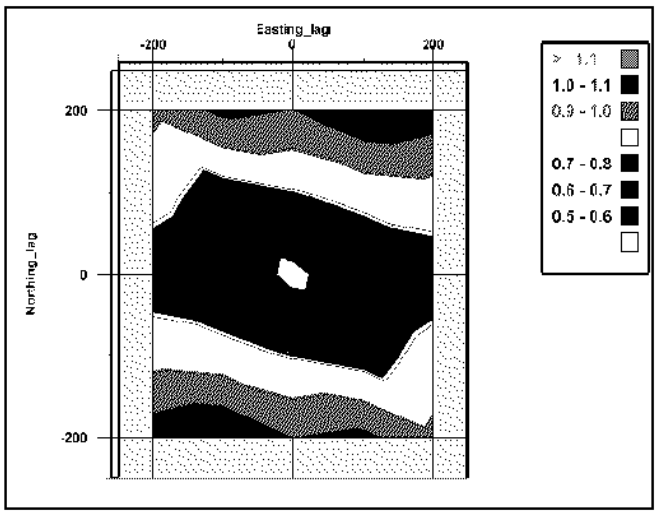


Figure 100: Plan view variogram map of Detail 5, domain 1

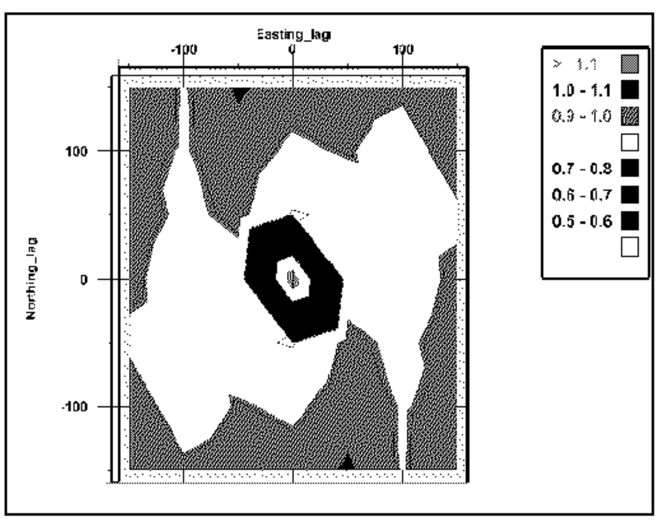


Figure 101: Plan view variogram map of Detail 5, domain 2

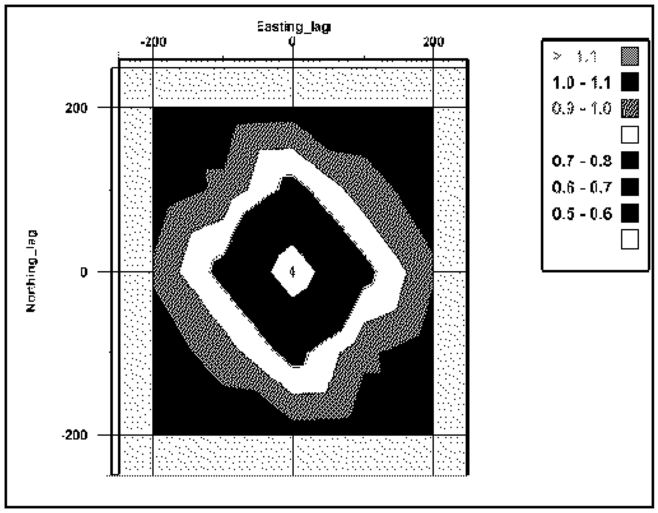


Figure 102: Plan view variogram map of Detail 6

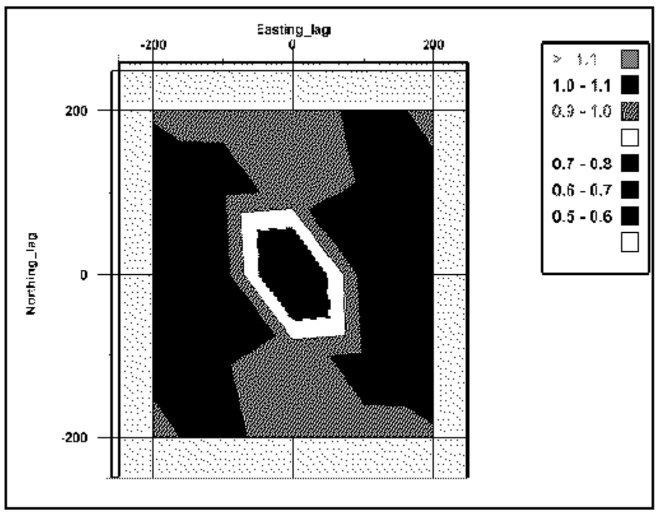


Figure 103: Plan view variogram map of Detail 7

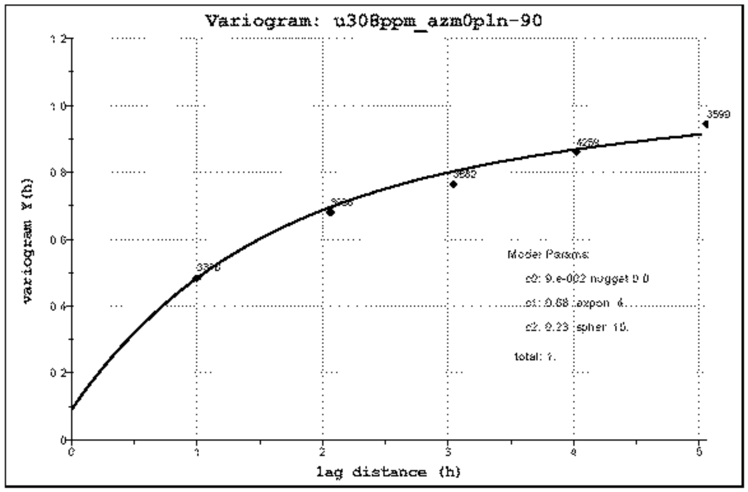


Figure 104: Down-hole variogram, Detail 1 domain 1

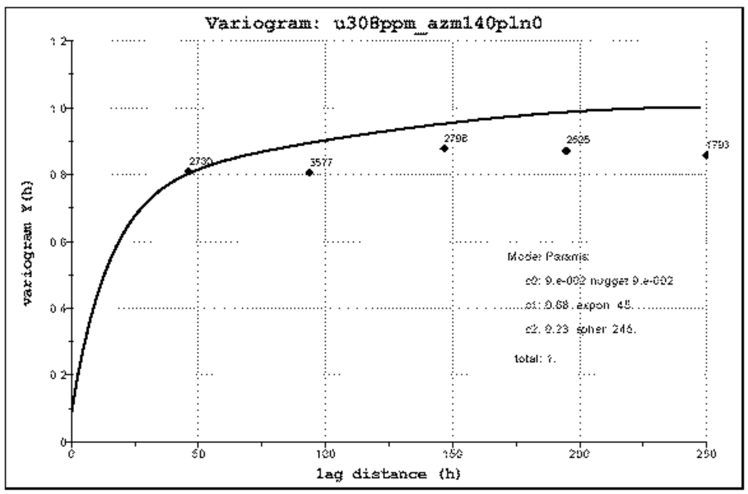


Figure 105: Along-strike variogram, Detail 1 domain 1

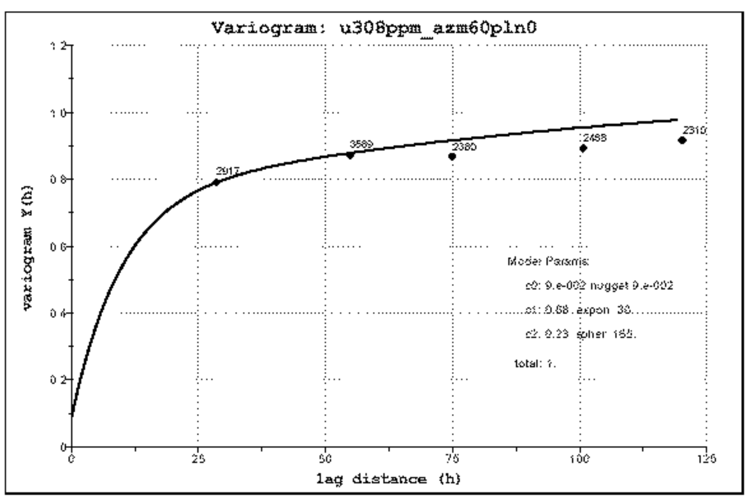


Figure 106: Across-strike variogram, Detail 1 domain 1

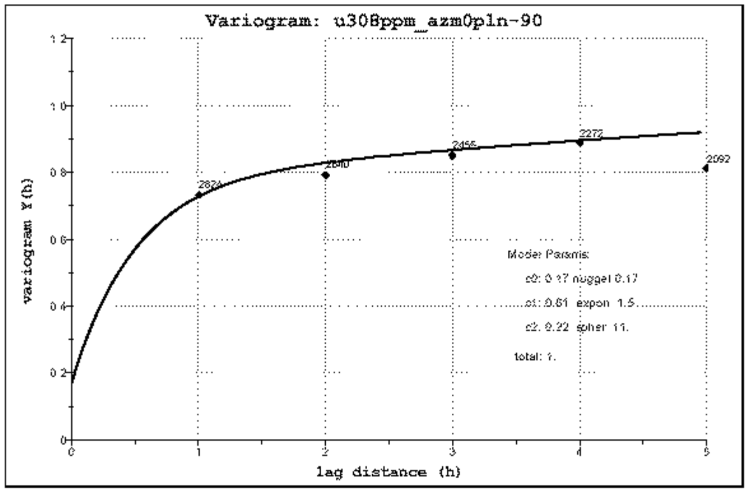


Figure 107: Down-hole variogram, Detail 2 domain 1

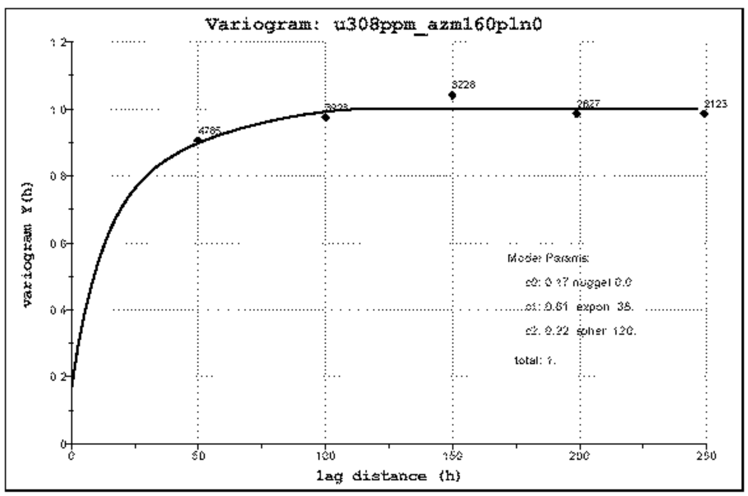


Figure 108: Along-strike variogram, Detail 2 domain 1

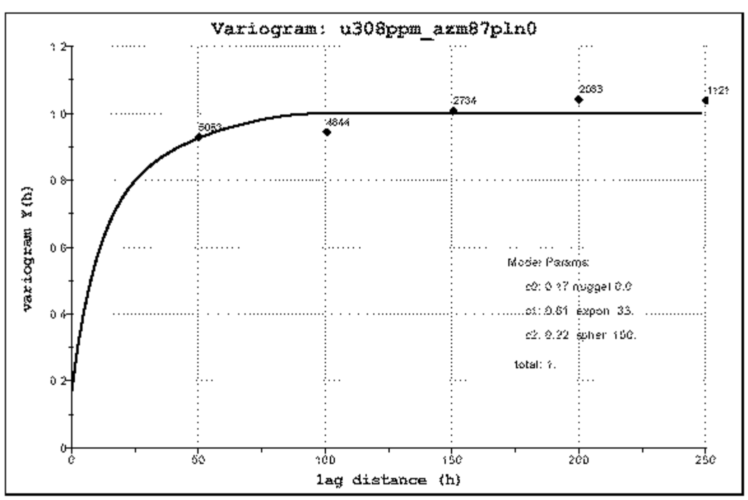


Figure 109: Across-strike variogram, Detail 2 domain 1

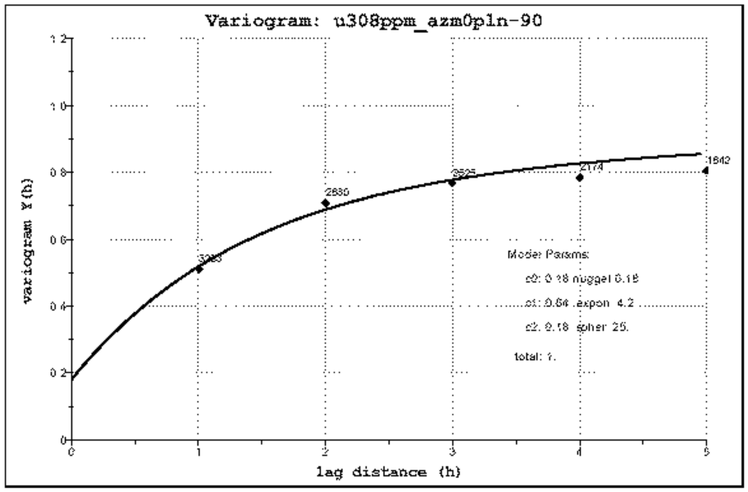


Figure 110: Down-hole variogram, Detail 2 domain 2

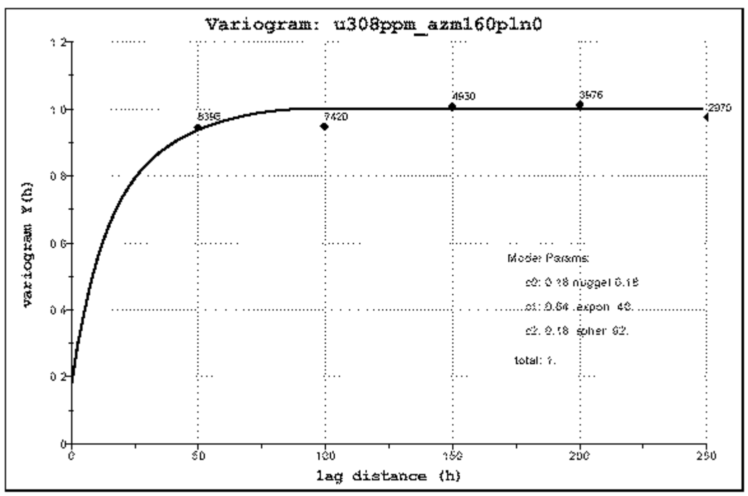


Figure 111: Along-strike variogram, Detail 2 domain 2

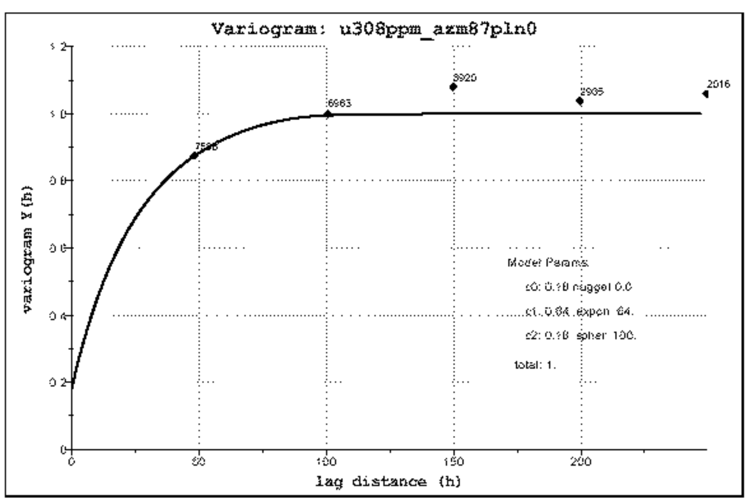


Figure 112: Across-strike variogram, Detail 2 domain 2

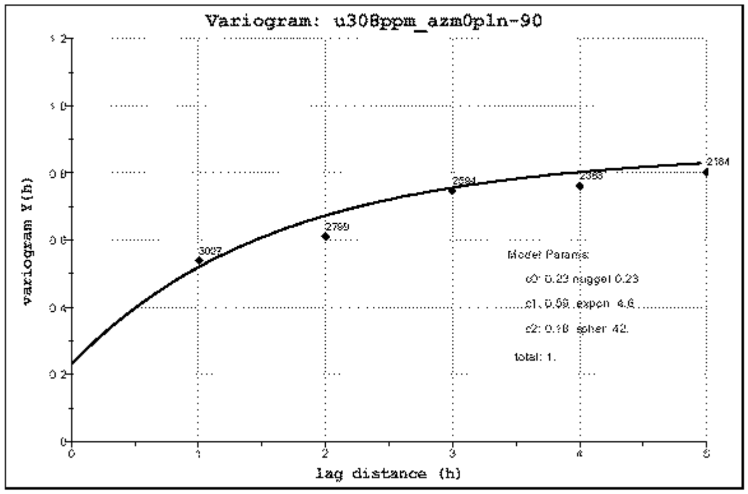


Figure 113: Down-hole variogram, Detail 3

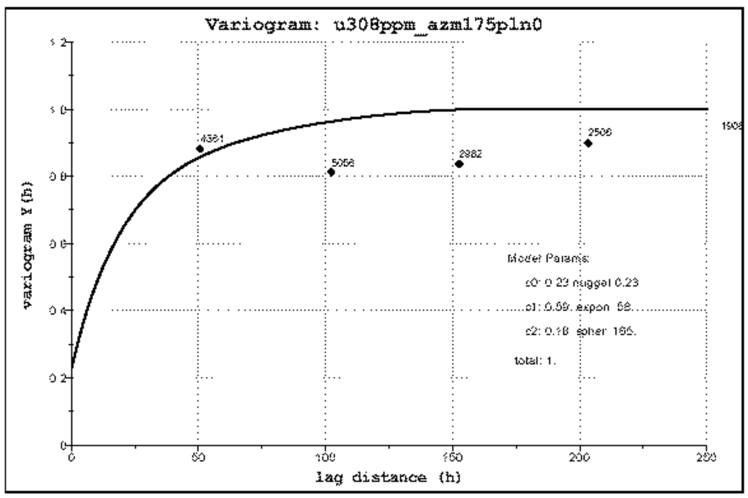


Figure 114: Along-strike variogram, Detail 3

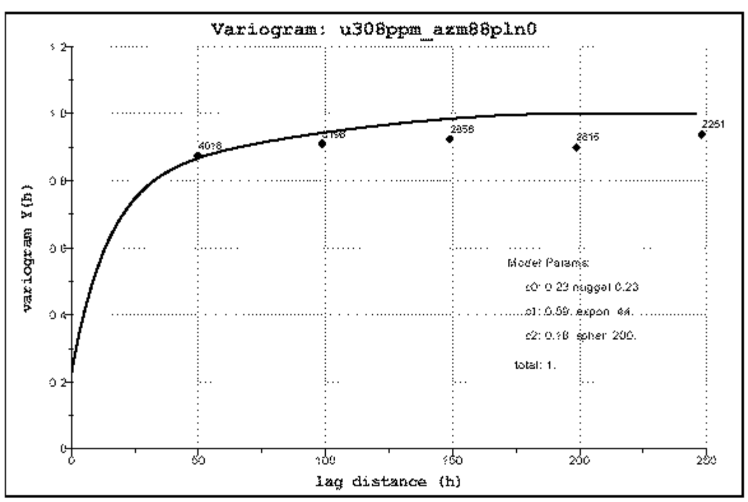


Figure 115: Across-strike variogram, Detail 3

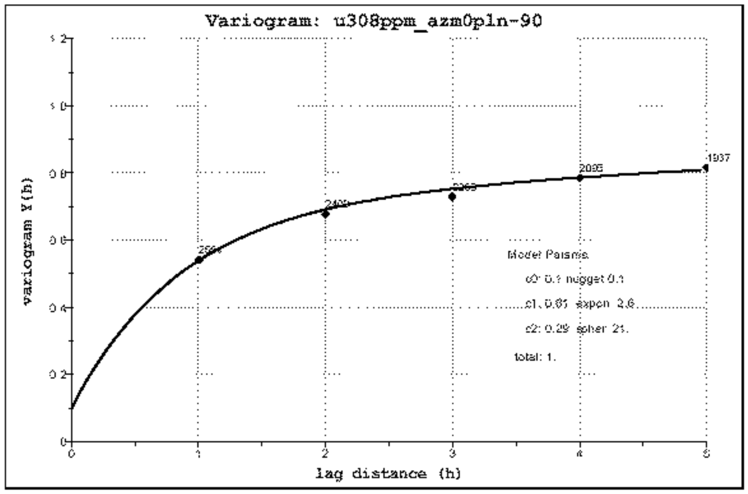


Figure 116: Down-hole variogram, Detail 4

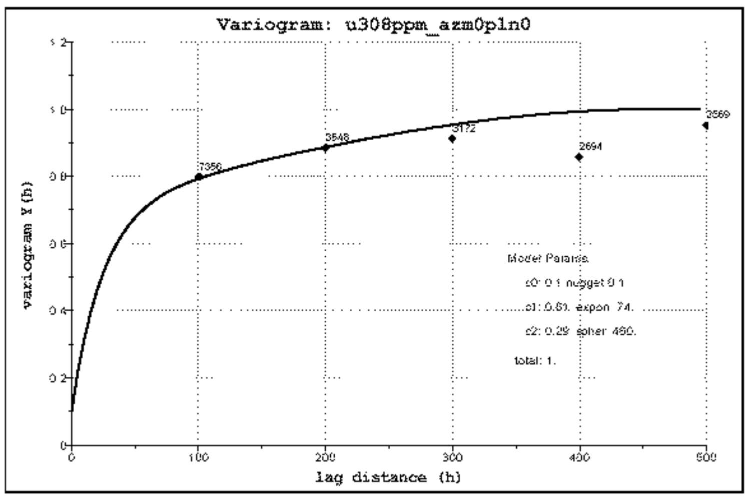


Figure 117: Along-strike variogram, Detail 4

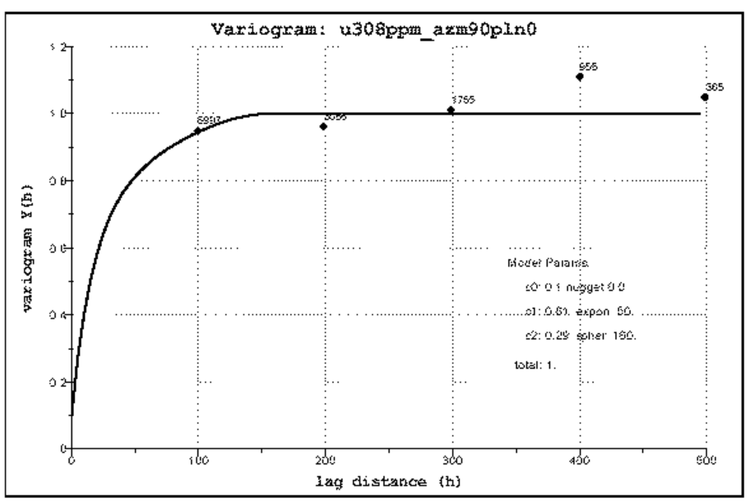


Figure 118: Across-strike variogram, Detail 4

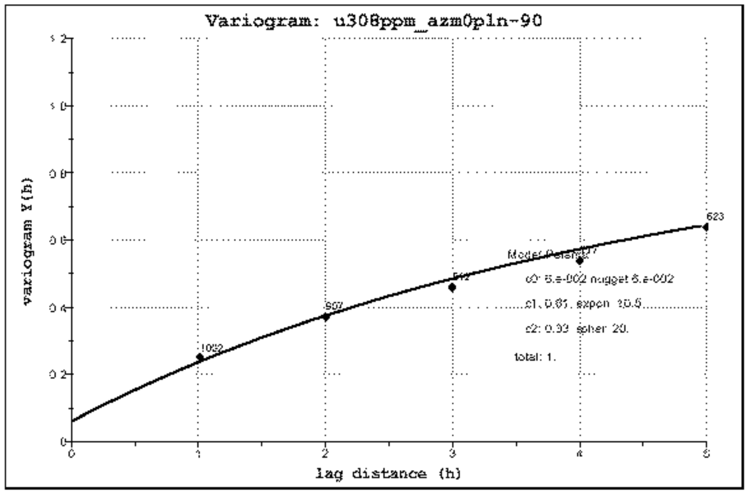


Figure 119: Down-hole variogram, Detail 5 domain 1

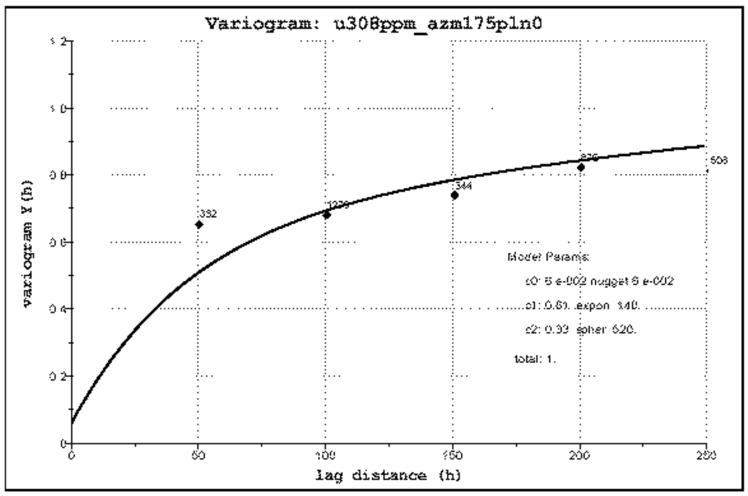


Figure 120: Along-strike variogram, Detail 5 domain 1

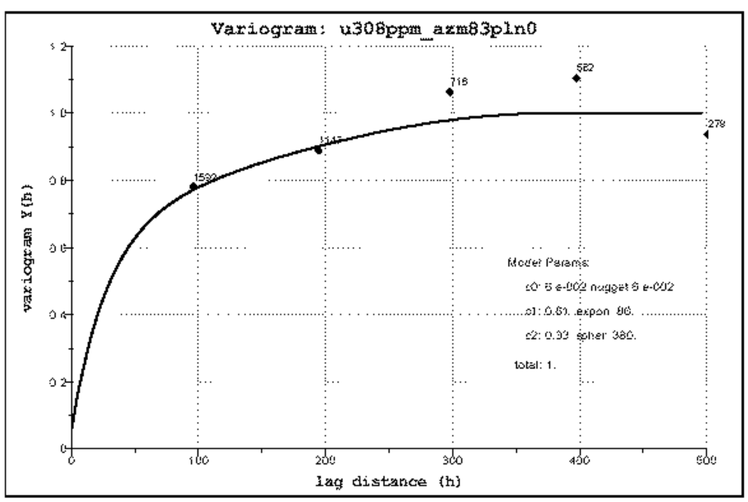


Figure 121: Across-strike variogram, Detail 5 domain 1

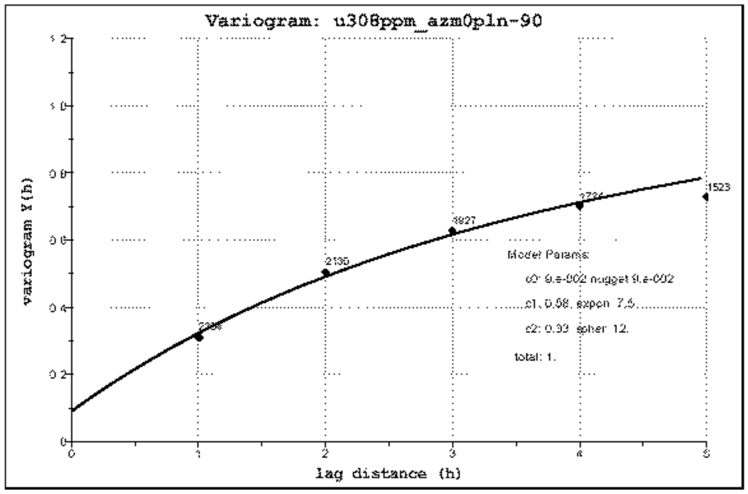


Figure 122: Down-hole variogram, Detail 5 domain 2

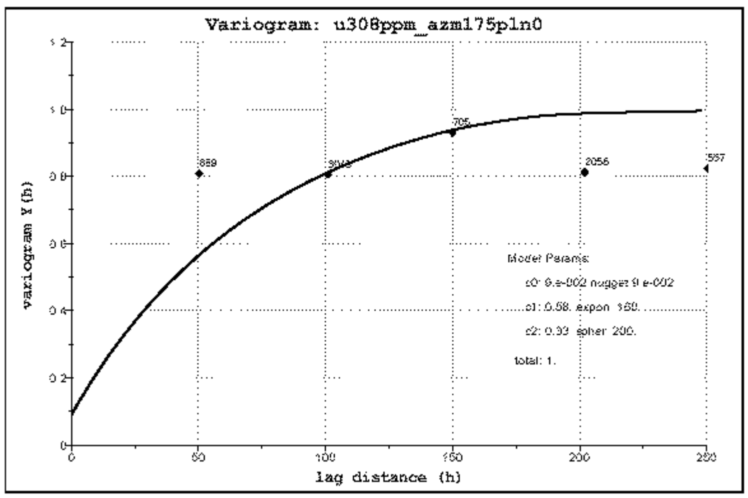


Figure 123: Along-strike variogram, Detail 5 domain 2

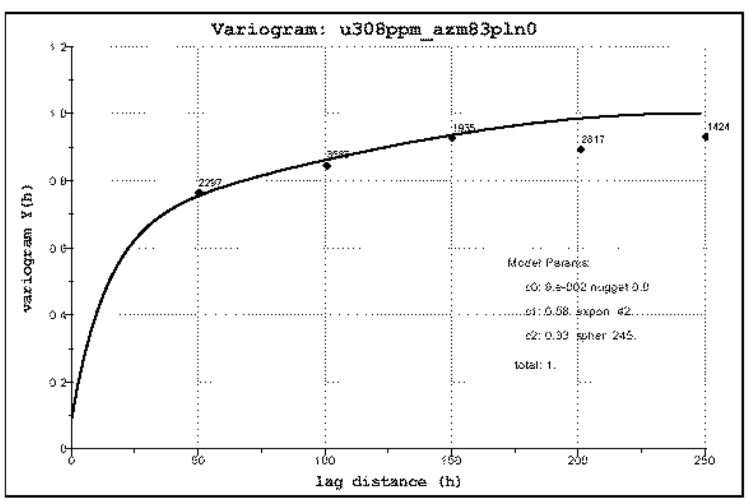


Figure 124: Across-strike variogram, Detail 5 domain 2

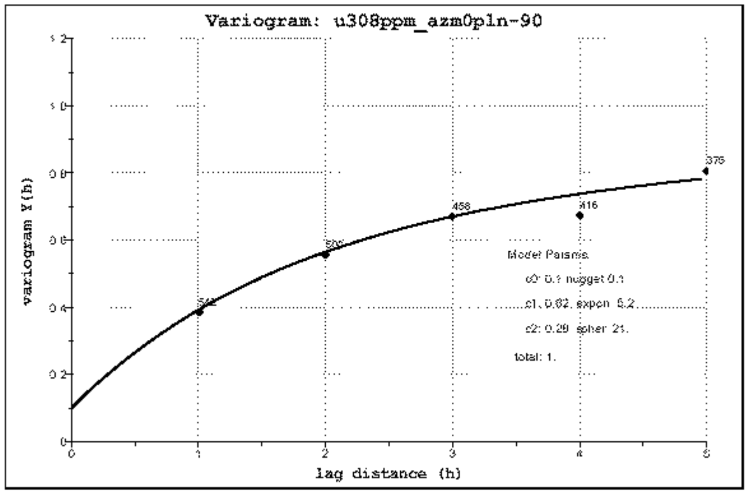


Figure 125: Down-hole variogram, Detail 6

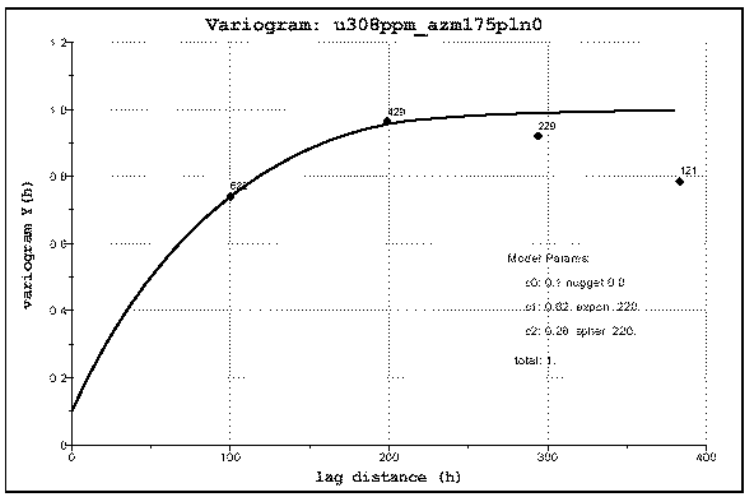


Figure 126: Along-strike variogram, Detail 6

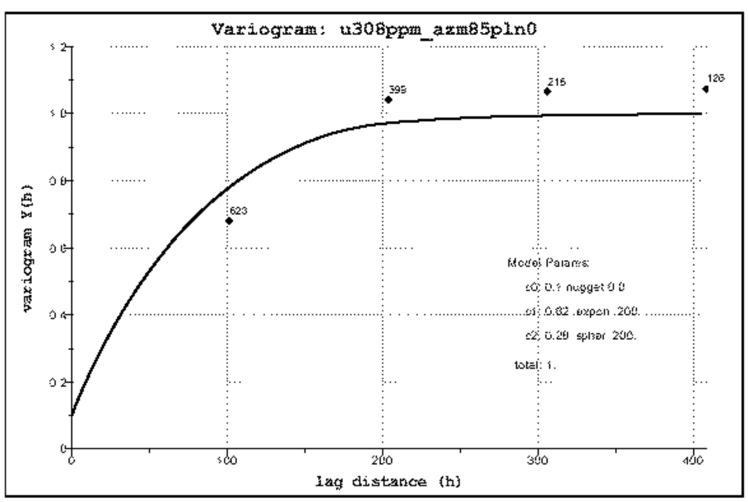


Figure 127: Across-strike variogram, Detail 6

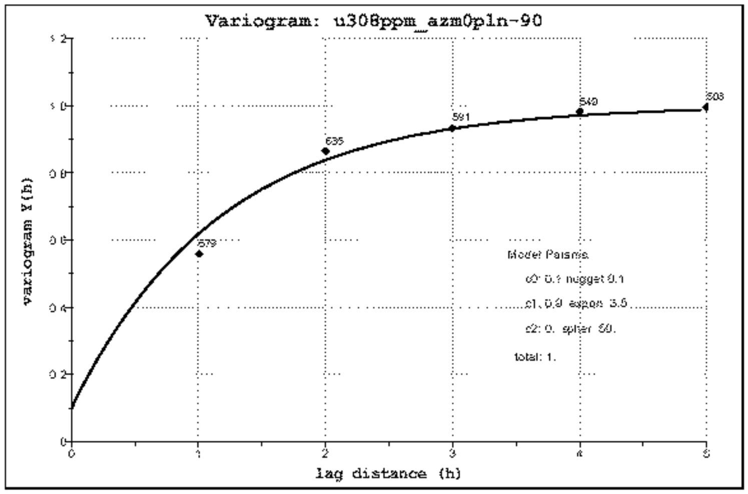


Figure 128: Down-hole variogram, Detail 7

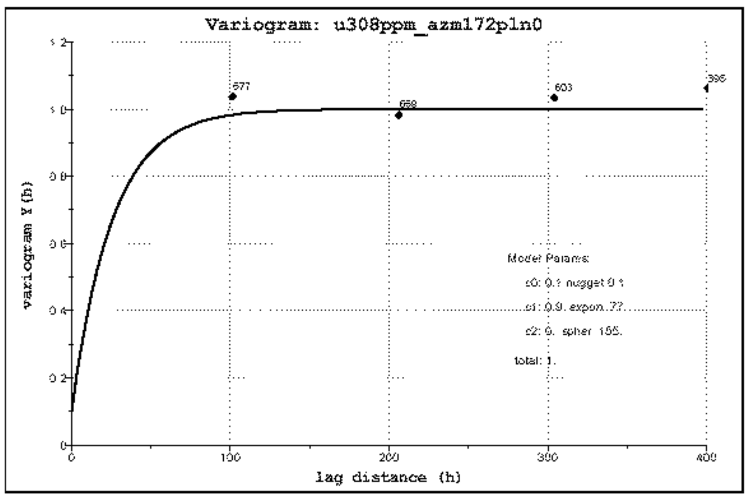


Figure 129: Along-strike variogram, Detail 7

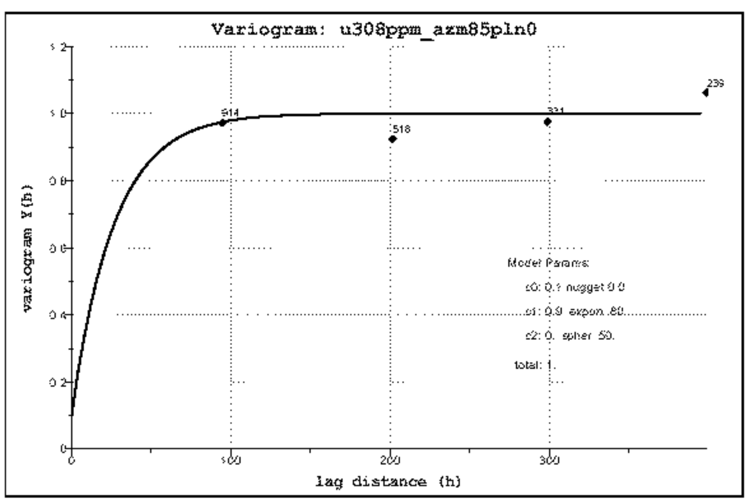


Figure 130: Across-strike variogram, Detail 7

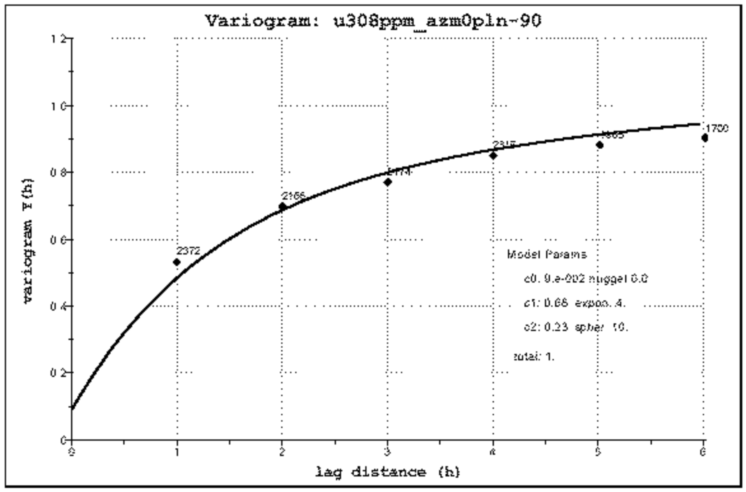


Figure 131: Variogram using mine drilling only, vertical direction

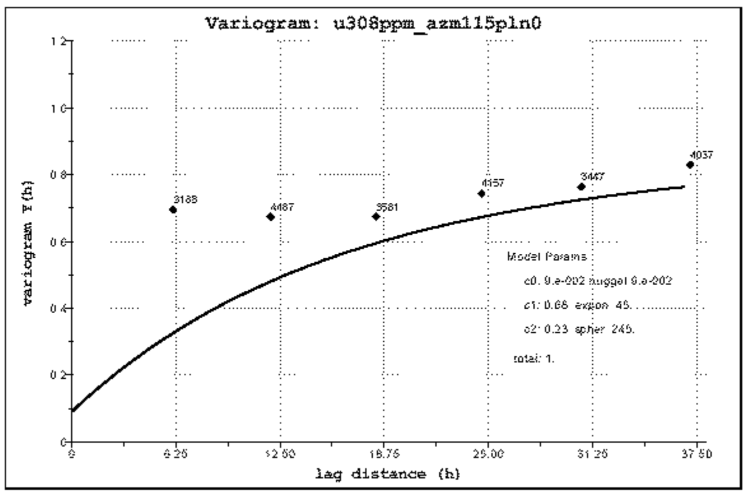


Figure 132: Variogram using mine drilling only, along-strike direction

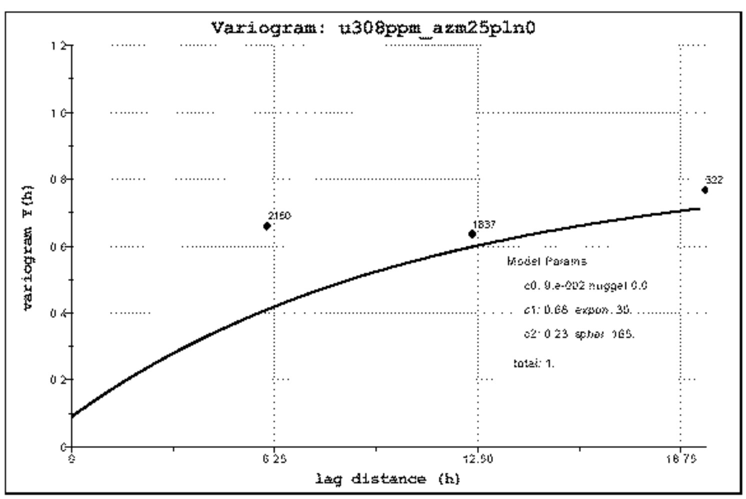


Figure 133: Variogram using mine drilling only, across-strike direction

17.8 Indicator Variograms

Sample data from each of Details 1 through 5 and Detail 7, and each of the domains in Details 1, 2 and 5, that lie above basement and above the water table were transformed to indicator data using probability thresholds at P = 0.4, 0.5, 0.6, 0.65, 0.7, 0.75, 0.8, 0.85, 0.9, 0.95, 0.97 and 0.99. In Detail 6 indicator thresholds were set at P = 0.5, 0.6, 0.7, 0.8, 0.85, 0.9, 0.95, 0.97 and 0.99. The election to use the relatively high first threshold was based on the large proportions of samples in each resource area that derive from samples that were deliberately not assayed and have been assumed to be below detection limit. Had these data been in distinct geographic areas it would have been feasible to remove them prior to estimation however they mainly represent barren and low-grade material over- and underlying the mineralisation. It was deemed desirable that they inform the estimates of resources.

For each of Details, and for each domain in Details 1, 2 and 5, experimental indicator variograms were calculated and modelled. Figure 134 shows an example set of down-hole indicator variograms and fitted models for domain 1 of Detail 1. Relative nuggets increase, and ranges decrease at increasing indicator thresholds as expected.

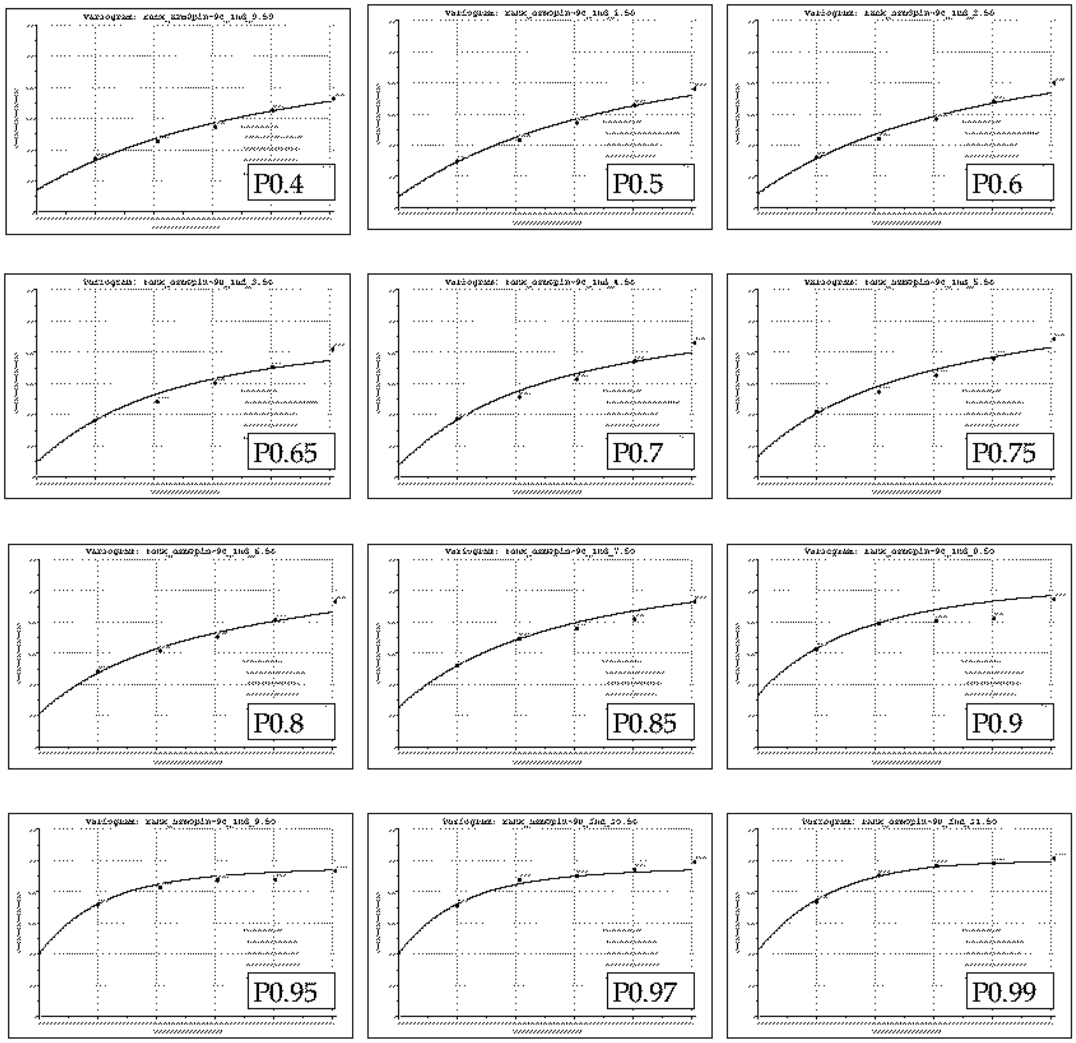


Figure 134: Down-hole indicator variograms, Detail 1 domain 1

17.9 Indicator Kriging Parameters

The input parameters to Indicator Kriging of the Langer Heinrich mineralisation include:

- Indicator variogram models describing the spatial continuity of indicator variables within each domain at each indicator threshold.
- Variograms describing the spatial continuity of U_3O_8 grades within each domain.
- Mean U_3O_8 grades of each of the indicator classes within each domain.

Figure 135 to Figure 143 list the indicator variogram models applied in each of the geological domains. The last variogram model listed in each table is the variogram model of U_3O_8 grades, used for calculation of variance adjustments.

Table 18 to Table 26 list the conditional statistics of sample data in each of the modelling domains. The statistics in Detail 1 domain 1 were calculated on data that exclude the close-spaced drilling in the trial mining area but those samples were included in the data that inform the indicator kriging estimates.

Table 9 shows the grid framework and kriging search parameters used in the indicator kriging models. Within each Detail, the boundaries between domains and between subdomains were treated as soft boundaries in the kriging process.

Variogram Models for Grade Thresholds in this Domain

Grade (eu3o8ppm threshold)		c0 sill	c1 sill	c1 type	c1 X	c1 Y	c1 Z	c2 sill	c2 type	c2 X	c2 Y	c2 Z	c3 sill	c3 type	c3 X	c3 Y	c3 Z	sill tot	ax1	ang	ax2	ang	ax3	ang
		0.14	0.48	exp	150	105	7.5	0.38	sph	.430	290	18	0		0	0	0	1	z	-20		0		0
1	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	7.5	[illegible]	[illegible]	430.0	[illegible]	[illegible]	[illegible]		[illegible]	[illegible]	[illegible]	[illegible]		-20		0		0
2	20.000	0.07	0.53	exp	130.0	90.0	7.0	0.40	sph	430.0	365.0	16.0	0.00		0.0	0.0	0.0	1.00		-20		0		0
3	10.000	0.09	0.54	exp	130.0	82.0	7.0	0.37	sph	430.0	390.0	16.0	0.00		0.0	0.0	0.0	1.00		-20		0		0
4	50.000	0.09	0.54	exp	115.0	70.0	5.0	0.37	sph	420.0	305.0	19.0	0.00		0.0	0.0	0.0	1.00		-20		0		0
5	25.000	0.07	0.57	exp	105.0	60.0	5.0	0.36	sph	410.0	255.0	14.0	0.00		0.0	0.0	0.0	1.00		-20		0		0
6	30.000	0.13	0.45	exp	120.0	45.0	4.5	0.42	sph	380.0	200.0	11.0	0.00		0.0	0.0	0.0	1.00		-20		0		0
7	95.000	0.21	0.38	exp	45.0	31.0	4.0	0.41	sph	370.0	170.0	10.0	0.00		0.0	0.0	0.0	1.00		-20		0		0
8	29.000	0.25	0.43	exp	44.0	36.0	4.5	0.32	sph	325.0	145.0	8.0	0.00		0.0	0.0	0.0	1.00		-20		0		0
9	89.000	0.33	0.46	exp	56.0	49.0	3.7	0.21	sph	270.0	125.0	7.0	0.00		0.0	0.0	0.0	1.00		-20		0		0
10	254.00	0.40	0.49	exp	40.0	46.0	3.0	0.11	sph	170.0	110.0	15.0	0.00		0.0	0.0	0.0	1.00		-30		0		0
11	589.00	0.40	0.52	exp	34.0	38.0	3.5	0.08	sph	75.0	53.0	6.5	0.00		0.0	0.0	0.0	1.00		-20		0		0
12	998.00	0.42	0.54	exp	34.0	35.0	3.5	0.04	sph	65.0	47.0	3.5	0.00		0.0	0.0	0.0	1.00		-20		0		0
13	3710.0	0.09	0.68	exp	45.0	30.0	4.0	0.23	sph	245.0	165.0	10.0	0.00		0.0	0.0	0.0	1.00		-20		0		0
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Figure 135: U_3O_8 and indicator variogram models applied to Detail 1, domain 1

Variogram Models for Grade Thresholds in this Domain

Grade (u308ppm threshold)		c0 sill	c1 sill	c1 type	c1 X	c1 Y	c1 Z	c2 sill	c2 type	c2 X	c2 Y	c2 Z	c3 sill	c3 type	c3 X	c3 Y	c3 Z	sill tot	ax1	ang	ax2	ang	ax3	ang
		0.24	0.45	exp	40	49	5.5	0.31	sph	160	240	36	0		0	0	0	1		0		0		0
1	15.000	0.24	0.45	exp	40.0	49.0	5.5	0.31	sph	160.0	240.0	36.0	0.00		0.0	0.0	0.0	1.00		0		0		0
2	15.000	0.13	0.58	exp	52.0	78.0	9.5	0.29	sph	210.0	230.0	36.0	0.00		0.0	0.0	0.0	1.00		0		0		0
3	40.000	0.07	0.53	exp	43.0	50.0	4.5	0.40	sph	180.0	165.0	36.0	0.00		0.0	0.0	0.0	1.00		0		0		0
4	60.000	0.07	0.58	exp	50.0	46.0	4.5	0.35	sph	185.0	165.0	36.0	0.00		0.0	0.0	0.0	1.00		0		0		0
5	90.000	0.10	0.51	exp	45.0	33.0	3.4	0.39	sph	170.0	165.0	15.0	0.00		0.0	0.0	0.0	1.00		0		0		0
6	10.000	0.15	0.51	exp	56.0	38.0	3.0	0.34	sph	170.0	165.0	15.0	0.00		0.0	0.0	0.0	1.00		0		0		0
7	60.000	0.19	0.51	exp	50.0	38.0	3.0	0.30	sph	160.0	150.0	15.0	0.00		0.0	0.0	0.0	1.00		0		0		0
8	01.000	0.23	0.51	exp	46.0	37.0	3.0	0.26	sph	150.0	135.0	15.0	0.00		0.0	0.0	0.0	1.00		0		0		0
9	75.000	0.28	0.51	exp	44.0	36.0	3.0	0.21	sph	135.0	115.0	13.0	0.00		0.0	0.0	0.0	1.00		0		0		0
10	9.000	0.39	0.51	exp	56.0	56.0	3.0	0.10	sph	115.0	95.0	7.0	0.00		0.0	0.0	0.0	1.00		0		0		0
11	60.000	0.42	0.51	exp	45.0	54.0	3.0	0.07	sph	70.0	87.0	6.0	0.00		0.0	0.0	0.0	1.00		0		0		0
12	135.00	0.48	0.52	exp	40.0	38.0	2.0	0.00		0.0	0.0	0.0	0.00		0.0	0.0	0.0	1.00		0		0		0
13	76.00	0.17	0.61	exp	38.0	33.0	1.5	0.22	sph	120.0	100.0	11.0	0.00		0.0	0.0	0.0	1.00		0		0		0
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Figure 136: U_3O_8 and indicator variogram models applied to Detail 2, domain 1

Variogram Models for Grade Thresholds in this Domain

Grade	u308ppm threshold	c0 sill	c1 sill	c1 type	c1 X	c1 Y	c1 Z	c2 sill	c2 type	c2 X	c2 Y	c2 Z	c3 sill	c3 type	c3 X	c3 Y	c3 Z	sill tol	ax1	ax1 ang	ax2	ax2 ang	ax3	ax3 ang
		0.11	0.53	exp	145	77	10	0.36	sph	225	160	22	0		0	0	0	1		0		0		0
1	20.000	0.11	0.53	exp	145.0	77.0	10.0	0.36	sph	225.0	160.0	22.0	0.00		0.0	0.0	0.0	1.00		0		0		0
2	70.000	0.08	0.45	exp	90.0	100.0	6.0	0.47	sph	225.0	130.0	16.0	0.00		0.0	0.0	0.0	1.00		0		0		0
3	30.000	0.08	0.45	exp	70.0	130.0	5.0	0.47	sph	220.0	130.0	14.0	0.00		0.0	0.0	0.0	1.00		0		0		0
4	10.000	0.09	0.48	exp	49.0	125.0	5.0	0.43	sph	205.0	125.0	16.0	0.00		0.0	0.0	0.0	1.00		0		0		0
5	30.000	0.10	0.54	exp	44.0	125.0	5.5	0.36	sph	195.0	125.0	19.0	0.00		0.0	0.0	0.0	1.00		0		0		0
6	30.000	0.14	0.54	exp	60.0	125.0	5.5	0.32	sph	180.0	125.0	32.0	0.00		0.0	0.0	0.0	1.00		0		0		0
7	30.000	0.18	0.59	exp	48.0	115.0	6.5	0.23	sph	175.0	115.0	48.0	0.00		0.0	0.0	0.0	1.00		0		0		0
8	39.000	0.19	0.58	exp	47.0	105.0	6.5	0.23	sph	160.0	105.0	48.0	0.00		0.0	0.0	0.0	1.00		0		0		0
9	27.000	0.27	0.48	exp	37.0	70.0	7.5	0.25	sph	95.0	90.0	48.0	0.00		0.0	0.0	0.0	1.00		0		0		0
10	460.00	0.33	0.48	exp	32.0	44.0	13.0	0.19	sph	55.0	72.0	48.0	0.00		0.0	0.0	0.0	1.00		0		0		0
11	300.00	0.42	0.48	exp	30.0	42.0	13.0	0.10	sph	50.0	64.0	16.0	0.00		0.0	0.0	0.0	1.00		0		0		0
12	300.00	0.50	0.48	exp	30.0	42.0	11.0	0.02	sph	50.0	64.0	16.0	0.00		0.0	0.0	0.0	1.00		0		0		0
13	5789.0	0.18	0.64	exp	40.0	64.0	4.2	0.18	sph	92.0	100.0	25.0	0.00		0.0	0.0	0.0	1.00		0		0		0

Read Model | Enter

Figure 137: U_3O_8 and indicator variogram models applied to Detail 2, domain 2

Variogram Models for Grade Thresholds in this Domain

Grade	u308ppm threshold	c0 sill	c1 sill	c1 type	c1 X	c1 Y	c1 Z	c2 sill	c2 type	c2 X	c2 Y	c2 Z	c3 sill	c3 type	c3 X	c3 Y	c3 Z	sill tol	ax1	ax1 ang	ax2	ax2 ang	ax3	ax3 ang
		0.23	0.41	exp	200	135	9	0.36	sph	440	500	56	0		0	0	0	1	z	-45		0		0
1	15.000	0.23	0.41	exp	200.0	135.0	9.0	0.36	sph	440.0	500.0	56.0	0.00		0.0	0.0	0.0	1.00		-45		0		0
2	30.000	0.05	0.45	exp	156.0	78.0	9.5	0.50	sph	480.0	500.0	66.0	0.00		0.0	0.0	0.0	1.00		-45		0		0
3	03.000	0.08	0.50	exp	89.0	74.0	9.5	0.42	sph	440.0	520.0	66.0	0.00		0.0	0.0	0.0	1.00		-45		0		0
4	36.000	0.07	0.65	exp	110.0	84.0	9.5	0.28	sph	450.0	520.0	66.0	0.00		0.0	0.0	0.0	1.00		-45		0		0
5	30.000	0.10	0.64	exp	92.0	56.0	9.5	0.26	sph	400.0	475.0	50.0	0.00		0.0	0.0	0.0	1.00		-45		0		0
6	30.000	0.16	0.66	exp	92.0	64.0	9.5	0.18	sph	375.0	405.0	50.0	0.00		0.0	0.0	0.0	1.00		-45		0		0
7	70.000	0.20	0.68	exp	92.0	72.0	10.0	0.12	sph	375.0	310.0	13.0	0.00		0.0	0.0	0.0	1.00		-45		0		0
8	30.000	0.21	0.68	exp	74.0	60.0	10.0	0.11	sph	355.0	195.0	15.0	0.00		0.0	0.0	0.0	1.00		-45		0		0
9	70.000	0.22	0.78	exp	62.0	60.0	7.0	0.00		0.0	0.0	0.0	0.00		0.0	0.0	0.0	1.00		-45		0		0
10	40.000	0.39	0.62	exp	68.0	66.0	6.5	0.00		0.0	0.0	0.0	0.00		0.0	0.0	0.0	1.01		-45		0		0
11	07.000	0.43	0.57	exp	58.0	54.0	6.5	0.00		0.0	0.0	0.0	0.00		0.0	0.0	0.0	1.00		-45		0		0
12	370.00	0.23	0.77	exp	45.0	40.0	2.5	0.00		0.0	0.0	0.0	0.00		0.0	0.0	0.0	1.00		-45		0		0
13	60.00	0.23	0.59	exp	58.0	44.0	4.6	0.18	sph	165.0	200.0	42.0	0.00		0.0	0.0	0.0	1.00		-45		0		0

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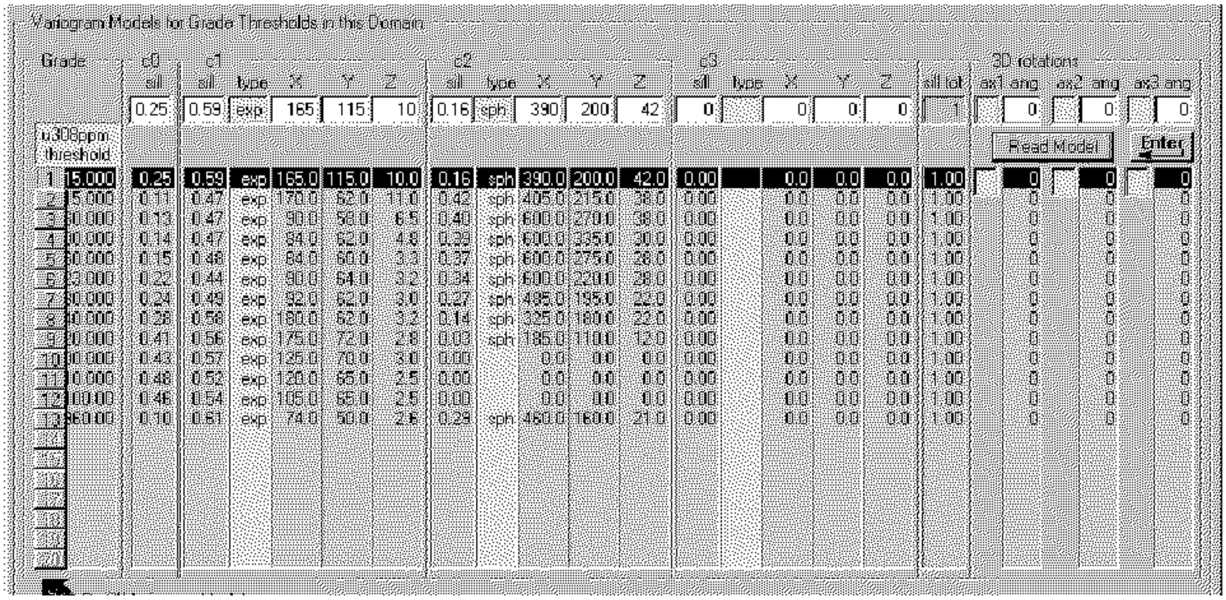
Figure 138: U_3O_8 and indicator variogram models applied to Detail 3



Variogram Models for Grade Thresholds in this Domain

Grade (u308ppm threshold)		c0 sill	c1 sill	c1 type	c1 X	c1 Y	c1 Z	c2 sill	c2 type	c2 X	c2 Y	c2 Z	c3 sill	c3 type	c3 X	c3 Y	c3 Z	sill tot	3D rotations ax1	ax1 ang	ax2	ax2 ang	ax3	ax3 ang
		0.25	0.59	exp	165	115	10	0.16	sph	390	200	42	0		0	0	0	1		0		0		0
1	15.000	0.25	0.59	exp	165.0	115.0	10.0	0.16	sph	390.0	200.0	42.0	0.00		0.0	0.0	0.0	1.00		0		0		0
2	5.000	0.11	0.47	exp	170.0	62.0	11.0	0.42	sph	405.0	215.0	38.0	0.00		0.0	0.0	0.0	1.00		0		0		0
3	0.000	0.13	0.47	exp	90.0	58.0	6.5	0.40	sph	600.0	270.0	38.0	0.00		0.0	0.0	0.0	1.00		0		0		0
4	0.000	0.14	0.47	exp	84.0	62.0	4.8	0.39	sph	600.0	335.0	30.0	0.00		0.0	0.0	0.0	1.00		0		0		0
5	0.000	0.15	0.48	exp	84.0	60.0	3.3	0.37	sph	600.0	275.0	28.0	0.00		0.0	0.0	0.0	1.00		0		0		0
6	3.000	0.22	0.44	exp	90.0	64.0	3.2	0.34	sph	600.0	220.0	28.0	0.00		0.0	0.0	0.0	1.00		0		0		0
7	0.000	0.24	0.49	exp	92.0	62.0	3.0	0.27	sph	485.0	195.0	22.0	0.00		0.0	0.0	0.0	1.00		0		0		0
8	0.000	0.28	0.58	exp	180.0	62.0	3.2	0.14	sph	325.0	180.0	22.0	0.00		0.0	0.0	0.0	1.00		0		0		0
9	0.000	0.41	0.56	exp	175.0	72.0	2.8	0.03	sph	185.0	110.0	12.0	0.00		0.0	0.0	0.0	1.00		0		0		0
10	0.000	0.43	0.57	exp	125.0	70.0	3.0	0.00		0.0	0.0	0.0	0.00		0.0	0.0	0.0	1.00		0		0		0
11	0.000	0.48	0.52	exp	120.0	65.0	2.5	0.00		0.0	0.0	0.0	0.00		0.0	0.0	0.0	1.00		0		0		0
12	00.00	0.46	0.54	exp	105.0	65.0	2.5	0.00		0.0	0.0	0.0	0.00		0.0	0.0	0.0	1.00		0		0		0
13	60.00	0.10	0.61	exp	74.0	50.0	2.6	0.29	sph	460.0	160.0	21.0	0.00		0.0	0.0	0.0	1.00		0		0		0
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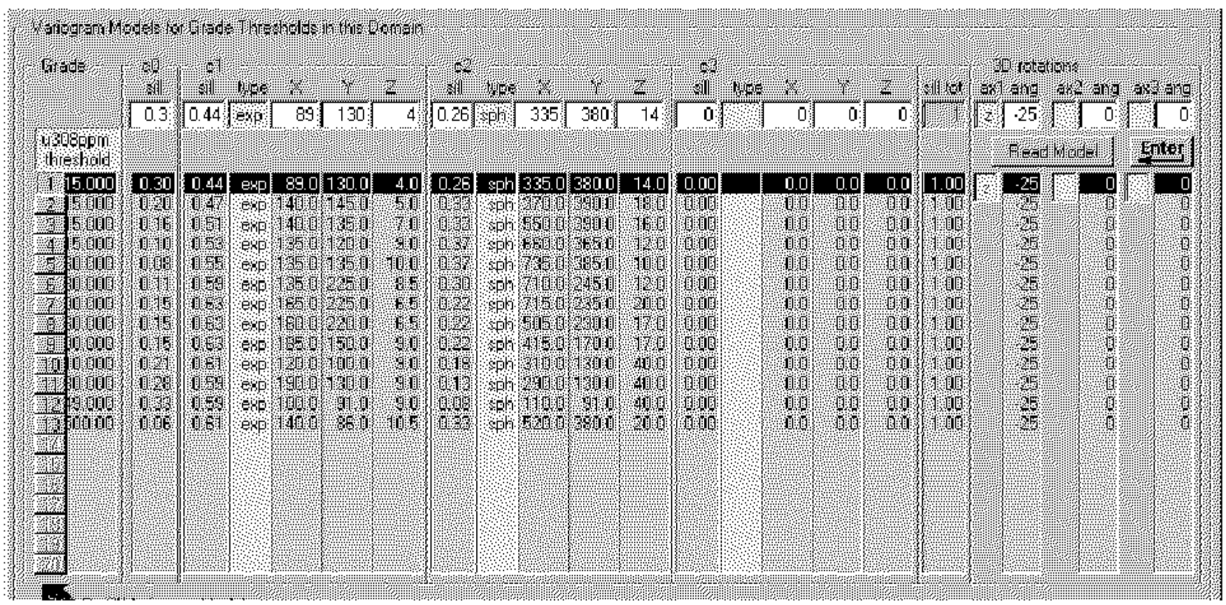
Figure 139: U_3O_8 and indicator variogram models applied to Detail 4



Variogram Models for Grade Thresholds in this Domain

Grade (u308ppm threshold)		c0 sill	c1 sill	c1 type	c1 X	c1 Y	c1 Z	c2 sill	c2 type	c2 X	c2 Y	c2 Z	c3 sill	c3 type	c3 X	c3 Y	c3 Z	sill tot	3D rotations ax1	ax1 ang	ax2	ax2 ang	ax3	ax3 ang
		0.3	0.44	exp	89	130	4	0.26	sph	335	380	14	0		0	0	0	1	z	-25		0		0
1	15.000	0.30	0.44	exp	89.0	130.0	4.0	0.26	sph	335.0	380.0	14.0	0.00		0.0	0.0	0.0	1.00	z	-25		0		0
2	5.000	0.20	0.47	exp	140.0	145.0	5.0	0.33	sph	370.0	390.0	18.0	0.00		0.0	0.0	0.0	1.00		-25		0		0
3	5.000	0.16	0.51	exp	140.0	135.0	7.0	0.33	sph	550.0	390.0	16.0	0.00		0.0	0.0	0.0	1.00		-25		0		0
4	5.000	0.10	0.53	exp	135.0	120.0	9.0	0.37	sph	660.0	365.0	12.0	0.00		0.0	0.0	0.0	1.00		-25		0		0
5	0.000	0.08	0.55	exp	135.0	135.0	10.0	0.37	sph	735.0	385.0	10.0	0.00		0.0	0.0	0.0	1.00		-25		0		0
6	0.000	0.11	0.59	exp	135.0	225.0	8.5	0.30	sph	710.0	245.0	12.0	0.00		0.0	0.0	0.0	1.00		-25		0		0
7	0.000	0.15	0.63	exp	185.0	225.0	6.5	0.22	sph	715.0	235.0	20.0	0.00		0.0	0.0	0.0	1.00		-25		0		0
8	0.000	0.15	0.63	exp	180.0	220.0	6.5	0.22	sph	505.0	230.0	17.0	0.00		0.0	0.0	0.0	1.00		-25		0		0
9	0.000	0.15	0.63	exp	185.0	150.0	9.0	0.22	sph	415.0	170.0	17.0	0.00		0.0	0.0	0.0	1.00		-25		0		0
10	0.000	0.21	0.61	exp	120.0	100.0	9.0	0.18	sph	310.0	130.0	40.0	0.00		0.0	0.0	0.0	1.00		-25		0		0
11	0.000	0.28	0.59	exp	190.0	130.0	9.0	0.13	sph	290.0	130.0	40.0	0.00		0.0	0.0	0.0	1.00		-25		0		0
12	9.000	0.33	0.59	exp	100.0	91.0	9.0	0.08	sph	110.0	91.0	40.0	0.00		0.0	0.0	0.0	1.00		-25		0		0
13	00.00	0.06	0.61	exp	140.0	86.0	10.5	0.33	sph	520.0	380.0	20.0	0.00		0.0	0.0	0.0	1.00		-25		0		0
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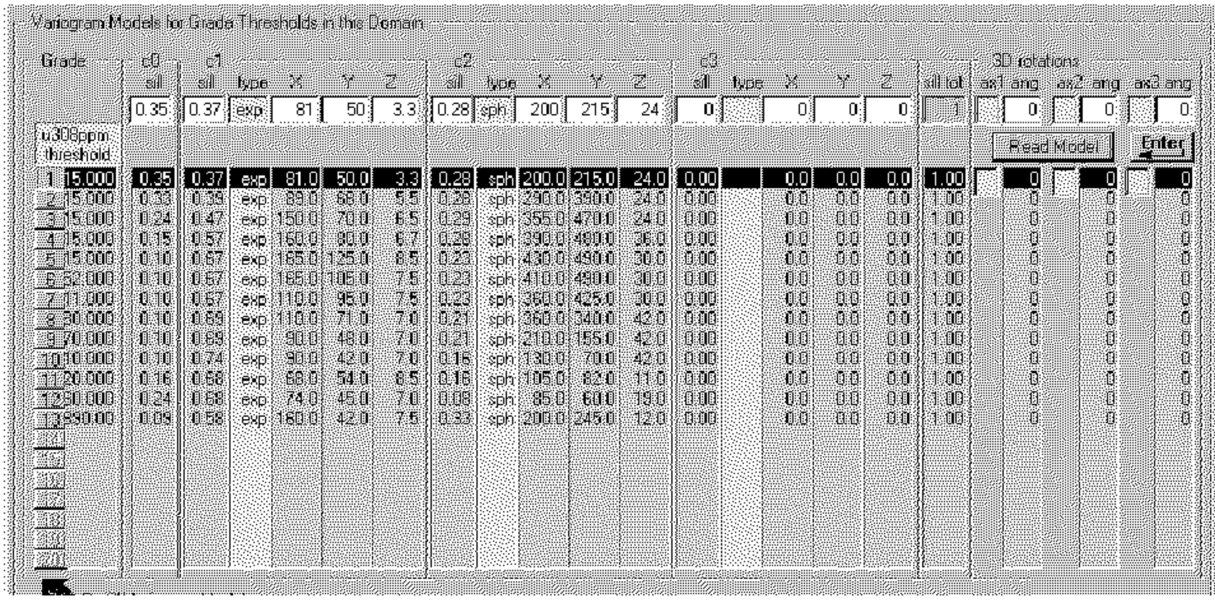
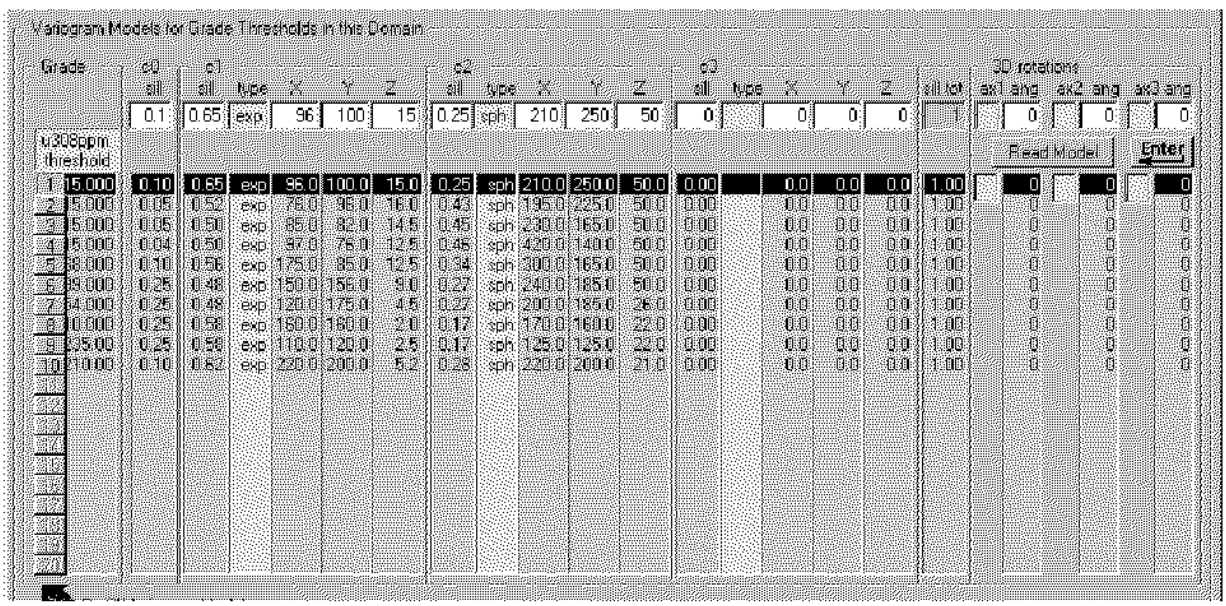
Figure 140: U_3O_8 and indicator variogram models applied to Detail 5, domain 1



Variogram Models for Grade Thresholds in this Domain

Grade	u308ppm threshold	c0 sill	c1 sill	c1 type	c1 X	c1 Y	c1 Z	c2 sill	c2 type	c2 X	c2 Y	c2 Z	c3 sill	c3 type	c3 X	c3 Y	c3 Z	sill tot	ax1 ang	ax2 ang	ax3 ang
		0.35	0.37	exp	81	50	3.3	0.28	sph	200	215	24	0		0	0	0	1	0	0	0
1	15.000	0.35	0.37	exp	81.0	50.0	3.3	0.28	sph	200.0	215.0	24.0	0.00		0.0	0.0	0.0	1.00	0	0	0
2	15.000	0.33	0.39	exp	89.0	68.0	5.5	0.28	sph	290.0	390.0	24.0	0.00		0.0	0.0	0.0	1.00	0	0	0
3	15.000	0.24	0.47	exp	150.0	70.0	6.5	0.29	sph	355.0	470.0	24.0	0.00		0.0	0.0	0.0	1.00	0	0	0
4	15.000	0.15	0.57	exp	160.0	80.0	6.7	0.28	sph	390.0	480.0	36.0	0.00		0.0	0.0	0.0	1.00	0	0	0
5	15.000	0.10	0.67	exp	165.0	125.0	8.5	0.23	sph	430.0	490.0	30.0	0.00		0.0	0.0	0.0	1.00	0	0	0
6	52.000	0.10	0.67	exp	165.0	105.0	7.5	0.23	sph	410.0	490.0	30.0	0.00		0.0	0.0	0.0	1.00	0	0	0
7	11.000	0.10	0.67	exp	110.0	95.0	7.5	0.23	sph	360.0	425.0	30.0	0.00		0.0	0.0	0.0	1.00	0	0	0
8	30.000	0.10	0.69	exp	110.0	71.0	7.0	0.21	sph	360.0	340.0	42.0	0.00		0.0	0.0	0.0	1.00	0	0	0
9	70.000	0.10	0.69	exp	90.0	48.0	7.0	0.21	sph	210.0	155.0	42.0	0.00		0.0	0.0	0.0	1.00	0	0	0
10	10.000	0.10	0.74	exp	90.0	42.0	7.0	0.16	sph	130.0	70.0	42.0	0.00		0.0	0.0	0.0	1.00	0	0	0
11	20.000	0.16	0.68	exp	68.0	54.0	8.5	0.16	sph	105.0	82.0	11.0	0.00		0.0	0.0	0.0	1.00	0	0	0
12	50.000	0.24	0.68	exp	74.0	45.0	7.0	0.08	sph	85.0	60.0	19.0	0.00		0.0	0.0	0.0	1.00	0	0	0
13	990.00	0.09	0.58	exp	160.0	42.0	7.5	0.33	sph	200.0	245.0	12.0	0.00		0.0	0.0	0.0	1.00	0	0	0

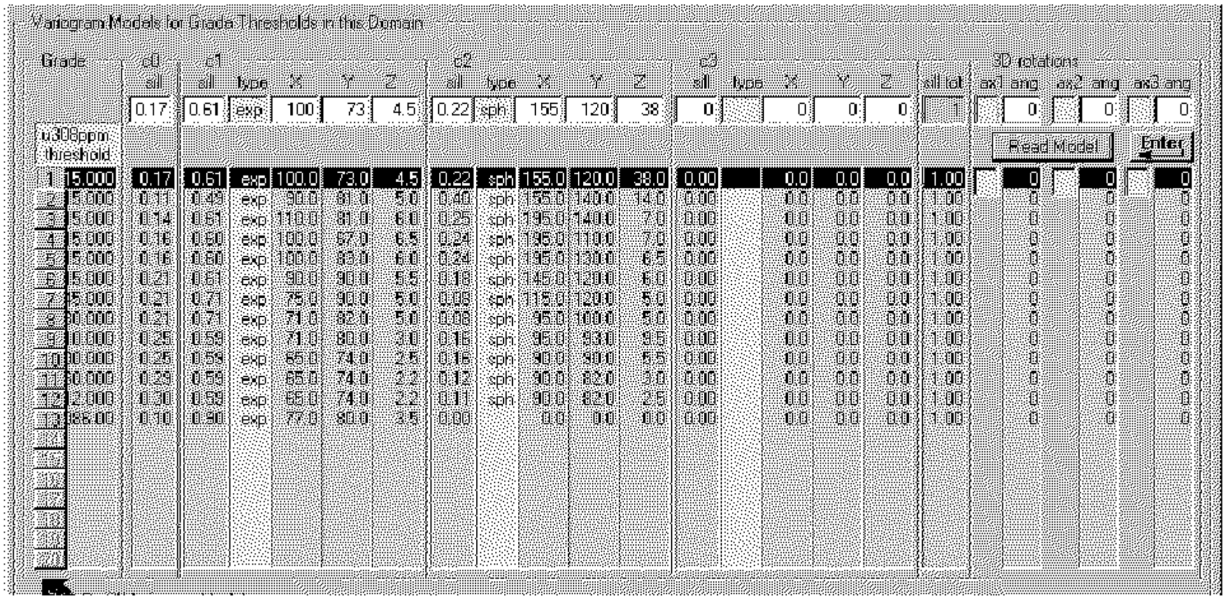
Figure 141: U_3O_8 and indicator variogram models applied to Detail 5, domain 2



Variogram Models for Grade Thresholds in this Domain

Grade	u308ppm threshold	c0 sill	c1 sill	c1 type	c1 X	c1 Y	c1 Z	c2 sill	c2 type	c2 X	c2 Y	c2 Z	c3 sill	c3 type	c3 X	c3 Y	c3 Z	sill tot	ax1 ang	ax2 ang	ax3 ang
		0.1	0.65	exp	96	100	15	0.25	sph	210	250	50	0		0	0	0	1	0	0	0
1	15.000	0.10	0.65	exp	96.0	100.0	15.0	0.25	sph	210.0	250.0	50.0	0.00		0.0	0.0	0.0	1.00	0	0	0
2	5.000	0.05	0.52	exp	76.0	96.0	16.0	0.43	sph	195.0	225.0	50.0	0.00		0.0	0.0	0.0	1.00	0	0	0
3	5.000	0.05	0.50	exp	85.0	82.0	14.5	0.45	sph	230.0	165.0	50.0	0.00		0.0	0.0	0.0	1.00	0	0	0
4	5.000	0.04	0.50	exp	97.0	76.0	12.5	0.46	sph	420.0	140.0	50.0	0.00		0.0	0.0	0.0	1.00	0	0	0
5	8.000	0.10	0.56	exp	175.0	85.0	12.5	0.34	sph	300.0	165.0	50.0	0.00		0.0	0.0	0.0	1.00	0	0	0
6	9.000	0.25	0.48	exp	150.0	156.0	9.0	0.27	sph	240.0	185.0	50.0	0.00		0.0	0.0	0.0	1.00	0	0	0
7	4.000	0.25	0.48	exp	120.0	175.0	4.5	0.27	sph	200.0	185.0	26.0	0.00		0.0	0.0	0.0	1.00	0	0	0
8	0.000	0.25	0.58	exp	160.0	160.0	2.0	0.17	sph	170.0	160.0	22.0	0.00		0.0	0.0	0.0	1.00	0	0	0
9	35.00	0.25	0.58	exp	110.0	120.0	2.5	0.17	sph	125.0	125.0	22.0	0.00		0.0	0.0	0.0	1.00	0	0	0
10	10.00	0.10	0.62	exp	220.0	200.0	5.2	0.28	sph	220.0	200.0	21.0	0.00		0.0	0.0	0.0	1.00	0	0	0

Figure 142: U_3O_8 and indicator variogram models applied to Detail 6



Variogram Models for Grade Thresholds in this Domain

Grade (u3o8ppm threshold)		c0 sill	c1 sill	c1 type	c1 X	c1 Y	c1 Z	c2 sill	c2 type	c2 X	c2 Y	c2 Z	c3 sill	c3 type	c3 X	c3 Y	c3 Z	sill tot	ax1 ang	ax2 ang	ax3 ang
		0.17	0.61	exp	100	73	4.5	0.22	sph	155	120	38	0		0	0	0	1	0	0	0
1	15.000	0.17	0.61	exp	100.0	73.0	4.5	0.22	sph	155.0	120.0	38.0	0.00		0.0	0.0	0.0	1.00	0	0	0
2	5.000	0.11	0.49	exp	90.0	81.0	5.0	0.40	sph	155.0	140.0	14.0	0.00		0.0	0.0	0.0	1.00	0	0	0
3	5.000	0.14	0.61	exp	110.0	81.0	6.0	0.25	sph	195.0	140.0	7.0	0.00		0.0	0.0	0.0	1.00	0	0	0
4	5.000	0.16	0.60	exp	100.0	67.0	6.5	0.24	sph	195.0	110.0	7.0	0.00		0.0	0.0	0.0	1.00	0	0	0
5	5.000	0.16	0.60	exp	100.0	83.0	6.0	0.24	sph	195.0	130.0	6.5	0.00		0.0	0.0	0.0	1.00	0	0	0
6	5.000	0.21	0.61	exp	90.0	90.0	5.5	0.18	sph	145.0	120.0	6.0	0.00		0.0	0.0	0.0	1.00	0	0	0
7	5.000	0.21	0.71	exp	75.0	90.0	5.0	0.08	sph	115.0	120.0	5.0	0.00		0.0	0.0	0.0	1.00	0	0	0
8	0.000	0.21	0.71	exp	71.0	82.0	5.0	0.08	sph	95.0	100.0	5.0	0.00		0.0	0.0	0.0	1.00	0	0	0
9	0.000	0.25	0.59	exp	71.0	80.0	3.0	0.16	sph	95.0	93.0	9.5	0.00		0.0	0.0	0.0	1.00	0	0	0
10	0.000	0.25	0.59	exp	65.0	74.0	2.5	0.16	sph	90.0	90.0	5.5	0.00		0.0	0.0	0.0	1.00	0	0	0
11	0.000	0.29	0.59	exp	65.0	74.0	2.2	0.12	sph	90.0	82.0	3.0	0.00		0.0	0.0	0.0	1.00	0	0	0
12	2.000	0.30	0.59	exp	65.0	74.0	2.2	0.11	sph	90.0	82.0	2.5	0.00		0.0	0.0	0.0	1.00	0	0	0
13	86.00	0.10	0.90	exp	77.0	80.0	3.5	0.00		0.0	0.0	0.0	0.00		0.0	0.0	0.0	1.00	0	0	0

Read Model

Enter

Figure 143: U_3O_8 and indicator variogram models applied to Detail 7

	Detail 1 dom1sub1			Detail 1 dom1sub2		
Cumulative Probability	Grade Threshold	Class Mean	Class Data	Grade Threshold	Class Mean	Class Data
0.40	36	8	3539	36	15	3663
0.50	73	53	885	74	52	916
0.60	134	100	884	155	111	916
0.65	180	158	443	212	182	458
0.70	233	205	442	284	247	458
0.75	300	265	443	366	326	458
0.80	380	335	442	478	420	458
0.85	480	426	442	643	556	458
0.90	632	555	443	880	750	458
0.95	915	750	442	1470	1113	458
0.97	1150	1015	177	2010	1728	183
0.99	1832	1411	177	3667	2567	183
1.00	10637	2841	89	15547	5857	92

Table 18: Conditional statistics of data in Detail 1, domain 1

	Detail 2 dom1sub1			Detail 2 dom1sub2		
Cumulative Probability	Grade Threshold	Class Mean	Class Data	Grade Threshold	Class Mean	Class Data
0.40	12	1	1232	8	1	1109
0.50	32	21	308	20	14	277
0.60	53	43	308	40	29	277
0.65	70	61	154	57	48	139
0.70	85	76	154	79	67	139
0.75	101	94	154	110	93	138
0.80	132	116	154	160	134	139
0.85	180	158	154	226	186	139
0.90	249	209	154	330	272	138
0.95	380	310	154	609	432	139
0.97	510	437	62	900	734	55
0.99	950	682	62	1871	1205	56
1.00	3770	1518	31	6087	3486	28

Table 19: Conditional statistics of data in Detail 2, domain 1

	Detail 2 dom2sub1			Detail 2 dom2sub2		
Cumulative Probability	Grade Threshold	Class Mean	Class Data	Grade Threshold	Class Mean	Class Data
0.40	25	3	1237	39	10	1765
0.50	50	38	309	80	56	442
0.60	90	68	309	144	109	441
0.65	128	106	155	186	165	221
0.70	170	148	155	240	212	220
0.75	230	196	154	310	274	221
0.80	310	269	155	384	344	221
0.85	408	351	155	512	440	220
0.90	590	490	154	728	618	221
0.95	1000	762	155	1200	960	221
0.97	1417	1189	62	1504	1394	88
0.99	2510	1865	62	2205	1773	88
1.00	15789	5415	31	14784	3350	45

Table 20: Conditional statistics of data in Detail 2, domain 2

	Detail 3 dom1sub1			Detail 3 dom1sub2		
Cumulative Probability	Grade Threshold	Class Mean	Class Data	Grade Threshold	Class Mean	Class Data
0.30	15	15	273	15	11	175
0.40	50	28	272	17	15	175
0.50	95	71	273	43	28	176
0.60	160	125	273	93	66	175
0.70	230	196	272	170	130	175
0.75	269	245	137	217	194	88
0.80	310	287	136	280	243	87
0.85	350	331	136	376	319	88
0.90	420	384	137	555	451	87
0.95	577	489	136	930	697	88
0.97	767	656	55	1278	1072	35
0.99	1270	968	54	2100	1629	35
1.00	7160	2090	28	5564	3063	18

Table 21: Conditional statistics of data in Detail 3

	Detail 4 dom1sub1			Detail 4 dom1sub2		
Cumulative Probability	Grade Threshold	Class Mean	Class Data	Grade Threshold	Class Mean	Class Data
0.30	30	19	133	15	15	50
0.40	53	40	134	15	15	50
0.50	109	80	134	46	23	50
0.60	199	151	133	135	89	50
0.70	270	233	134	217	177	50
0.75	320	296	67	270	245	25
0.80	367	345	66	320	302	25
0.85	432	398	67	374	348	25
0.90	550	493	67	490	427	25
0.95	747	630	67	625	551	25
0.97	870	813	26	742	672	10
0.99	1200	1044	27	1140	975	10
1.00	2960	1906	14	1790	1460	6

Table 22: Conditional statistics of data in Detail 4

	Detail 5 dom1sub1			Detail 5 dom1sub2		
Cumulative Probability	Grade Threshold	Class Mean	Class Data	Grade Threshold	Class Mean	Class Data
0.30	15	12	87	15	15	116
0.40	18	15	87	15	15	117
0.50	60	38	88	26	19	116
0.60	120	91	87	53	38	116
0.70	224	176	87	100	75	117
0.75	260	240	44	146	122	58
0.80	325	289	43	203	177	58
0.85	399	354	44	267	237	58
0.90	480	444	43	342	319	58
0.95	620	544	44	577	449	58
0.97	746	662	17	708	640	24
0.99	1491	970	18	1500	1065	23
1.00	4787	2066	9	7259	3312	12

Table 23: Conditional statistics of data in Detail 5, domain 1

	Detail 5 dom2sub1			Detail 5 dom2sub2		
Cumulative Probability	Grade Threshold	Class Mean	Class Data	Grade Threshold	Class Mean	Class Data
0.30	9	6	205	15	15	181
0.40	15	13	206	22	16	181
0.50	15	15	205	55	36	181
0.60	27	17	206	113	81	181
0.70	60	40	205	187	154	180
0.75	95	73	103	239	212	91
0.80	140	114	103	295	269	91
0.85	192	170	102	360	322	90
0.90	280	234	103	496	423	90
0.95	433	350	103	700	586	91
0.97	560	498	41	895	779	36
0.99	740	604	41	1381	1091	36
1.00	1625	1028	21	2110	1777	19

Table 24: Conditional statistics of data in Detail 5, domain 2

	Detail 6 dom1sub1			Detail 6 dom1sub2		
Cumulative Probability	Grade Threshold	Class Mean	Class Data	Grade Threshold	Class Mean	Class Data
0.30	5	4	65	4	3	52
0.40	7	6	65	5	5	52
0.50	10	8	65	7	6	52
0.60	19	14	65	9	8	52
0.70	44	31	64	15	13	52
0.75	81	58	33	15	15	26
0.80	140	110	33	18	16	26
0.85	217	177	32	30	24	26
0.90	313	269	32	63	44	26
0.95	587	419	33	173	113	26
0.97	819	670	13	274	223	11
0.99	1608	1015	13	504	425	10
1.00	3210	2184	7	841	586	6

Table 25: Conditional statistics of data in Detail 6

	Detail 7 dom2sub1			Detail 7 dom2sub2		
Cumulative Probability	Grade Threshold	Class Mean	Class Data	Grade Threshold	Class Mean	Class Data
0.40	2	1	782	6	1	2970
0.50	3	3	196	12	9	743
0.60	4	4	195	22	17	743
0.65	5	4	98	29	25	371
0.70	6	5	98	39	33	371
0.75	7	6	98	54	46	372
0.80	10	9	97	77	64	371
0.85	12	10	98	116	95	371
0.90	20	16	98	187	149	372
0.95	30	24	98	392	264	371
0.97	41	36	39	621	490	149
0.99	65	50	39	1333	902	148
1.00	179	89	20	19962	3437	75

Table 26: Conditional statistics of data in Detail 7

17.10 Block Support Adjustment (Variance Adjustment)

17.10.1 General

The block support adjustment is one of the most important properties of a recoverable resource model based on non-linear estimation methods like MIK. It is an essential part of the model and involves important assumptions about the nature of the block grade distribution within each panel of the model.

Indicator Kriging provides a direct and reliable estimate of the histogram of grades of sample-sized units within each panel of the model provided the panel dimensions are of an appropriate size. However, ore is not selected on sample-sized units during mining; it is selected by shovels that have a minimum mining width and loaded into trucks that are despatched to either ore or waste. The flexibility of digging equipment and the size of the trucking equipment provide an indication of the size of the smallest block of rock that will be mined as ore or waste. To estimate with some accuracy the resources in a deposit that will be recovered with a certain set of mining equipment, the histogram of grades of sample-sized units in a panel provided by MIK must be adjusted to account for the size of the mining block.

There are a number of adjustment methods that can be used and most of these are described well in Journel & Huijbregts (1978) or Isaaks & Srivastava (1989). These methods make three reasonable assumptions:

- The average grade of sample-sized units and blocks within the panel is the same and is equal to the estimated average grade of the panel.
- The variance, or spread, of the block grades within the panel is less than the variance of grades of sample-sized units within the panel and the change of variance from sample-sized units to blocks can be calculated from the variogram of metal grades.

- The approximate shape of the histogram of block grades can be reasonably predicted by some appropriate assumptions.

17.10.2 The Variance Adjustment

The size of the variance adjustment needed to obtain the variance of the block grade distribution within the panel can be calculated using the rule of additivity of variances, which in the case of block support adjustment is often called Krige's Relationship:

Var(samples in a panel) = Var(samples in a block) + Var (blocks in a panel)

The variance of sample grades in a panel and the variance of samples within a block can be directly calculated from the variogram of metal grades for the particular domain. The ratio of Var(blocks in panel) to Var(samples in panel) is that required to implement the block support adjustment.

17.10.3 Shape of the Block grade Distribution

There are a number of rules of thumb that are useful when making judgements about the shape of the block grade distribution within each panel and they relate to the size of the variance adjustment ratio:

- If the variance adjustment ratio is greater than 0.7, it may be useful to assume that the shape of the histogram of block grades is similar to that of the histogram of grades of sample-sized units. This is known as the Affine Correction method. Its application to deposits sensitive to extreme sample grades is usually inappropriate.

- If the variance adjustment ratio is between 0.3 and 0.7 and the information adjustment is negligible, then the Indirect Lognormal Correction method of Isaaks & Srivastava (1989) can be useful. This is a rule of thumb based on the experience of the authors.

- If the variance adjustment ratio is less than 0.3, it is reasonable to assume there is a high degree of symmetrization in the block grade histogram. If the histogram of sample grades in a panel is positively skewed, the histogram of block grades is assumed to be lognormal in shape. If the histogram of sample grades in a panel is approximately symmetrical or negatively skewed, the block grade histogram is assumed to be normal in shape. The theoretical support for these assumptions comes from the Central Limit Theorem of probability. The theory supports the interpretation that as the variance adjustment ratio becomes very small, the shape of the block grade distribution must approach that of a normal distribution. This fact can also be demonstrated using geostatistical conditional simulation. In H&S's implementation of MIK, this approach is called the Lognormal-Normal Correction method. As implemented by H&S, the shape of the histogram of sample-sized units is assessed on a panel-by-panel basis.

17.10.4 The Information Effect

The variance adjustment described above is only part of the adjustment required in many mineral deposits where the short scale variation in metal grades is extreme. This variance adjustment provides an estimate of the variance of true block grades under the assumption that grade control selection will operate with knowledge of the true block grades. While this assumption is never absolutely true, it can be a reasonable assumption in some deposits where the short scale variability is small and the grade control sampling density is high. In many deposits, however, an additional variance adjustment must be undertaken to account for the "Information Effect".

In the absence of production information or grade control sampling, the Information Effect ratio is based on the variograms of metal grade and on the grade control sample spacing expected to be used during mining.

17.10.5 Variance Adjustments Applied to the Langer Heinrich Models

Variance adjustment ratios applied in estimating Langer Heinrich U_3O_8 resources are listed in *Table 27*. These ratios have been applied using the Indirect Lognormal Correction method (i.e., incorporating symmetrization of block grade distributions). Selective mining (SMU) dimensions of 5mE x 5mN x 2mRL and grade control sample spacing of 5mE x 5mN x 1mRL have been assumed.

	Panel to block adjustment	Information effect	Total ratio
Detail 1, domain 1	0.499	0.914	0.456
Detail 2, domain 1	0.312	0.823	0.257
Detail 2, domain 2	0.463	0.908	0.420
Detail 3	0.406	0.770	0.313
Detail 4	0.404	0.802	0.324
Detail 5, domain 1	0.579	0.861	0.499
Detail 5, domain 2	0.579	0.861	0.499
Detail 6	0.406	0.802	0.326
Detail 7	0.400	0.876	0.350

Table 27: Variance adjustments applied to the Langer Heinrich resource model

17.11 Resource Classification

Panels in the resource model were allocated an initial confidence category based on the number and location of samples used to estimate proportions and grade of each panel. The approach is based on the principle that larger numbers of samples, which are more evenly distributed throughout the search neighbourhood, will provide a more reliable estimate. The number of samples and the particular geographic configurations that may qualify the panel as Measured rather than Indicated or Inferred are essentially the domain of the Competent Person. The search parameters used to decide the initial classification of a panel resource in this study are:

- *Minimum number of samples found in the search neighbourhood.*
  For Measured and Indicated resources, this parameter is set to sixteen. For Inferred resources, a minimum of eight samples is required. This parameter ensures that the panel estimate is generated from a reasonable number of sample data.

- *Minimum number of spatial octants informed.*
  The space around the centre of a panel being estimated is divided into eight octants by the axial planes of the data search ellipsoid. This parameter ensures that the samples informing an estimate are relatively evenly spread around the panel and do not all come from one drill hole. For Measured and Indicated resources, at least four octants must contain at least one sample. For Inferred panels, at least two octants must contain data.

- *The distance to informing data.*
  The search radii define how far the kriging program may look in any direction to find samples to include in the estimation of resources in a panel. Panel dimensions and the sampling density in various directions usually influence the length of these radii. It is essential that the search radii be kept as short as possible while still achieving the degree of resolution required in the model. For Measured resources the east, north and vertical radii were set to 55, 55 and 4 metres respectively. For Indicated category, these radii were expanded by 30 per cent. For Inferred category the plan view search radii were expanded to 100 x 100 metres and the vertical search radius maintained at 5.2 metres.

17.12 Post-processing and Reclassification

The proportion of each panel lying below the basement interface was estimated using the triangulated basement surface and, at each cut-off grade, the tonnage and contained U_3O_8 estimated above cut-off was reduced by taking:

- Original ore tonnes x proportion above basement = final ore tonnes
- Original contained U_3O_8 x proportion above basement = final contained U_3O_8

17.13 Resource Estimates

Estimated resources above a series of cut-off grade are listed in *Table 28* to *Table 35*. Grade-tonnage curves for Measured and Indicated resources in Details 1, 2, 3 and 5 are shown in Figure 144 to Figure 147. Estimates have been trimmed to the current surface topography.

Figure 148 to Figure 171 show representative plan views of resource panels and informing data in Detail 1. In Figure 148 to Figure 151 resource panels are coloured by estimated mean U_3O_8 grade. Figure 152 to Figure 155 show panels coloured and scaled by recoverable proportion above 0.25kg/t cut-off and Figures Figure 156 to Figure 159 show panels coloured by resource confidence category.

Figures Figure 160 to Figure 171 show a series of cross-sections through Detail 1 with model panels displayed by mean U_3O_8 grade, recoverable proportion and confidence category as for the plan views.

cut-off	Measured			Indicated			Inferred		
kg/t	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8
0.10	18,334,218	0.54	9,984	8,597,146	0.41	3,486	7,381,664	0.35	2,614
0.20	13,955,534	0.67	9,400	5,779,232	0.53	3,069	4,788,501	0.47	2,237
0.25	**12,244,678**	**0.74**	**9,009**	**4,783,168**	**0.59**	**2,844**	**3,846,258**	**0.53**	**2,024**
0.30	10,727,963	0.80	8,587	3,963,037	0.66	2,614	3,172,949	0.58	1,838
0.35	9,436,945	0.87	8,164	3,283,682	0.73	2,393	2,343,044	0.67	1,568
0.40	8,335,424	0.93	7,748	2,762,314	0.79	2,195	1,908,912	0.74	1,403
0.45	7,359,334	1.00	7,328	2,325,697	0.86	2,009	1,529,418	0.81	1,241
0.50	6,488,408	1.07	6,911	1,963,228	0.93	1,834	1,224,238	0.90	1,096
0.55	5,716,426	1.14	6,500	1,676,675	1.00	1,681	996,460	0.98	975
0.60	5,026,727	1.21	6,098	1,418,975	1.08	1,533	789,228	1.08	854
0.65	4,425,368	1.29	5,720	1,206,211	1.16	1,398	631,762	1.19	755
0.70	3,942,340	1.37	5,391	1,055,199	1.23	1,295	548,204	1.27	699

Table 28: Estimated resources in Detail 1

cut-off	Measured			Indicated			Inferred		
kg/t	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8
0.10	9,473,327	0.40	3,817	10,929,255	0.35	3,865	16,698,406	0.44	7,295
0.20	6,044,810	0.55	3,311	6,543,227	0.49	3,218	10,499,741	0.61	6,387
0.25	**4,837,609**	**0.63**	**3,037**	**5,132,057**	**0.56**	**2,899**	**8,676,262**	**0.69**	**5,975**
0.30	3,968,971	0.70	2,796	4,142,655	0.63	2,626	7,389,125	0.76	5,621
0.35	3,257,587	0.79	2,562	3,335,232	0.71	2,358	6,278,870	0.84	5,256
0.40	2,652,481	0.88	2,333	2,705,218	0.78	2,121	5,369,712	0.91	4,911
0.45	2,218,085	0.97	2,146	2,241,814	0.86	1,923	4,697,734	0.98	4,624
0.50	1,895,048	1.05	1,992	1,884,888	0.93	1,751	4,164,005	1.05	4,369
0.55	1,639,930	1.13	1,855	1,606,999	1.00	1,604	3,773,669	1.10	4,162
0.60	1,425,549	1.21	1,729	1,402,411	1.06	1,484	3,447,737	1.15	3,974
0.65	1,257,271	1.29	1,621	1,225,928	1.12	1,372	3,146,050	1.20	3,782
0.70	1,128,819	1.36	1,533	1,085,724	1.18	1,276	2,867,571	1.25	3,596

Table 29: Estimated resources in Detail 2

cut-off	Measured			Indicated			Inferred		
kg/t	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8
0.10	7,354,596	0.31	2,310	3,250,484	0.27	861	7,503,976	0.31	2,320
0.20	4,932,459	0.39	1,945	1,577,395	0.39	621	3,587,606	0.49	1,761
0.25	**3,749,968**	**0.45**	**1,679**	**1,103,043**	**0.47**	**514**	**2,712,215**	**0.58**	**1,566**
0.30	2,750,030	0.51	1,405	787,163	0.54	428	2,201,507	0.65	1,426
0.35	2,011,213	0.58	1,166	581,023	0.62	361	1,841,136	0.71	1,308
0.40	1,501,102	0.65	976	443,880	0.70	310	1,552,223	0.77	1,200
0.45	1,147,971	0.72	826	349,022	0.77	269	1,315,387	0.84	1,099
0.50	898,937	0.79	708	280,404	0.84	237	1,117,628	0.90	1,004
0.55	716,011	0.85	611	230,164	0.91	210	957,853	0.96	920
0.60	578,322	0.92	532	191,487	0.98	188	826,200	1.02	844
0.65	473,996	0.99	467	161,508	1.04	169	715,529	1.08	774
0.70	392,240	1.05	411	137,856	1.11	153	621,684	1.14	710

Table 30: Estimated resources in Detail 3

cut-off	Measured			Indicated			Inferred		
kg/t	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8
0.10	-	-	-	-	-	-	19,834,483	0.29	5,820
0.20	-	-	-	-	-	-	12,142,111	0.39	4,688
0.25	**-**	**-**	**-**	**-**	**-**	**-**	**9,264,862**	**0.44**	**4,043**
0.30	-	-	-	-	-	-	6,982,937	0.49	3,417
0.35	-	-	-	-	-	-	5,218,942	0.55	2,845
0.40	-	-	-	-	-	-	3,901,295	0.60	2,352
0.45	-	-	-	-	-	-	2,938,103	0.66	1,944
0.50	-	-	-	-	-	-	2,232,611	0.72	1,608
0.55	-	-	-	-	-	-	1,713,649	0.78	1,335
0.60	-	-	-	-	-	-	1,329,898	0.84	1,114
0.65	-	-	-	-	-	-	1,038,971	0.90	931
0.70	-	-	-	-	-	-	817,486	0.95	780

Table 31: Estimated resources in Detail 4

cut-off	Measured			Indicated			Inferred		
kg/t	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8
0.10	4,295,231	0.30	1,308	8,153,684	0.30	2,450	15,807,371	0.27	4,277
0.20	2,494,315	0.42	1,045	4,553,185	0.42	1,928	7,557,936	0.41	3,105
0.25	**1,886,599**	**0.48**	**909**	**3,438,038**	**0.49**	**1,679**	**5,546,862**	**0.48**	**2,655**
0.30	1,433,860	0.55	785	2,627,715	0.55	1,456	4,171,843	0.55	2,277
0.35	1,097,756	0.62	676	2,033,870	0.62	1,264	3,201,957	0.61	1,961
0.40	849,115	0.69	583	1,593,282	0.69	1,099	2,497,974	0.68	1,697
0.45	664,493	0.76	504	1,264,587	0.76	959	1,973,447	0.75	1,473
0.50	527,807	0.83	439	1,014,967	0.83	840	1,572,380	0.81	1,281
0.55	425,543	0.91	385	823,724	0.90	739	1,263,152	0.88	1,117
0.60	347,889	0.98	340	675,236	0.97	653	1,023,803	0.95	977
0.65	287,528	1.05	302	558,176	1.04	579	837,268	1.03	859
0.70	241,281	1.12	271	466,461	1.11	517	689,862	1.10	758

Table 32: Estimated resources in Detail 5

cut-off	Measured			Indicated			Inferred		
kg/t	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8
0.10	-	-	-	-	-	-	6,832,210	0.31	2,112
0.20	-	-	-	-	-	-	3,920,095	0.43	1,692
0.25	**-**	**-**	**-**	**-**	**-**	**-**	**3,047,339**	**0.49**	**1,496**
0.30	-	-	-	-	-	-	2,393,752	0.55	1,317
0.35	-	-	-	-	-	-	1,892,172	0.61	1,154
0.40	-	-	-	-	-	-	1,498,988	0.67	1,007
0.45	-	-	-	-	-	-	1,190,446	0.74	875
0.50	-	-	-	-	-	-	945,114	0.80	759
0.55	-	-	-	-	-	-	751,254	0.87	656
0.60	-	-	-	-	-	-	604,874	0.94	572
0.65	-	-	-	-	-	-	498,217	1.01	504
0.70	-	-	-	-	-	-	416,730	1.08	449

Table 33: Estimated resources in Detail 6

cut-off	Measured			Indicated			Inferred		
kg/t	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8
0.10	-	-	-	-	-	-	23,143,412	0.42	9,708
0.20	-	-	-	-	-	-	13,132,460	0.63	8,247
0.25	**-**	**-**	**-**	**-**	**-**	**-**	**10,303,694**	**0.74**	**7,601**
0.30	-	-	-	-	-	-	8,388,039	0.84	7,066
0.35	-	-	-	-	-	-	7,059,843	0.94	6,625
0.40	-	-	-	-	-	-	5,989,741	1.04	6,220
0.45	-	-	-	-	-	-	5,134,560	1.14	5,849
0.50	-	-	-	-	-	-	4,509,305	1.23	5,544
0.55	-	-	-	-	-	-	4,004,067	1.32	5,273
0.60	-	-	-	-	-	-	3,621,951	1.39	5,048
0.65	-	-	-	-	-	-	3,249,279	1.48	4,806
0.70	-	-	-	-	-	-	2,871,876	1.58	4,551

Table 34: Estimated resources in Detail 7

cut-off	Measured			Indicated			Inferred		
kg/t	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8
0.10	39,457,372	0.44	17,419	30,930,569	0.34	10,663	97,201,522	0.35	34,146
0.20	27,427,118	0.57	15,701	18,453,039	0.48	8,835	55,628,450	0.51	28,118
0.25	**22,718,853**	**0.64**	**14,634**	**14,456,305**	**0.55**	**7,936**	**43,397,492**	**0.58**	**25,360**
0.30	18,880,824	0.72	13,574	11,520,570	0.62	7,124	34,700,152	0.66	22,961
0.35	15,803,502	0.80	12,568	9,233,807	0.69	6,377	27,835,964	0.74	20,718
0.40	13,338,122	0.87	11,640	7,504,693	0.76	5,725	22,718,844	0.83	18,791
0.45	11,389,882	0.95	10,804	6,181,120	0.83	5,160	18,779,095	0.91	17,105
0.50	9,810,200	1.02	10,049	5,143,487	0.91	4,662	15,765,281	0.99	15,660
0.55	8,497,910	1.10	9,352	4,337,562	0.98	4,234	13,460,104	1.07	14,438
0.60	7,378,487	1.18	8,700	3,688,109	1.05	3,858	11,643,691	1.15	13,382
0.65	6,444,162	1.26	8,110	3,151,823	1.12	3,517	10,117,076	1.23	12,411
0.70	5,704,680	1.33	7,607	2,745,240	1.18	3,241	8,833,413	1.31	11,542

Table 35: Total Langer Heinrich estimated resources

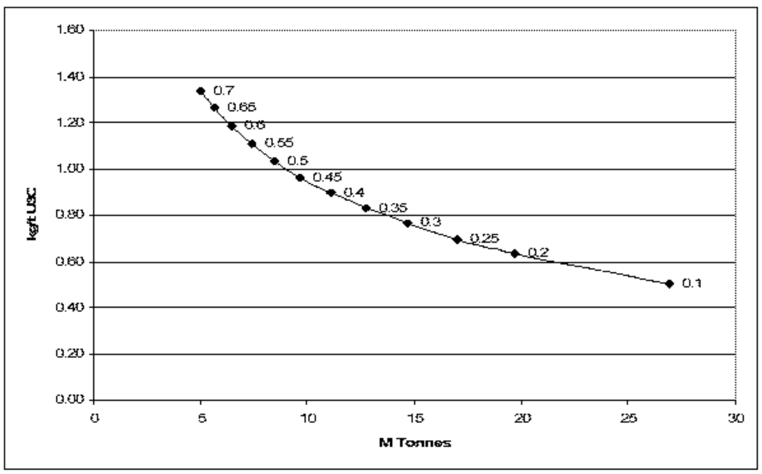


Figure 144: Grade-tonnage curve for measured and indicated resources, Detail 1

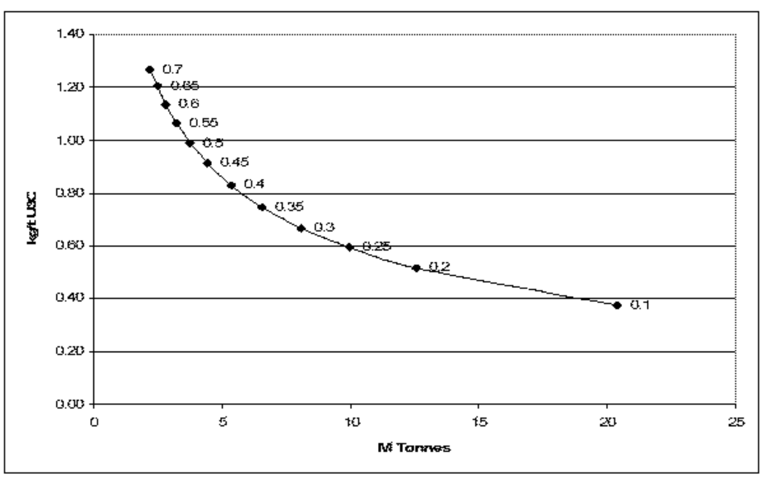


Figure 145: Grade-tonnage curve for measured and indicated resources, Detail 2

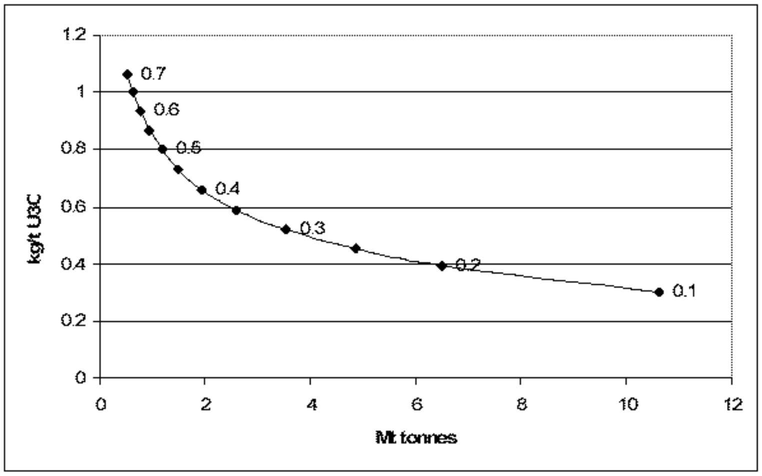


Figure 146: Grade-tonnage curve for measured and indicated resources, Detail 3

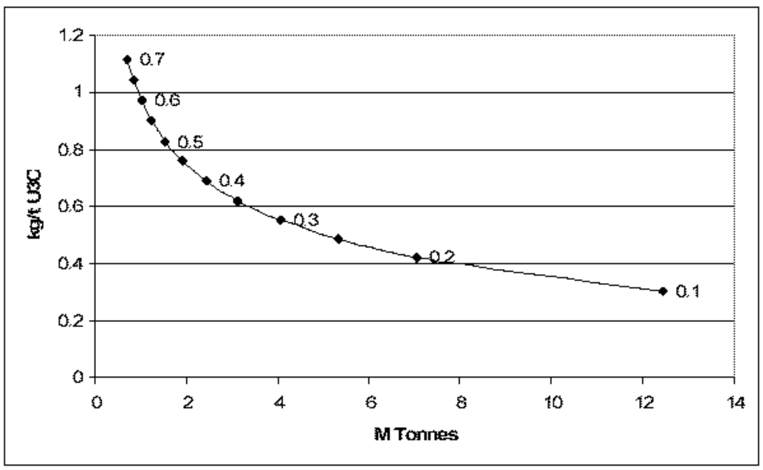


Figure 147: Grade-tonnage curve for measured and indicated resources, Detail 5

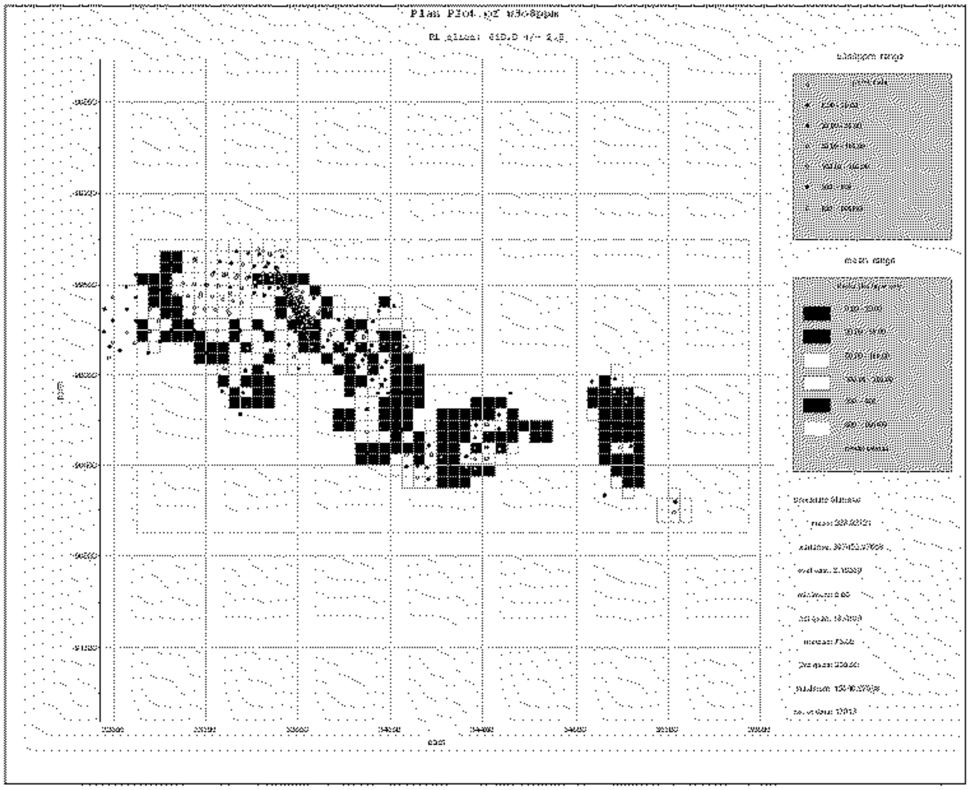


Figure 148: Detail 1 608-612RL, mean U_3O_8 grades

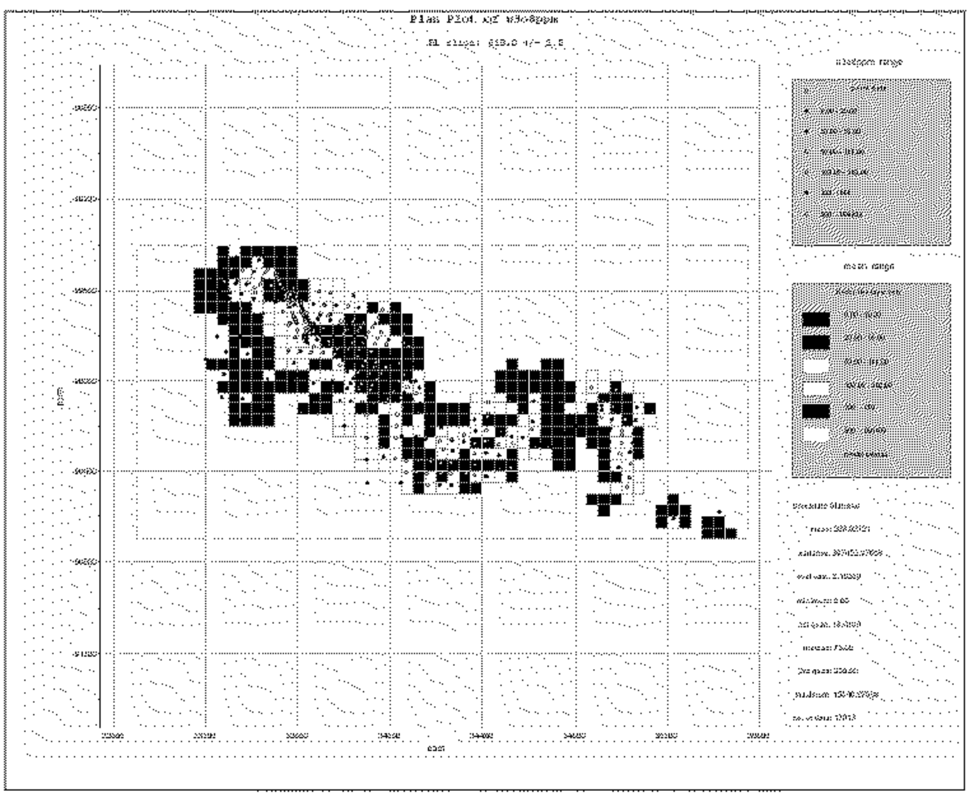


Figure 149: Detail 1 616-620RL, mean U_3O_8 grades

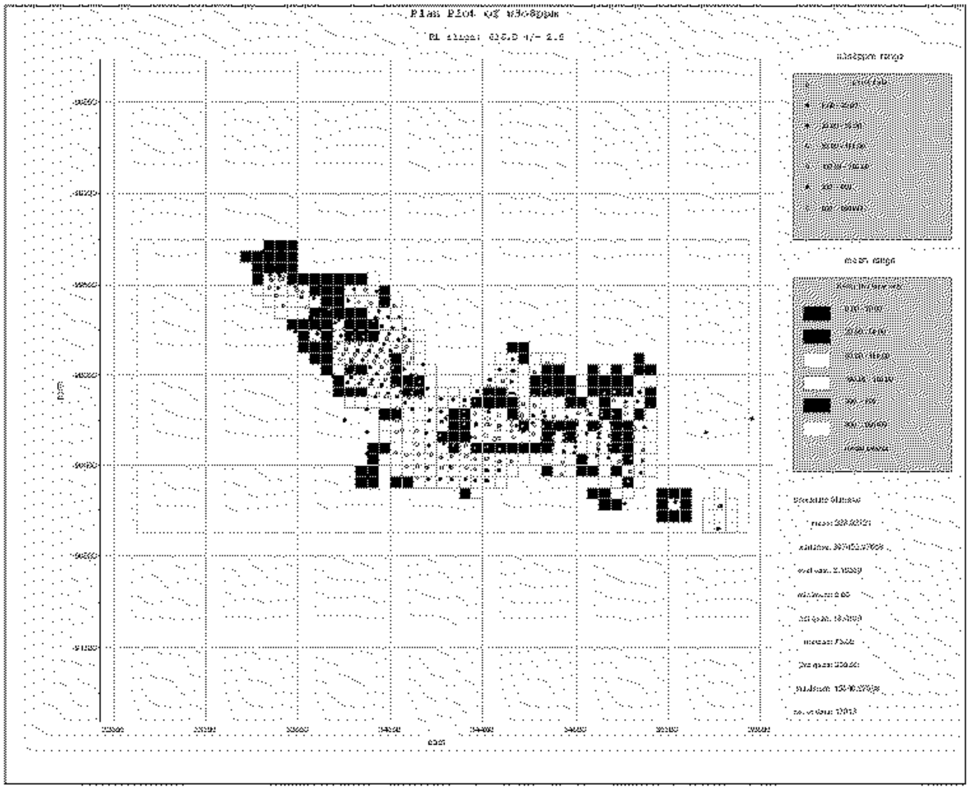


Figure 150: Detail 1 624-628RL, mean U_3O_8 grades

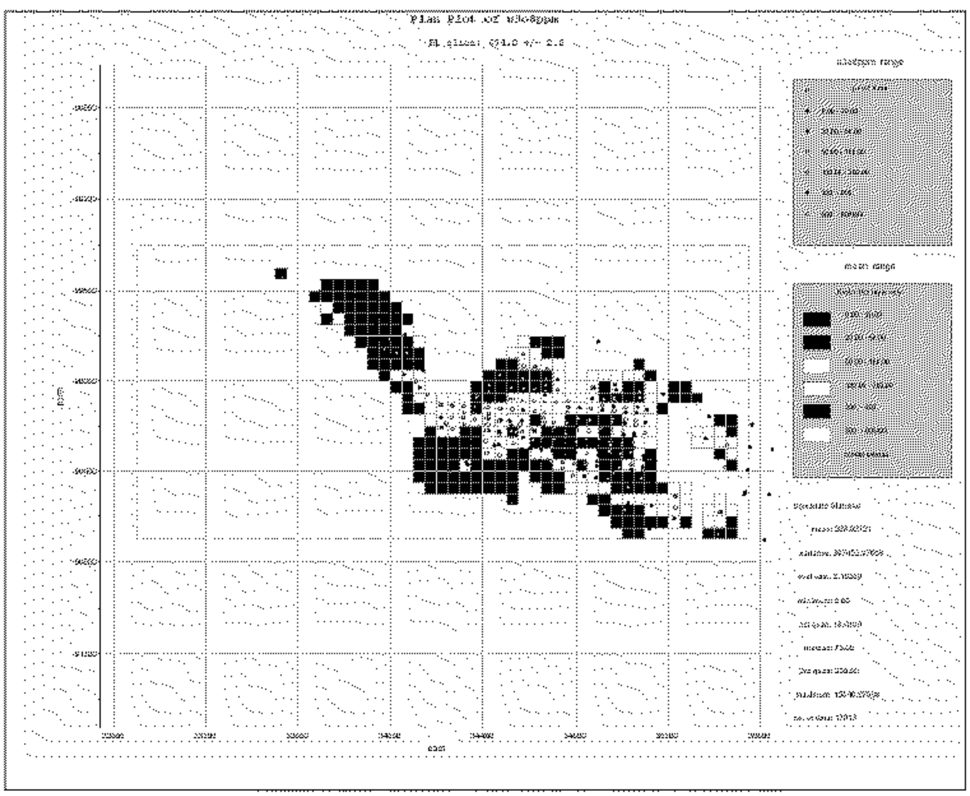


Figure 151: Detail 1 632-636RL, mean U_3O_8 grades

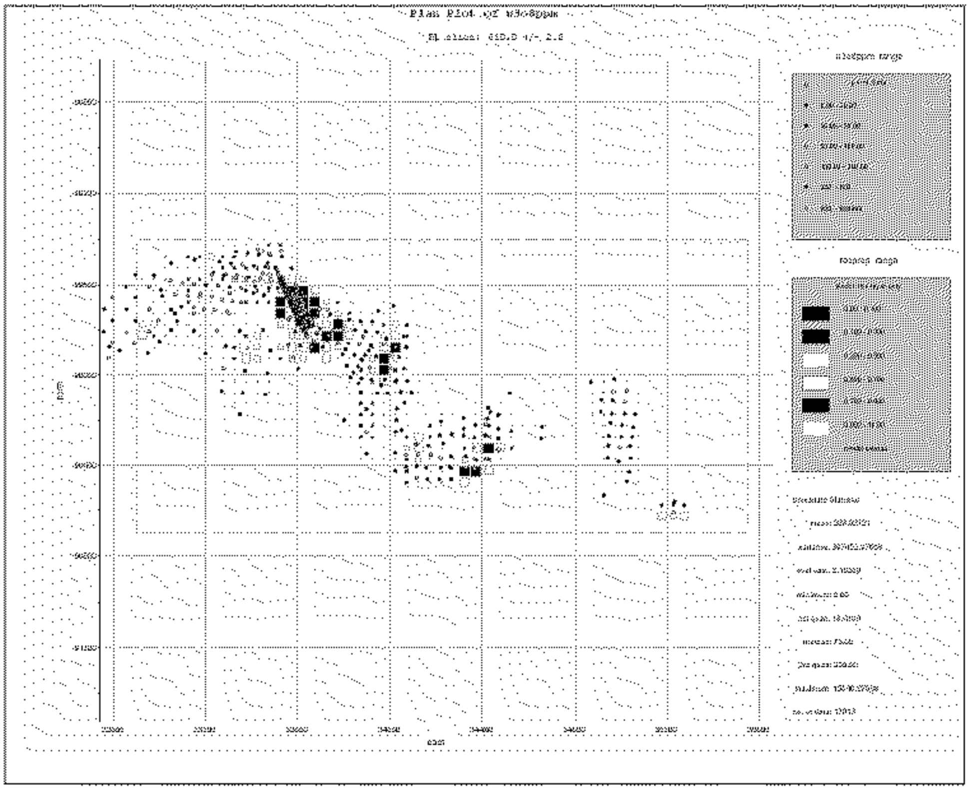


Figure 152: Detail 1 608-612RL, recoverable proportions at 0.25kg/t U_3O_8 cut-off

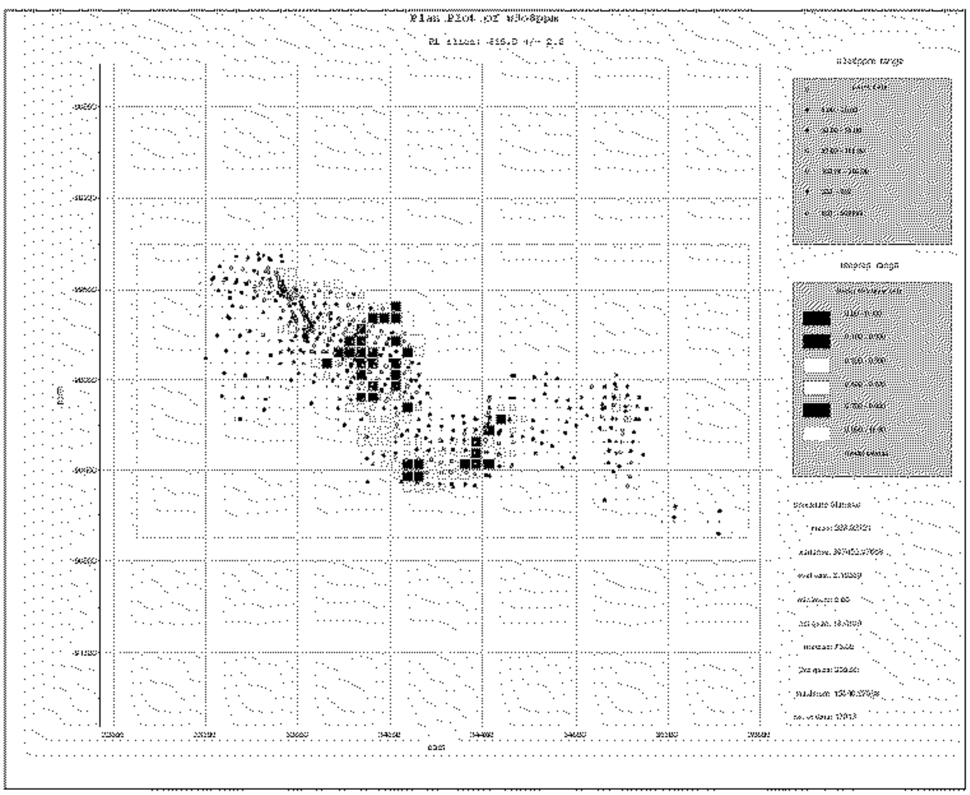


Figure 153: Detail 1 616-620RL, recoverable proportions at 0.25kg/t U_3O_8 cut-off

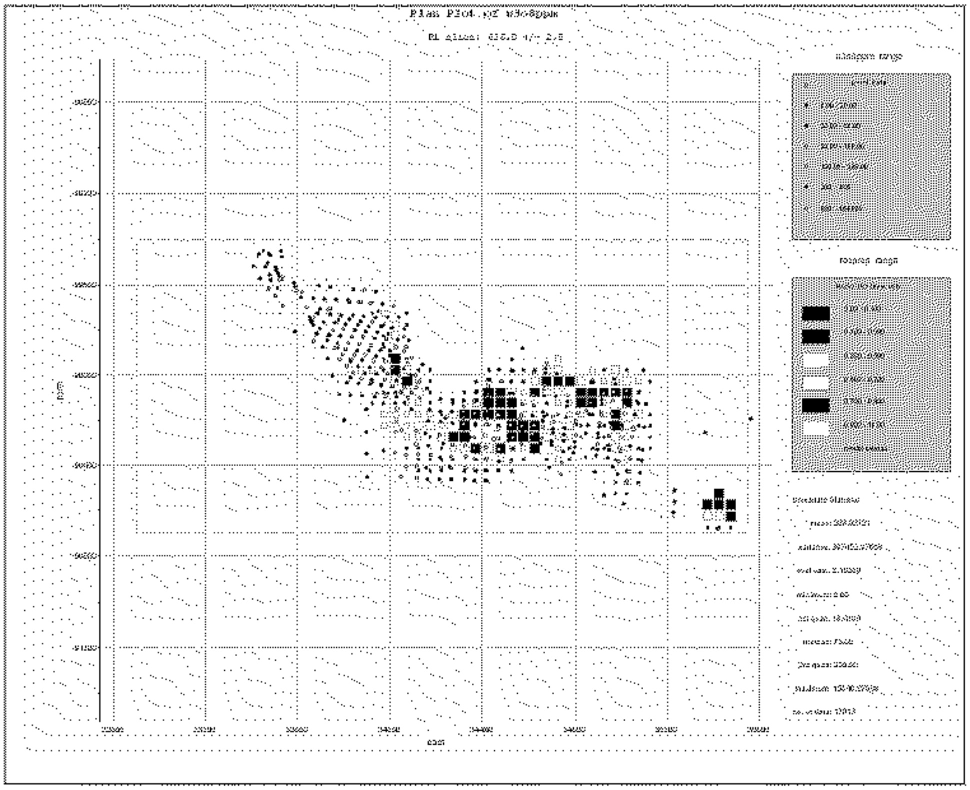


Figure 154: Detail 1 624-628RL, recoverable proportions at 0.25kg/t U_3O_8 cut-off

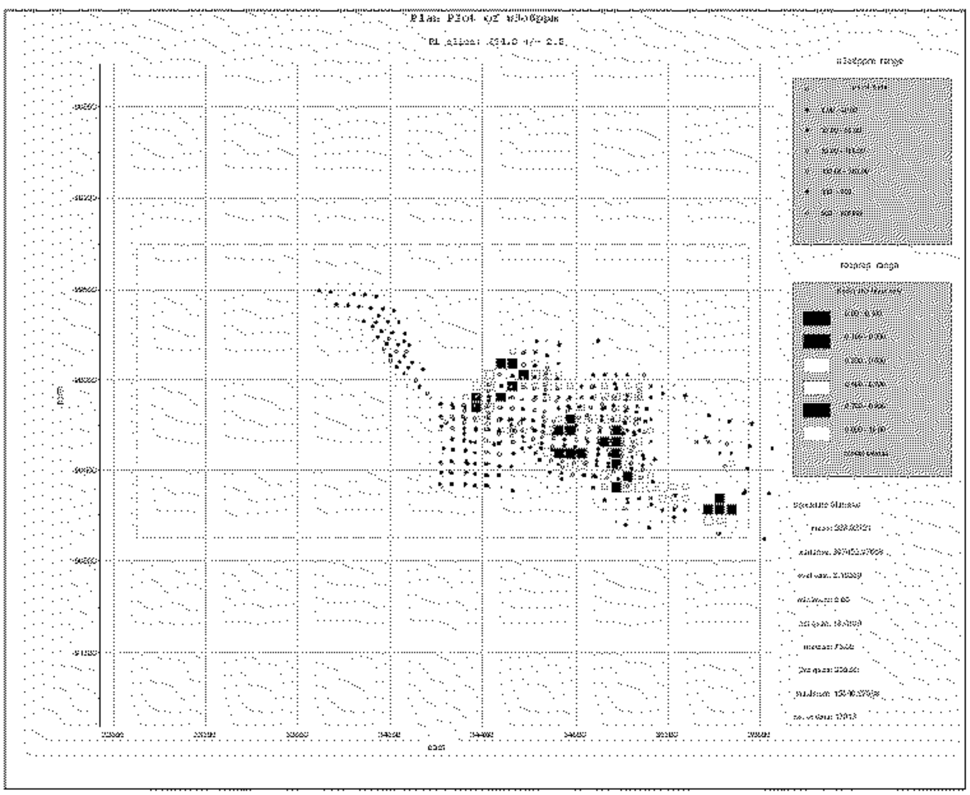


Figure 155: Detail 1 632-636RL, recoverable proportions at 0.25kg/t U_3O_8 cut-off

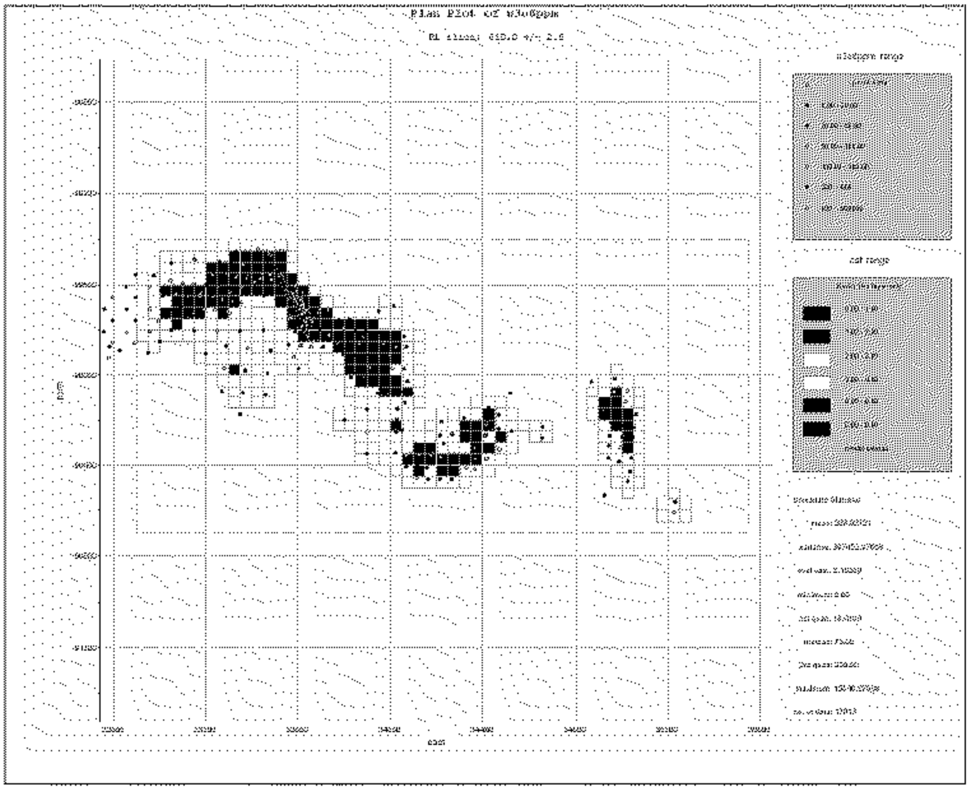


Figure 156: Detail 1 608-612RL, panel confidence categories

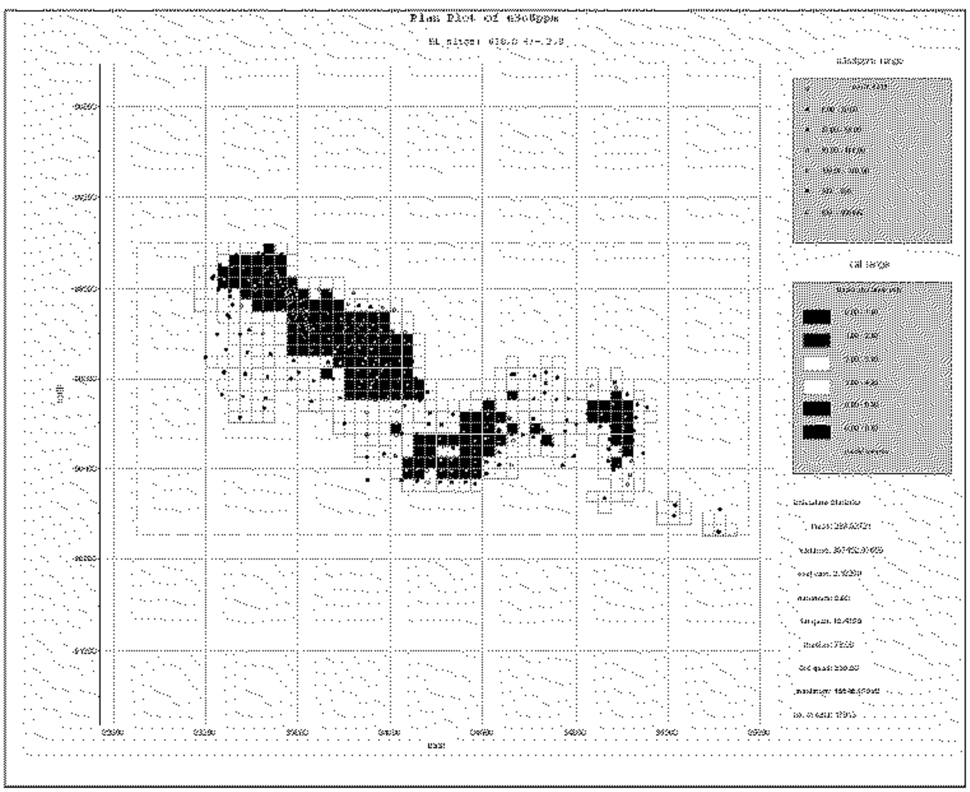


Figure 157: Detail 1 616-620RL, panel confidence categories

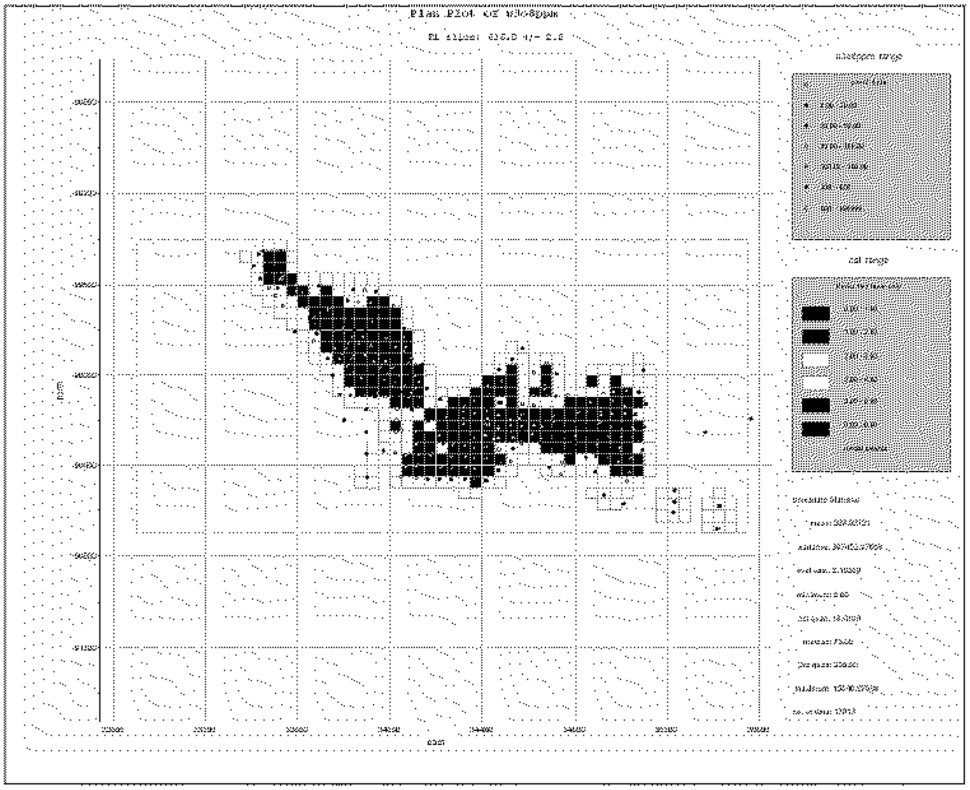


Figure 158: Detail 1 624-628RL, panel confidence categories

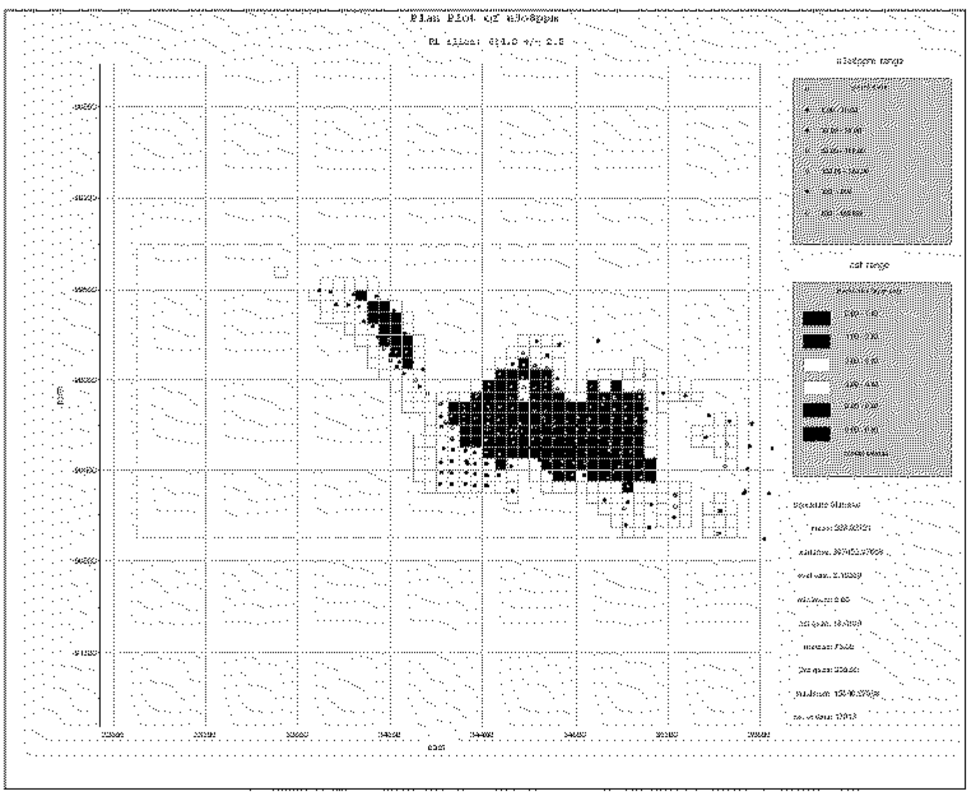


Figure 159: Detail 1 632-636RL, panel confidence categories

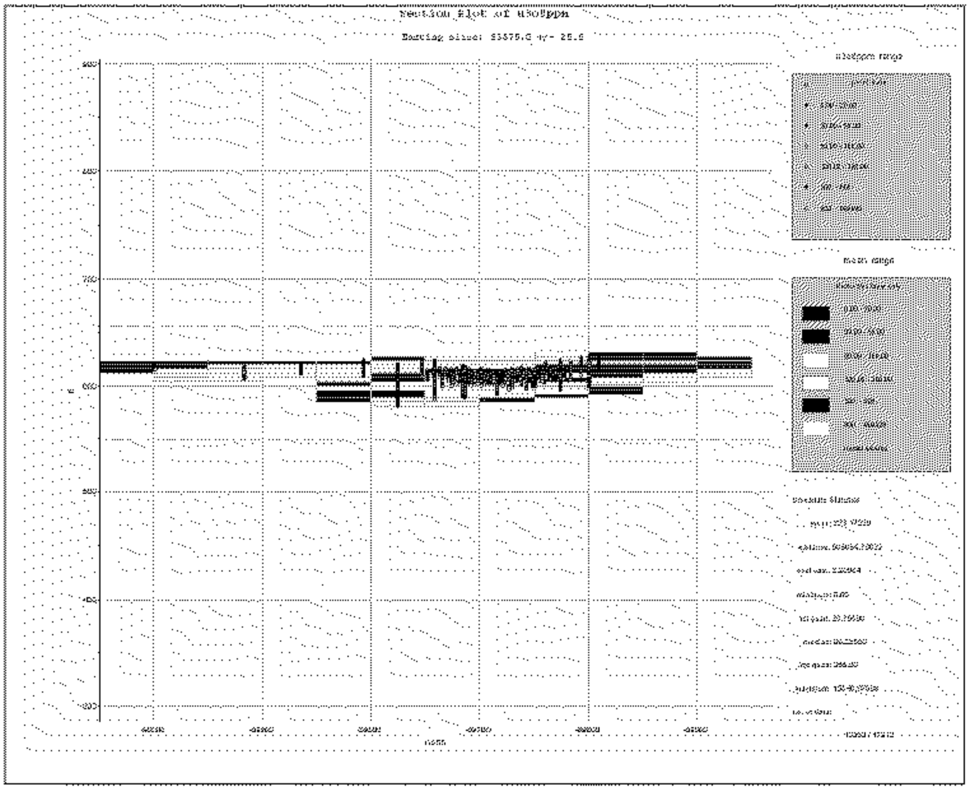


Figure 160: Detail 1 section 33575E, mean U_3O_8 grades

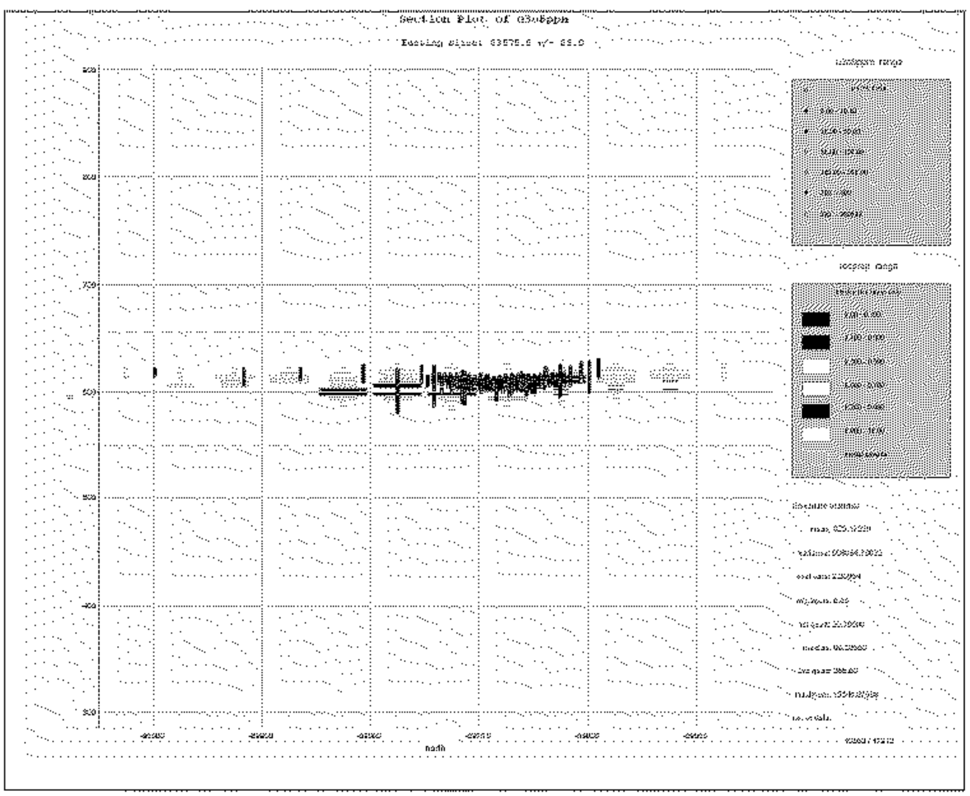


Figure 161: Detail 1 section 33575E, recoverable proportions at 0.25kg/t U_3O_8 cut-off

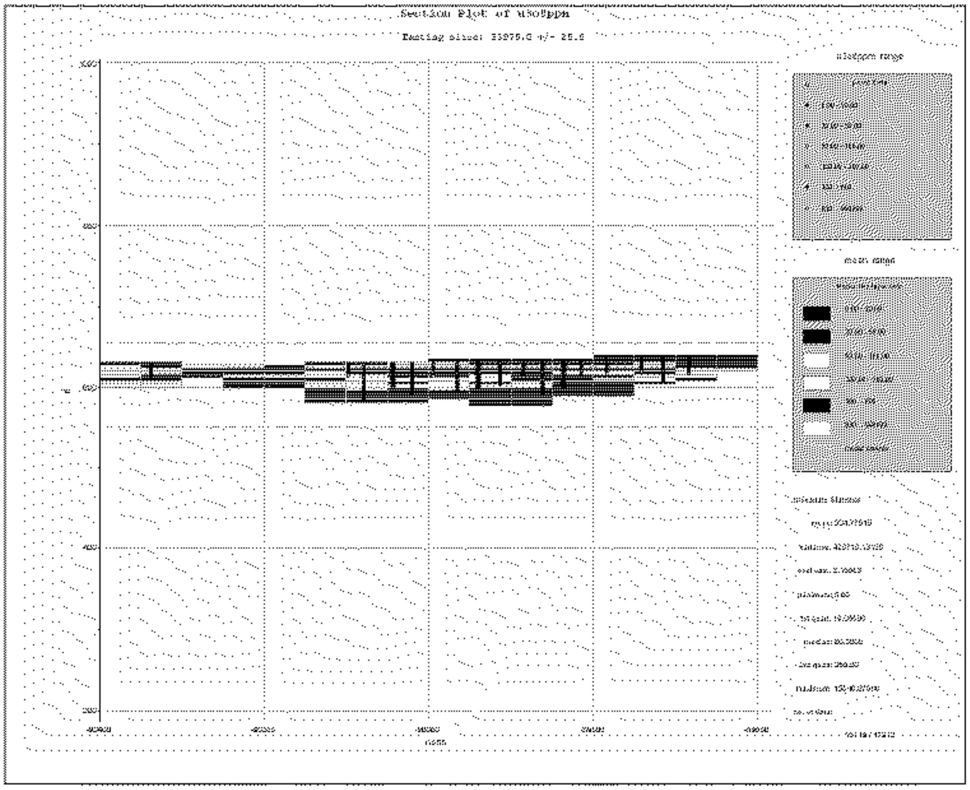


Figure 162: Detail 1 section 33975E, mean U_3O_8 grades

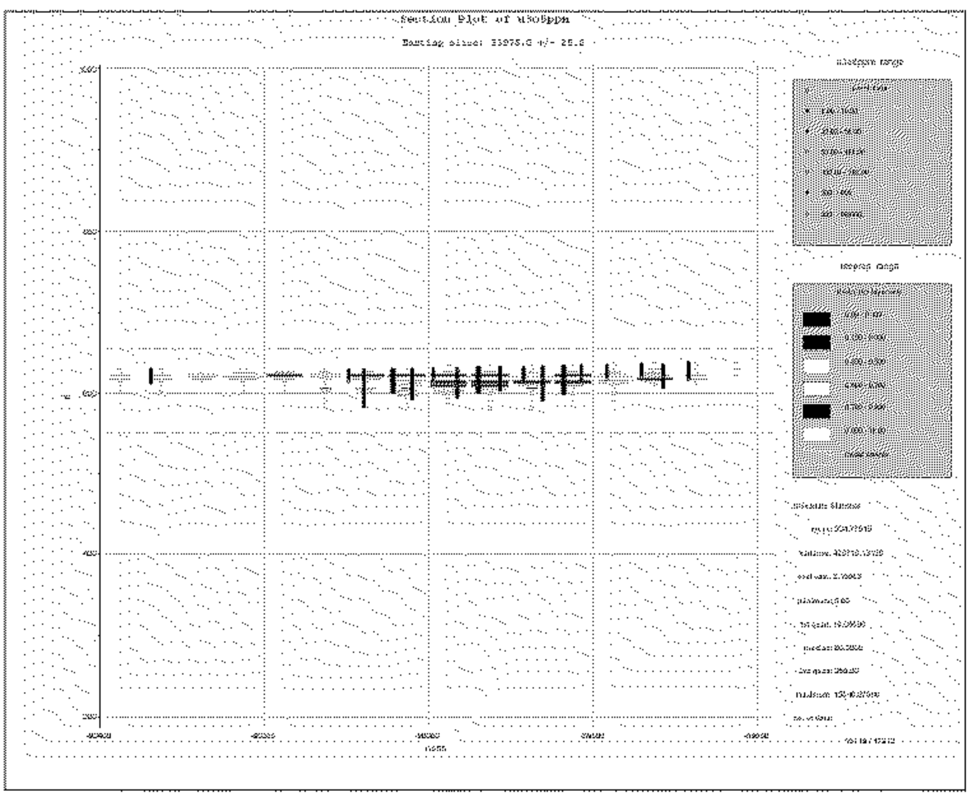


Figure 163: Detail 1 section 33975E, recoverable proportions at 0.25kg/t U_3O_8 cut-off

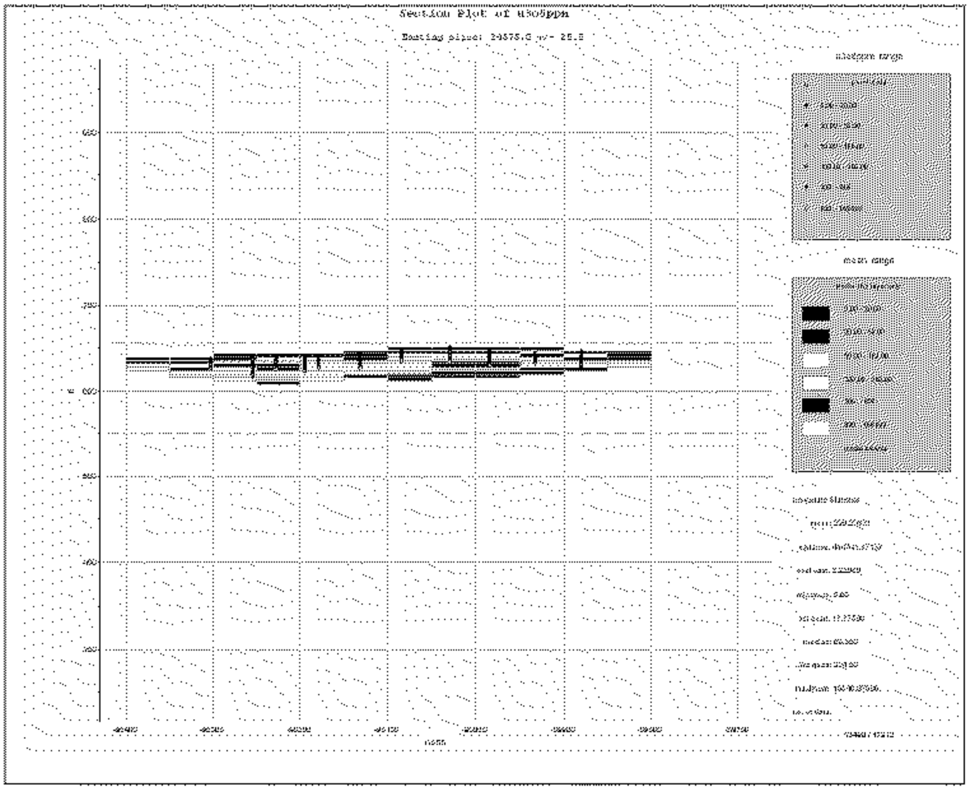


Figure 164: Detail 1 section 34575E, mean U_3O_8 grades

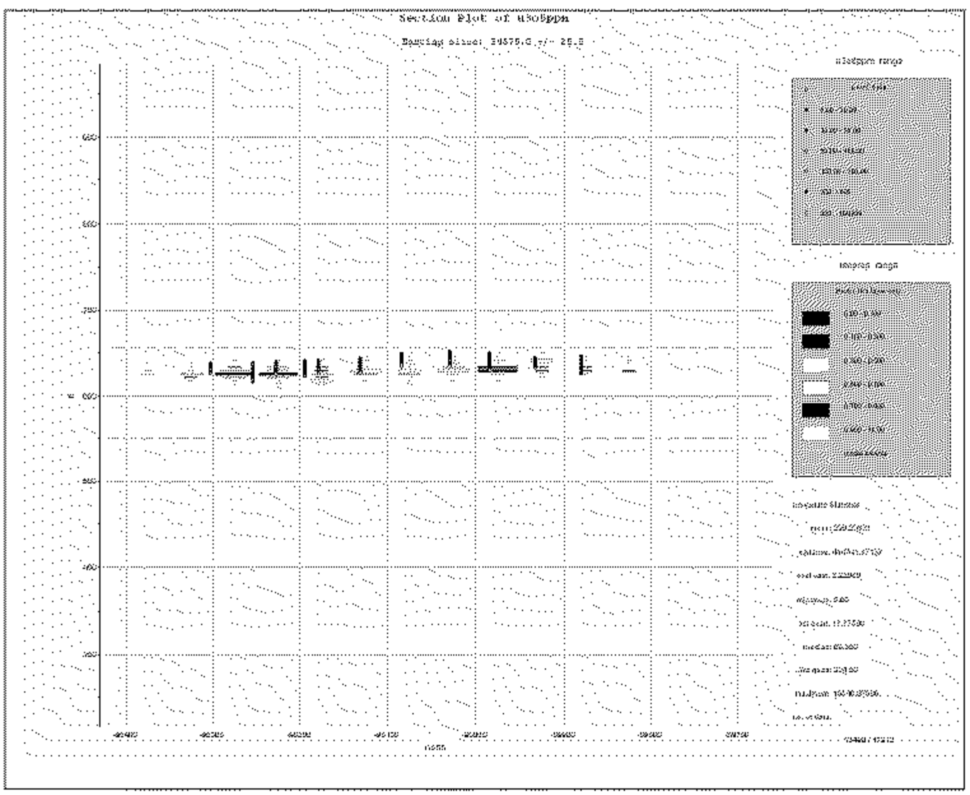


Figure 165: Detail 1 section 34575E, recoverable proportions at 0.25kg/t U_3O_8 cut-off

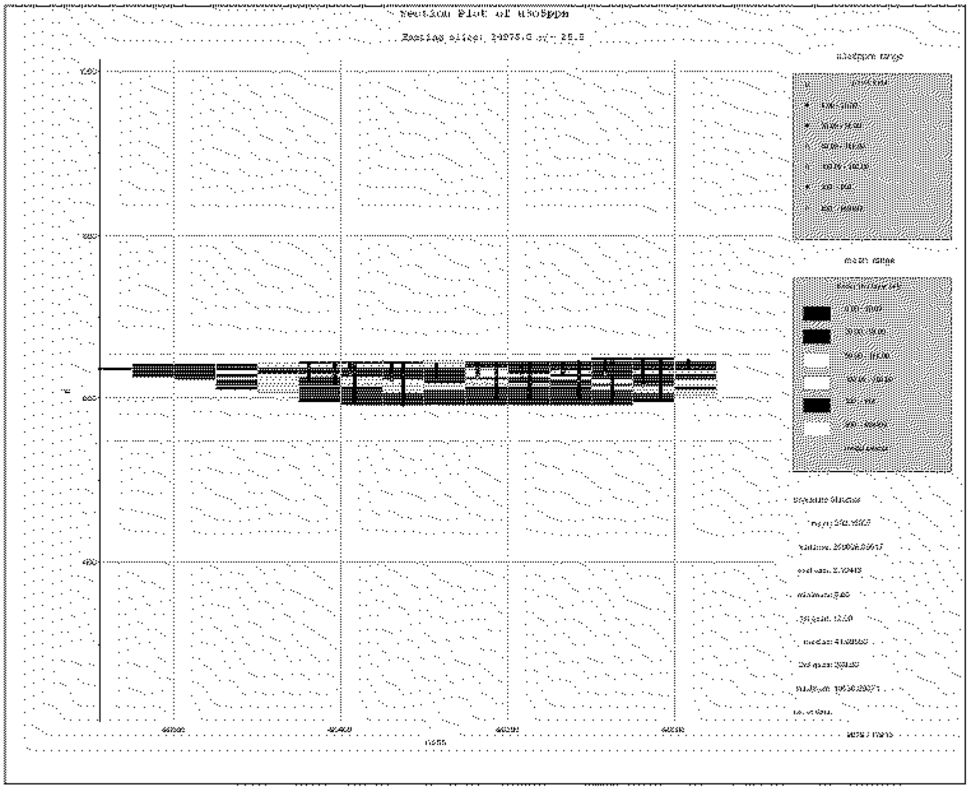


Figure 166: Detail 1 section 34975E, mean U_3O_8 grades

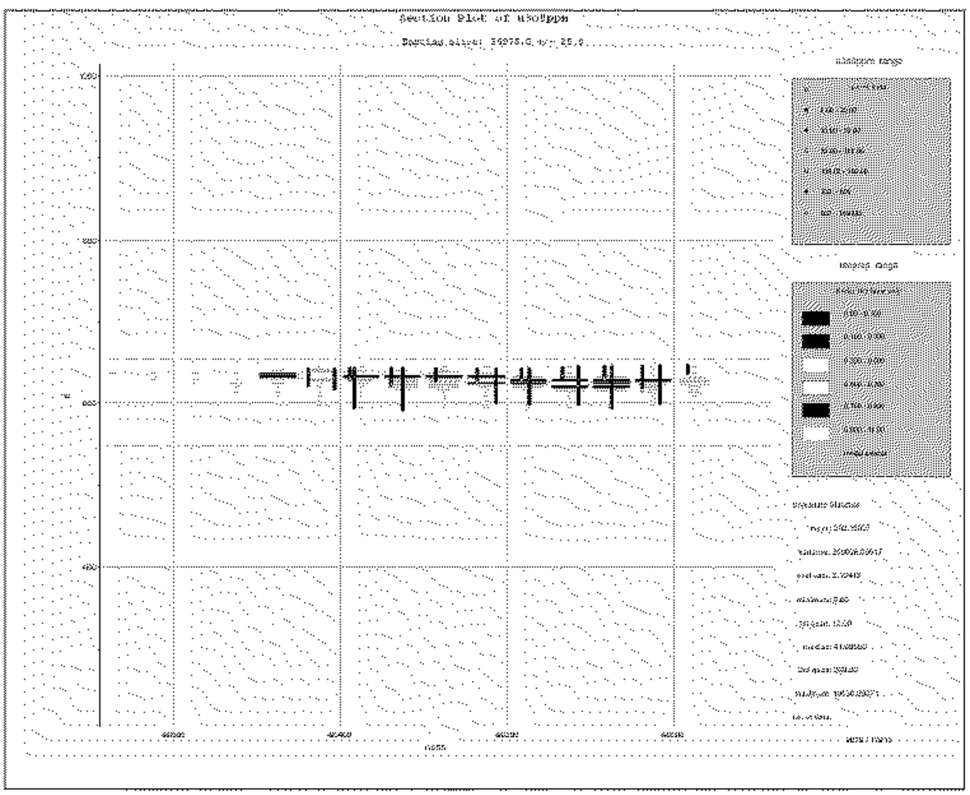


Figure 167: Detail 1 section 34975E, recoverable proportions at 0.25kg/t U_3O_8 cut-off

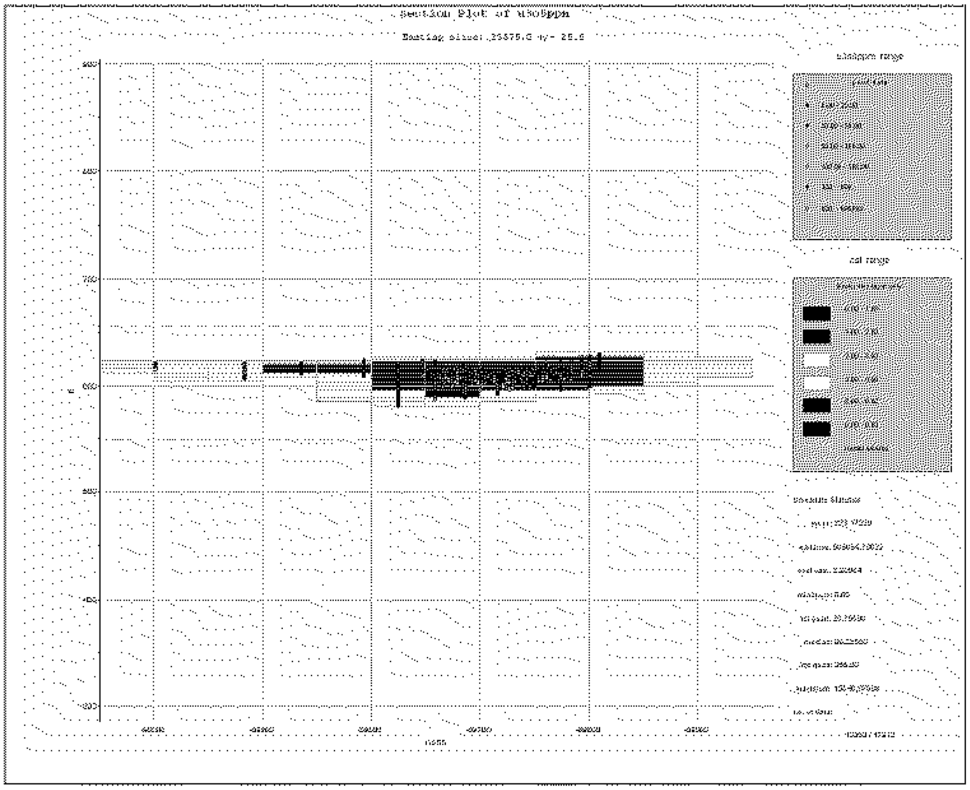


Figure 168: Detail 1 section 33575E, panel confidence categories

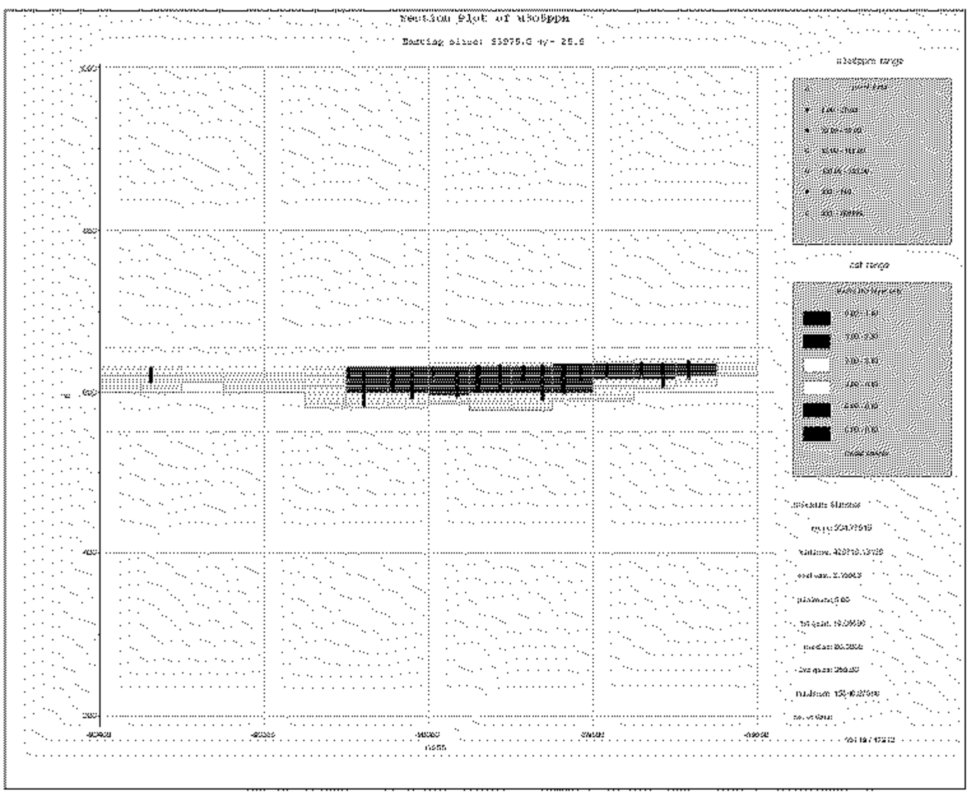


Figure 169: Detail 1 section 33975E, panel confidence categories

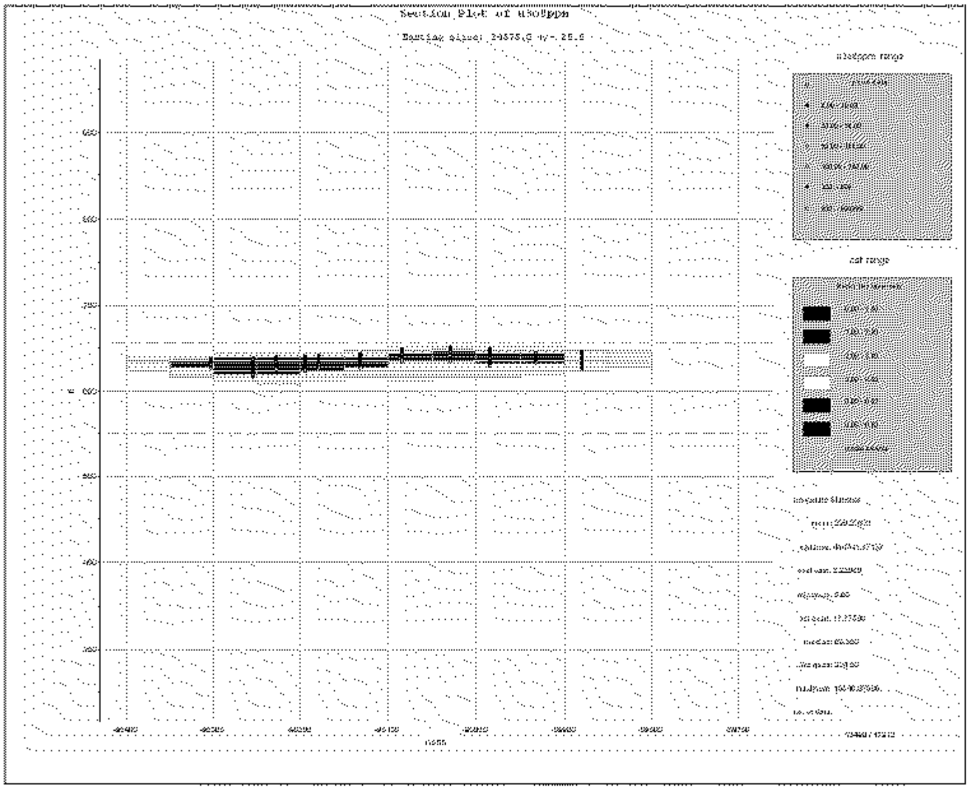


Figure 170: Detail 1 section 34575E, panel confidence categories

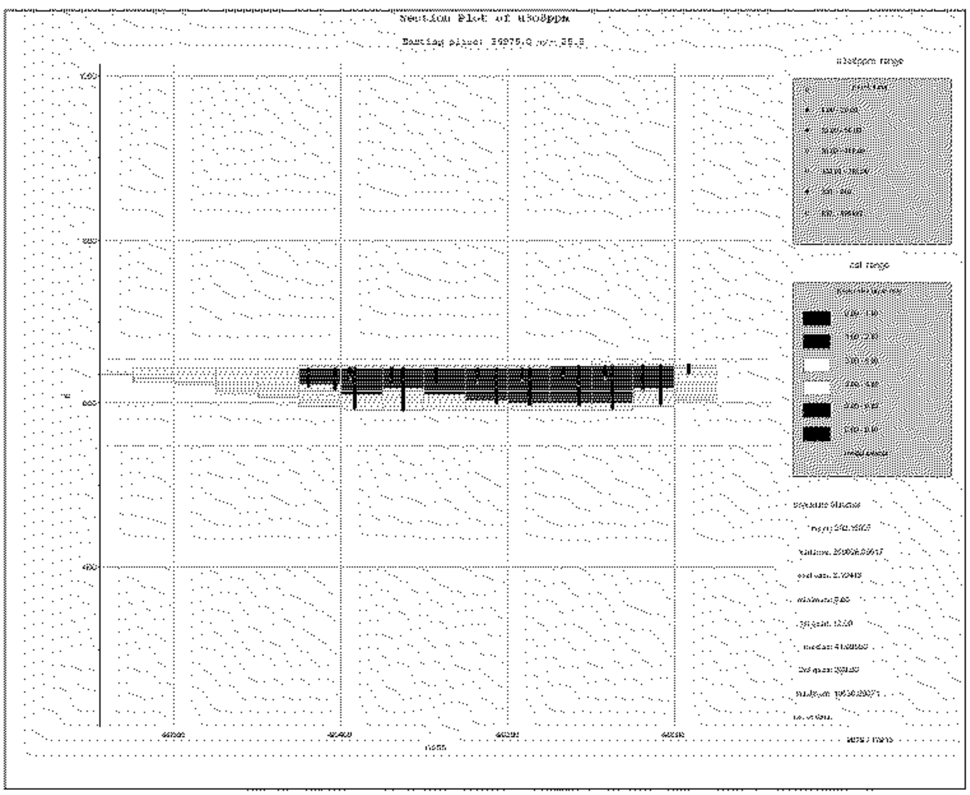


Figure 171: Detail 1 section 34975E, panel confidence categories

17.14 Other factors affecting resources

See NI43-101, Langer Heinrich, Namibia, Independent Technical Report, Resource and Reserve Estimation, 7th June 2005 for discussion regarding mining operations in association with economic and metallurgical factors which may affect the resources reported above. In the tables above, the 250ppm cut off grade has been highlighted as this is the minimum cut off grade to be used during mining operations, mineralisation is to be separated into low (250-400ppm), medium (400-650ppm) and high (>650ppm) grade stockpiles on mining.

To date mining activities on the site have been limited to topsoil removal and stockpiling along with waste stripping above the mineralisation to allow for the construction of infrastructure, ROM pads, roads and bunds. The pre-existing GENCOR trial mining stockpiles have been removed to the newly constructed ROM pad to provide a feed source for the commissioning of the plant.

18 Other Relevant Data and Information

18.1 Reconciliation to Gencor Trial Mining Results

The MIK model was cut to the surveyed pit and trench surfaces supplied, with top and bottom elevation limits of 624RL and 611RL applied to the pit to reflect the interval over which Gencor mining records apply. Tonnes and grades reporting from the model are compared to those from Gencor's mining records (Anon., 1980; Fletcher & Kuschke, 1979) in *Table 36* and *Table 37*. The conditional statistics of all drill samples within the two trial mining volumes are also listed, with tonnages reporting above cut-offs based on sample counts assuming no clustering of the sample data.

	Sample histogram			MIK model			Gencor		
cut-off	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8
0	236743	0.52	122.6	236743	0.56	133.3	236743*	0.35**	83.7
0.1	183640	0.65	119.7	179994	0.74	133.0	99486	0.84	83.4
0.3	118658	0.91	107.6	126106	0.98	123.0	80145	0.99	79.4
0.5	79296	1.16	92.1	87024	1.24	107.5	50014	1.35	67.6
0.75	48522	1.51	73.2	57695	1.55	89.6	27659	1.93	53.5
1	30487	1.88	57.4	39705	1.86	74.0	15754	2.75	43.4

Table 36: Comparison of estimates and material mined from mega-trench

* Calculated from total trench volume and 2.1g/cc bulk density

** Assumes average grade of 50ppm for material below 100ppm U_3O_8

	Sample histogram			MIK model			Gencor		
cut-off	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8
0	124207	0.91	112.5	124207	0.74	91.8	124207*	0.78**	96.7
0.1	101825	1.09	111.4	109051	0.84	91.4	77601	1.22	94.4
0.3	75974	1.40	106.4	83970	1.03	86.4	55155	1.60	88.1
0.5	54538	1.80	98.1	60962	1.27	77.3	-	-	-
0.75	39091	2.26	88.5	40500	1.60	64.7	-	-	-
1	28687	2.77	79.6	28450	1.91	54.3	-	-	-

Table 37: Comparison of estimates and material mined from trial pit

* Calculated from total pit volume between 624RL and 611RL and 2.1g/cc bulk density

** Assumes average grade of 50ppm for material below 100ppm U_3O_8

Clearly the recoverable tonnages and grades predicted by the MIK model do not match recorded production very well. It appears likely there are two reasons for this:

- Trial mining was centred on an area of abnormally high-grade mineralisation. Although indicator data from close-spaced drilling in the mega-trench and trial pit areas were used to inform the MIK model, the U_3O_8 grades in close-spaced drill holes were not used to calculate the conditional statistics that inform grade estimates in the model because of the effect of data clustering on those statistics. The MIK estimates thus represent a more "average" view of the resources in Detail 1, i.e., the precision of local estimates has been sacrificed for accuracy of global estimates.

- Gencor's mining has employed much greater selectivity than is considered achievable in the scale of operation being considered by LHU. Figure 172 and Figure 173 shows the grade-tonnage curves deriving from the figures in *Table 36* and *Table 37*. Gencor have mined lower tonnages at far higher average grades than would be predicted from interrogation of the histograms of sample grades. Because of the Volume-Variance Effect, the converse would normally be true, in sampling mineralisation by truck-loads rather than drill hole samples one would expect a higher tonnage at lower average grade to report as ore above any particular cut-off grade. Gencor have achieved very selective mining.

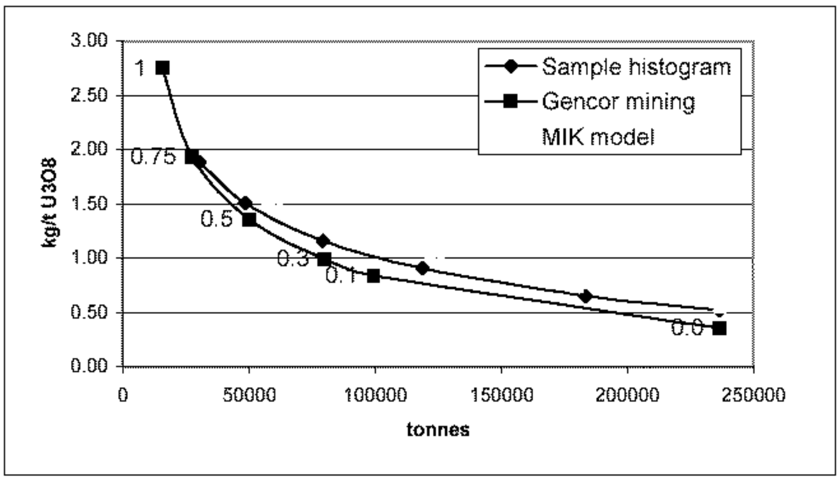


Figure 172: Grade-tonnage curves, mega-trench

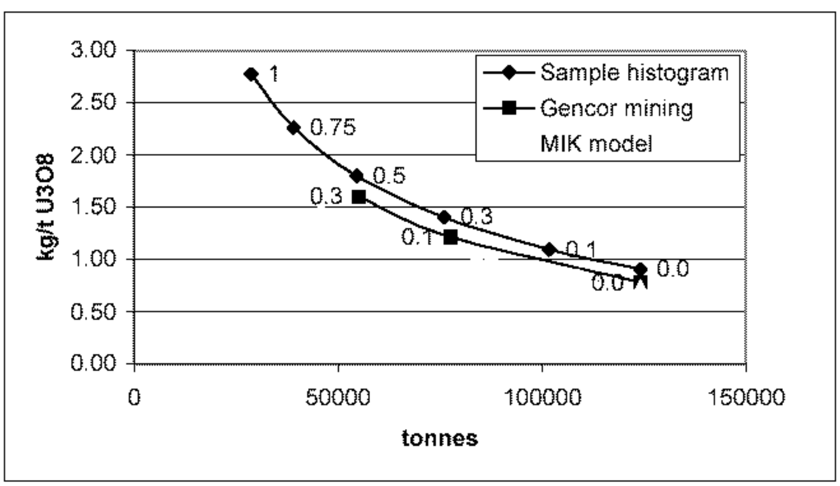


Figure 173: Grade-tonnage curves, trial pit

It may be concluded that Gencor's mining records are of little use in assessing the reliability of the MIK estimates but they do indicate that highly selective separation of ore and waste can be achieved in Langer Heinrich mineralisation. As there was no additional drilling in 2005 or 2006 that influenced the resource estimations in the area of the Gencor trial mining no comparison to the current resource is included.

18.2 Comparison to previous resource estimates

The drilling campaigns in 2005 and 2006 were primarily targeted at increasing resource confidence by infilling to 50m x 50m in areas where there was historical drilling at 100m x 100m or greater and extending the resource in areas which had been poorly defined by previous drilling.

cut-off	Measured			Indicated			Inferred		
kg/t	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8
0.10	29,138,999	0.48	13,912	19,666,192	0.40	7,912	49,443,941	0.42	20,951
0.20	21,548,719	0.59	12,781	13,317,770	0.52	6,972	33,843,382	0.55	18,645
0.25	18,175,810	0.66	12,030	10,915,440	0.59	6,440	27,570,707	0.63	17,262
0.30	15,241,477	0.74	11,208	9,007,375	0.66	5,900	22,015,781	0.71	15,703
0.35	12,807,931	0.81	10,432	7,488,928	0.72	5,417	17,274,929	0.82	14,212
0.40	10,943,806	0.89	9,755	6,252,425	0.79	4,950	14,168,228	0.92	13,016
0.45	9,444,854	0.96	9,103	5,112,712	0.88	4,475	11,667,096	1.02	11,947
0.50	8,232,632	1.04	8,532	4,231,227	0.96	4,048	9,773,623	1.14	11,102
0.55	7,231,954	1.11	8,008	3,551,533	1.04	3,698	7,993,114	1.27	10,153
0.60	6,391,335	1.18	7,536	3,075,204	1.11	3,427	6,681,224	1.40	9,371
0.65	5,677,801	1.25	7,087	2,692,300	1.18	3,184	5,882,033	1.51	8,908
0.70	5,046,325	1.32	6,653	2,348,903	1.26	2,951	5,163,561	1.63	8,419

Table 38: Previously reported resources (2005)

cut-off	Measured			Indicated			Inferred		
kg/t	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8	tonnes	kg/t	t U_3O_8
0.10	39,457,372	0.44	17,419	30,930,569	0.34	10,663	97,201,522	0.35	34,146
0.20	27,427,118	0.57	15,701	18,453,039	0.48	8,835	55,628,450	0.51	28,118
0.25	22,718,853	0.64	14,634	14,456,305	0.55	7,936	43,397,492	0.58	25,360
0.30	18,880,824	0.72	13,574	11,520,570	0.62	7,124	34,700,152	0.66	22,961
0.35	15,803,502	0.80	12,568	9,233,807	0.69	6,377	27,835,964	0.74	20,718
0.40	13,338,122	0.87	11,640	7,504,693	0.76	5,725	22,718,844	0.83	18,791
0.45	11,389,882	0.95	10,804	6,181,120	0.83	5,160	18,779,095	0.91	17,105
0.50	9,810,200	1.02	10,049	5,143,487	0.91	4,662	15,765,281	0.99	15,660
0.55	8,497,910	1.10	9,352	4,337,562	0.98	4,234	13,460,104	1.07	14,438
0.60	7,378,487	1.18	8,700	3,688,109	1.05	3,858	11,643,691	1.15	13,382
0.65	6,444,162	1.26	8,110	3,151,823	1.12	3,517	10,117,076	1.23	12,411
0.70	5,704,680	1.33	7,607	2,745,240	1.18	3,241	8,833,413	1.31	11,542

Table 39: Current resources (2006)

Comparison between *Table 38* and *Table 39* shows that there has been a significant increase in resource tonnes in all categories and at all cut off grades. As the methodologies and parameters employed in both resource estimations were substantially the same, the increase in Measured and Indicated can be attributed to infill drilling of the previously Inferred resources and the increase in Inferred resources is as a result of extensional drilling particularly in Details 2, 6 and 7.

19 Conclusions

Sufficient quality control data are available to indicate that XRF assays of both Acclaim's and Paladin's RC drill samples are both accurate and precise. Grades derived from down-hole radiometric logging compare closely to XRF assays, with a tendency for radiometric logs to return slightly higher U_3O_8 grades in high-grade mineralisation. This may relate to the way disequilibrium corrections have been applied to the radiometric data. The generally good agreement between U_3O_8 grades derived by the two methods indicates that they are compatible for the purposes of resource estimation.

The reliability of grades derived from Gencor's percussion drilling can be assessed by comparing them to the Acclaim data, to bulk samples from test shafts and to diamond core samples. U_3O_8 grades in percussion drill samples from above the water table compare closely to those from test shafts. Comparisons to nearest neighbour radiometric U_3O_8 grades in Acclaim RC drill holes indicates a possible bias to about 10 per cent higher grades in the percussion drill samples at grades above about 800ppm but the comparison is not definitive. Comparisons to grades from Gencor diamond core holes indicate that the core samples possibly under-represent the true grade of mineralisation, probably due to loss of matrix material during drilling. The most recent drilling by Paladin has served to reduce the reliance on the historical drilling by Gencor, it is proposed that this process of replacing the historical drilling is continued.

Variograms of U_3O_8 grades indicate that the continuity of grades is relatively poor over even quite short distances, not unlike that observed in some gold deposits. This is backed up by comparisons of nearest neighbour samples in drill holes and test shafts. However the overall continuity of mineralisation, the geological continuity, is quite strong in plan-view. Variograms based on areas of close-spaced sampling in Detail 1 have been used to guide modelling of the short-scale continuity of U_3O_8 grades in other areas.

Resources have been estimated using Multiple Indicator Kriging with block support correction. Estimates assume that grade control sampling at about 5mE x 5mN x 1mRL will be available prior to mining and a selective mining unit of approximately 5mE x 5mN x 2mRL. It is now probable that mining will take place using grade control information based on a 3.2mx 3.6m grid with radiometric probing of blast holes and that the SMU size will be reduced to 4mx4mx3m. Gencor's trial mining has demonstrated that highly selective mining can be achieved at relatively low production rates. Picking of upper and lower ore contacts will be important in mining and the use of technology such as laser or DGPS excavation control may considerably reduce mining dilution.

The resource definition drilling undertaken in 2005 and 2006 is considered to have been very successful in infilling and therefore increasing the resource confidence within existing areas of the deposit, as evidenced by the increase in quantity in both Measured and Indicated resource categories. The drilling program has also extended the area of known mineralisation, particularly in the area of Detail 7 and to the northern edge of Detail 2, as evidenced by the overall increase in Inferred resources particularly in these areas.

20 Recommendations

Paladin has developed the Langer Heinrich Uranium Orebody into an open cut mining operation treating approximately 1.5Mt of ore per annum to produce approximately 2.6Mlbs of uranium oxide for the export market following successful completion of the Bankable Feasibility Study.

The South African Engineers, GRD Minproc tendered successfully to carry out the BFS. Paladin undertook the overall management of the Project and responsibility for the completion of the BFS rested with this engineering group who liaised with Paladin and other consultants prior to its completion in December 2006. A conventional alkaline leach circuit followed by Resin-In-Pulp (RIP) extraction is now in place for the treatment of the ore.

Paladin now proposes to undertake additional drilling programs to further define the Mineral Resources within the tenement. These programs will be designed to progressively infill areas within the existing resource which are currently classified as Inferred with the intention of elevating the resource categorisation, it is also anticipated that some extensional drilling of the resource will be undertaken particularly in the area of Detail 7 and 6.

It is suggested that the budget for this drilling be:-

Period: January 2007 to June 2008,
Currency: Namibian dollars (N$6 = US$1)

Expenditure Classification	Total N$
Labour and Drilling	3,075,000
Consultants and Contractors	150,000
Materials and Utilities	82,500
Transportation and Communication	287,500
Outside Services	2,520,000
Insurance	111,000
Accommodation, Meals and Expenses	225,000
Other expenses	300,600
Total	6,751,600

Table 40: Budget for Resource definition drilling

In addition to the above, ongoing mining and processing operations will take place at the site.

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CERTIFICATE OF QUALIFICATION

I, Neil Schofield, MAIG and MAusIMM, do hereby certify that:

1. I am a Director of:

   Hellman & Schofield Pty Ltd
   Suite 6, 3 Trelawney St,
   EASTWOOD NSW 2119
   AUSTRALIA

2. I graduated with a BSc(Hons) degree in geology from University of Queensland in 1971. In addition I have obtained an MS in Applied Earth Sciences from Stanford University, California in 1988.

3. I am a Member of the Australian Institute of Geoscientists and the Australian Institute of Mining and Metallurgy

4. I have worked as a geologist for a total of 34 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of part of the technical report titled "Langer Heinrich, Namibia Resource Estimation" (the "Technical Report") and dated 8th January 2007 relating to the Property. I have not visited the property.

7. I have had an involvement in the Property since November 2006. The nature of this involvement includes a re-estimation of the Uranium resources for Details 3, 4, 5 and 6.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical report misleading.

9. I am independent of the issuer in accordance with section 1.4 of NI43-101.

10. I have read NI43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated 8 January, 2007

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Signature of Qualified Person

Neil A. Schofield

____

Name of Qualified Person

CERTIFICATE OF QUALIFICATION

I, David Princep, MAusIMM, B.Sc., hereby certify the following:

1. I am Principal Geologist with

   Paladin Resources Ltd.
   Grand Central,
   First Floor, 26 Railway Rd.
   SUBIACO WA 6904
   AUSTRALIA

2. I graduated with a BSc. degree in geology from University of Liverpool in 1976.

3. I am a Member of the Australasian Institute of Mining and Metallurgy.

4. I have eighteen years experience as a geologist in mining and evaluation of mineral properties within Australia and overseas.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of part of the technical report titled "Langer Heinrich, Namibia Resource Estimation" (the "Technical Report") and dated 8th January 2007 relating to the Property. I have visited the property on a number of occasions, the last being in December 2006.

7. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is that I prepared the technical report regarding the property effective date June 7th, 2005 whilst an employee of Hellman & Schofield. I was responsible for the re-estimation of the Uranium resources for Details 1, 2, and 7 in November 2005 also whilst an employee of Hellman & Schofield. I am responsible for the compilation of this Technical Report.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical report misleading.

9. I am not required to be independent of the issuer in accordance with section 1.4 of NI43-101.

10. I have read NI43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated 8 January, 2007

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Signature of Qualified Person

David J Princep

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Name of Qualified Person