Balausa Vanadium Project

Competent Person's Report

Prepared For:

Ferro Alloy Resources Limited

COMPETENT PERSONS:

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Document Approval

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IMPORTANT NOTICE

This report was prepared for Ferro Alloy Resources Limited by GBM Minerals Engineering Consultants Limited (GBM) and Geo Mineral Resources Limited (GMR) in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the “JORC Code”), 2012 Edition.

This CPR is compliant with the ESMA update of the CESR recommendations – The consistent implementation of the Commission Regulation (EC) No 809/2004 implementing the Prospectus Directive.

The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in GBM and GMR’s services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report.

This report is intended for use by Ferro Alloy Resources Limited for a prospectus which needs to be approved by the United Kingdom Listing Authority (“UKLA”) in connection with a listing on the London Stock Exchange.

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NOMENCLATURE

The following abbreviations have been used in this document.

Abbreviation	Description
AMV	Ammonium Metavanadate
Balausa	The Balausa Vanadium Project that is the subject of this report
BaV_2O_4	Phengite
DCF	Discounted Cash Flow
DM	Datamine^{TM}
ESIA	Environment and Social Impact Assessment
FAR	Ferro Alloy Resources Limited
FPO	Financial Services and Markets Act 2000 (Financial Promotion) Order 2005
GBM	GBM Minerals Engineering Consultants Limited
GKZ	Kazakh State Reserves Committee
GMR	Geo Mineral Resources Limited
H_2O_2	Hydrogen peroxide
IPD	Inverse Power Distance
IRR	Internal Rate of Return
JORC	Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves
Mtpa	Million tonnes per annum
MVA	Meta-Vanadate Crystal Slurry
Na_2CO_3	Sodium Carbonate
NPV	Net Present Value
NSL	Noise Sensitive Location
OB1	Ore Body 1
OB2	Ore Body 2
OB3	Ore Body 3
OB4	Ore Body 4
OB5	Ore Body 5
OK	Ordinary kriging
OVOS	Kazakh equivalent to ESIA
SEE	State Ecological Expertise
SER	State Environmental Review
TFB	TOO Firma Balausa
TOC	Total Organic Carbon
V_2O_5	Vanadium Oxide
V_2S_4	Patronite
VNIIKHT	An approved design institute which carries out metallurgical testwork.

SECTION 1 SUMMARY

1.1 PROJECT BACKGROUND

Ferro Alloy Resources Limited (“FAR”) commissioned GBM Minerals Engineering Consultants Limited (GBM) to prepare a Competent Person’s Report on the Balausa vanadium operations and development projects in the Shieli district, Kzylorda oblast, Kazakhstan. Geo Mineral Resources Limited (GMR) was engaged to produce a mineral resource report, according to acceptable international standards.

FAR’s principal operating subsidiary, TOO Firma Balausa (“TFB”) carries on the current processing operation and holds the development and mining rights to the Balasausqandiq vanadium deposit via a subsoil-use contract. In this report the terms “Company” or “Group” refer to FAR or TFB or both as the context requires.

The licenced area forms a large part of the Balasausqandiq vanadium deposit. This deposit is based on a geological resource which has progressively been delineated by a number of exploration phases since its discovery in 1940 by Soviet era geo-scientists. More recently, FAR have carried out further exploration drilling, trial open-pit mining operations and pilot plant optimisation studies using alternate metallurgical and mineral process treatment technologies.

The Company is currently operating a processing plant based on the former pilot plant, used for testing the proposed process for treating the company’s own ore, suitably adapted to treat purchased concentrates and thereby increased output to a commercial level. Having proven the operating effectiveness, the Company has embarked on plans to increase the maximum potential output to over 1,500 tonnes per annum of vanadium pentoxide by extending the size of the factory building, purchasing suitable additional equipment and improving infrastructure at a total capital cost of some $10.3m.

Up to the end of October 2018 some US$550,000 had already been spent towards this capital programme using funds generated from operations and the production from the treatment of bought-in concentrates had reached a rate of over 12 tonnes per month of vanadium pentoxide in the form of ammonium metavanadate (AMV). In April 2018 a new pre-roaster and the basics of associated leaching and precipitation equipment were installed which enable the company to treat a wider variety of higher grade secondary raw-materials in parallel with the existing production. Commissioning and testing has now been successfully completed and some six tonnes of AMV has been produced from the new raw materials. Further equipment and infrastructure improvements are required to expand production to the targeted level which is expected to be achieved in gradual steps, without any major shutdown, over the period until the end of the first quarter of 2020.

In parallel, FAR intend to develop the Balasausqandiq mine and to build a separate processing plant using a phased approach, with a 1 Mtpa processing facility being constructed initially (“Phase 1”), followed by an expansion to 4 Mtpa (“Phase 2”), to produce a total of some 22,000 tonnes per year of vanadium pentoxide in addition to the production from the currently operating plant.

This Competent Person’s Report provides an assessment of the proposed projects, including details of the current mineral resource, mining engineering, metallurgy, mineral processing, an estimation of capital and operating costs and financial analyses.

Note that within this Competent Person’s Report, the term “ore” is used in the strict JORC definition of the term as a “mineable reserve”. FTB has, until now, operated under the procedures required by the Kazakh State Reserves Committee (GKZ) and where appropriate, information under this system of reserve classification is given for additional information. Except where otherwise stated, all information in this report is given in conformity with the JORC 2012 requirements except as to the sections concerning historical exploration in the Soviet era where the term ‘ore’ is necessarily applied to the reserve classification system employed at the time.

1.2 GEOLOGY

There is an extensive history of geological exploration, especially during the former Soviet-era, since the vanadium was first discovered in 1940. However, as part of this project, research into the expansive complex geological processes that affected the Balasausqandiq deposit, from 1,000 million years ago to 1.5 million years ago, especially in the field of geo-tectonics, has provided a sound basis for supporting the assessment and modelling of this deposit and has allowed a much higher confidence level in the results. This deposit has geological characteristics in common with vanadium deposits in South China, when in primordial times, the Karatau mountains were juxtaposed within the same supercontinent.

The stratiform vanadium layer is associated with five very large orebodies and their surface expression can be traced for about 40 km. These orebodies are mostly confined to deep synclinal folds, where the primary carbonaceous vanadium rocks at depth are protected from weathering and oxidation processes. From historical data and from FAR’s drilling results, the global grades within these orebodies are relatively similar, and this uniformity is testament to the broad stable conditions during mineralogical deposition in a marine basin some 510 million years ago (mid-Cambrian).

1.3 MINERAL RESOURCES AND RESERVES

Potentially, the primary resource is huge, as expressed by the surface continuity of the vanadium mineralisation along strike. The reflection at depth of such observable surface mineralisation has been confirmed by FAR’s drilling of Ore Body 1 (OB1) and also confirmed from the more limited drilling of Ore Body 2 (OB2) and Ore Body 3 (OB3). Currently, based on the OB1 JORC resource, plus JORC-based Exploration Targets for OB2 to OB5, a total vanadium JORC resource of over 100 million tonnes is considered to be a rational prediction**.**

Table 1-1 provides details of the OB1 JORC (2012) resource, for both vanadium and by-products, while the Exploration Targets are summarised in Table 1-2 and Table 1-3.

Table 1-1: Schedule of Mineral Resources (JORC 2012)

JORC Vanadium Resource OB1				By-Products OB1 (primary ore only)						
JORC Class	V_2O_5 % Cut-off	V_2O_5 % Mean	Tonnes [m]	JORC indicated		JORC inferred		Total C% Mean	JORC inferred MoO_3 % mean	JORC inferred U_3O_8 % mean
				C% Mean	Tonnes [m]	C% Mean	Tonnes [m]			
Indicated	0.0	0.67	21.43	14.08	10.68	13.09	10.75	13.59	0.0300	0.0090
Inferred	0.0	0.67	1.56			13.43	1.56	13.43	0.0297	0.0085
Combined	0.0	0.67	22.99					13.58	0.0300	0.0090
Oxide cap inferred	0.0	0.89	1.33							
Total	**0.0**	**0.68**	**24.32**							

Table 1-2: JORC Based Exploration Target (JORC 2012 Guidelines)

Orebodies 2 to 5	Strike Length (km)	Tonnes [m]		V_2O_5 Grade Range [%]	
		From	To	From	To
Primary Zone	20.9	73	98	0.65	0.71
Oxide Zone		4.25	5.75	0.85	0.98
Combined		77.3	103.8	0.66	0.72

Table 1-3: JORC Based Exploration Target (JORC 2012 Guidelines) - By-Products applied to Ore Bodies 2 to 5 (Primary Zone Only)

Target	Global Grades based on OB1	Grade Range ± 5 %	
		From	To
Carbon	13.58 %	12.9	14.26
MoO_3	0.030 %	0.029	0.032
U_3O_8	0.009 %	0.009	0.009
REM	335 ppm	318	352
Total Tonnes (millions)		73	98

Table 1-4: JORC Based Mineral Reserves (JORC 2012 Guidelines) - Ore Body 1 only

Category	Reserve Tonnes (000)	Mean grade V_20_5 [%]
Probable	22,938	0.59

A GKZ reserve of 70 M tonnes was confirmed in 2014 as shown in Table 1-. This GKZ reserve is used as the basis for the Kazakhstan regulatory approval and control processes:

Table 1-5: 2014 GKZ Reserve Summary

Category	Reserve Tonnes (000)	Mean grade V_20_5 [%]
B	832	1.00
C1	15,649	0.75
C2	54,366	0.74
B+C1+C2	70,847	

Table 1-6: Metal equivalents in V2O5 terms (OB1)

Product	Estimated resource grade in situ	V2O5 equivalent
V2O5	0.67%	0.670%
Carbon and organics (in silica)	14.08%	0.187%
Uranium oxide concentrate	0.009%	0.022%
Molybdenum oxide concentrate	0.03%	0.029%
Total		0.908%

This table of metal equivalents is based on the indicated and inferred resource of OB1.

The ore also contains 335 ppm of REM but no value has yet been ascribed to it.

A further by-product, potassium alum, has not been included above as it is derived partly from ore and partly from processing. If it were included, it would add an additional 0.071% on a vanadium pentoxide equivalent basis.

1.4 METALLURGY AND MINERAL PROCESSING

The metallurgical pilot plant installed at the project site shows that the vanadium mineralisation is amenable to conventional comminution and autoclave processing techniques to produce a suite of saleable products. The pilot plant testwork shows a mineral recovery of over 90 %. The mineral process plant will use conventional industrial chemistry processes, standard equipment and machinery.

Run of mine ore will be crushed, milled and classified before thickening. The thickened material is decarbonized in an acid solution and re-thickened and filtered. The solid material is leached in autoclaves to produce a carbon silica by-product, which is then washed and dewatered.

The solution from the decarbonization stage contains the main product elements. The recovery of vanadium, uranium, molybdenum and rare earths is undertaken using three separate adsorption circuits. In each stage the target element is preferentially adsorbed onto ion-exchange resin, allowing the solution to flow on to the next stage. The target elements are desorbed from the loaded resins periodically and precipitated out of solution before drying and packaging.

The proposed operational mineral process plant will produce a suite of saleable products, including vanadium, carbon, uranium, molybdenum, rare earth elements (“REE”), potassium alum and others as defined in this report.

Following completion of the test programme, the pilot plant was adapted to treat concentrates and spent catalysts. This operation does not require several sections of the former pilot plant including the crushing and milling, autoclave leaching, and the by-product recovery sections, but the production process is otherwise similar and the output of vanadium is higher.

1.5 MINING OPERATION

Small scale mining is being carried on at a rate of 15,000 tonnes per annum from open pit and the ore is currently stockpiled. Subject to demand, some of the waste from this mining is crushed and sold as gravel for road-building and construction. The future mining operations will be on a much larger scale but will use similar methods, a conventional open pit employing standard equipment for drill, blast, load and haulage of the material to the process plant from the open pit.

The equipment used will be western standard equipment augmented by regional manufacturers where appropriate. Industry standard grade control techniques will be used to ensure the grade of the material for processing is in accordance with the business plan.

1.6 INFRASTRUCTURE

The current operations and project benefit from a significant amount of regional infrastructure, including high voltage electrical lines nearby, well-made access roads, local telecommunications and a regional railway. There is also site-specific infrastructure developed for the purposes of the initial trial mining and mineral processing operations.

The site has a reliable water supply, labour force accommodation block, engineering workshops, welfare facilities, an office and telecommunication facilities, which are suitable and sufficient for the current operation and typical for the region. All facilities are in accordance with local regulatory requirements.

The proposed operations will require connection to the adjacent High Voltage 110 kV power line, enhanced electrical reticulation, a new accommodation block, new railway siding facilities in Shieli and the existing infrastructure onsite will be augmented and refurbished to a higher operational condition.

1.7 ENVIRONMENTAL

The site is situated in a plain landscape, typically desert type of raised and lowered steppe. The climate in the area is typically sharp continental, with a hot, dry and prolonged summer with temperatures commonly exceeding 40 °C. The winter is relatively short with little snow. Ground frost permeation during the winter is approximately 0.3 to 0.4 m, the coldest month being January, with an average temperature of -6.9°C, but capable of reaching as low as -25°C.

The winds in the area are strong and frequent with a prevailing north, northeast and northwest direction. Annual precipitation in the northern slopes of the Great Karatau Range total approximately 151 mm.

There is no known designated habitat area nearby. The site is in full compliance with the national OVOS scheme for environmental stewardship.

1.8 LOGISTICS

The products from the operation are of high value and relatively low in volume so transport to customers is not difficult or expensive by either truck or rail. The most usual routes will be by truck to the railway at Shieli, 70 km from the plant, where there is a railway station along the main East-West road and rail transit linking Europe and the Russian Baltic, through Kyzylorda, Shymkent and Almaty, into China and on to the East coast, or by truck to the port of Riga from where it can be shipped. There is therefore good access to Europe, Russia and China as well as the local region.

Sales have been made in the past to Russia and China, and more recently to a UK customer for onward shipment to Taiwan. In future, when the 1 Mtpa mine output is achieved, the majority of output will be sold in the form of ferro-vanadium to steel producers or in the form of vanadium electrolyte for use in vanadium flow batteries.

1.9 FINANCIAL ANALYSES

GBM has audited the FAR cash flow model. Using a 10 % discount rate over the first 23 years of operations and the following prices for vanadium pentoxide:

Remainder of 2018	$24.00/lb
2019	$13.00/lb
2020	$10.00/lb
2021 onwards	$7.50/lb

The combined businesses have an NPV (post tax) of USD 2.0 billion. The IRR of the combined businesses (post tax) is 96 %.

The current price of vanadium pentoxide (29 October 2018) is over $26/lb. As a sensitivity, therefore, GBM has considered the case where the price of vanadium pentoxide for 2021 onwards is assumed to be $11/lb, which gives a combined NPV of $3.0 billion.

The main aspects of the cash flow models in US dollars are detailed in Table 1-5.

Table 1-5: Main Aspects of Cash Flow Models

Item	Value
Combined projects:	

Item	Value
Base case post tax asset IRR	96%
Base case post tax NPV (10 %)	US$ 2,048 million
• NPV (10 %) / IRR (processing expansion only)	US$73 million / 242 %
• NPV (10 %) / IRR (phases 1 and 2 mining and processing)	US$1,978 million / 89 %
Expansion of current processing operation	
Capital costs including working capital and contingency (from November 2018)	US$9.7m
Annual output (V2O5 basis, excluding by-products)	1,500 tonnes
Annual revenue (from 2021)	$23.6 million
Annual costs (from 2021)	$11.3 million
Annual net operating cash flow after tax (2021 to 2026)	$9.7 million
Annual net operating cash flow after tax (after expiry of tax incentive agreement)	$7.8 million
Phase 1 – 1 Mtpa mining and processing	
Capital costs including working capital and contingency	$100m
Ore treated per annum	1,000,000 tonnes
Annual output V2O5 (additional to above)	5,603 tonnes
Annual revenue	US$135 million
Annual costs including royalty	US$32 million
Annual operating cash generation after tax	US$103 million
Phase 2 – additional 3 Mtpa mining and processing	
Capital costs including working capital and contingency	US$225m
Ore treated per annum (total incl. Phase 1)	4,000,000 tonnes
Annual output V2O5 (total incl. Phase 1)	22,414 tonnes
Annual revenue (total incl. Phase 1)	US$541 million
Annual costs including royalty (total incl. Phase 1)	US$110 million
Annual operating cash generation after tax (total incl. Phase 1)	US$430million

The NPV and IRR figures assume that the expansion of current operations will be carried out during the remainder of 2018 until the end of the first quarter of 2020 and that production will increase in steps over that period. Final testing and detailed engineering for Phase 1 will start immediately upon receipt of finance in late 2018, construction will start in 2019 with start-up at the end of 2020. Construction of Phase 2 is scheduled to start in January 2022 with start-up in second half of 2023.

Within this cash flow model it is assumed that cash generated is kept within the company and used to fund the expansions to 1Mtpa and 4Mtpa. Funding for this programme will be substantially from earnings which will depends on both raw material and end product prices so cannot be predicted with any accuracy but is expected to be approximately as follows:

Table 1-6: Summary of Capital Requirements

	Capital costs (US$)		
	Expansion of current processing	**Phase 1 (1Mtpa)**	**Phase 2 (4 Mtpa)**
Initial equity funding 2018	US$5m	US$28m	-
Debt or bond 2019		US$58m	-
Funded from retained earnings	US$5m	US$14m	US$225m
Total capital requirement	**US$10m**	**US$100m**	**US$225m**

SECTION 2 INTRODUCTION AND TERMS OF REFERENCE

2.1 COMPANY BACKGROUND

FAR is currently registered in Guernsey, The Channel Islands. FAR has five subsidiaries, as shown in Figure 2-1.

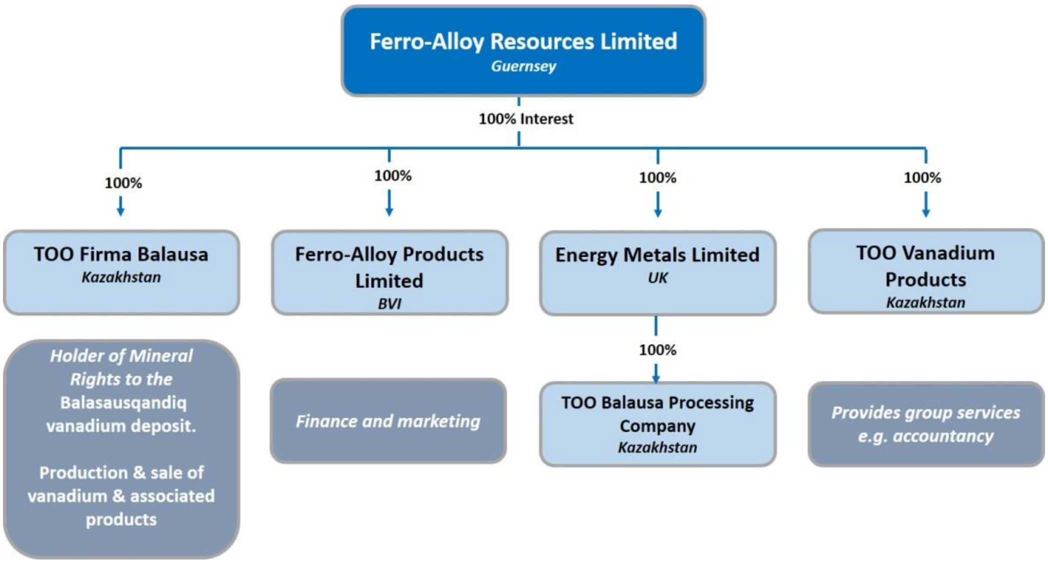


Figure 2-1: FAR Group Structure

The current directors of FAR are Nicholas Bridgen (Chief Executive), Andrey Kuznetsov (Chief Operating Officer and Director of TOO Firma Balausa Company, Almaty), Christopher Thomas (non-Executive Director) and James Turian, (non-executive director).

TFB was formed in 1996 when it was granted an exploration licence for the project. This was converted to an exploitation licence in 1998. The merger with FAR occurred in 2000 and the construction of the pilot plant at Balausa was announced in 2006.

Management of the Balausa vanadium project is supported by the technical staff based at the regional office of TFB in Shieli, approximately 70 km from the deposit.

Currently, TFB employs approximately 141 staff, working two shifts each day, with each shift remaining on site for 15 days followed by 15 off each month throughout the year. The existing management structure of TFB is shown in Figure 2-2 below.

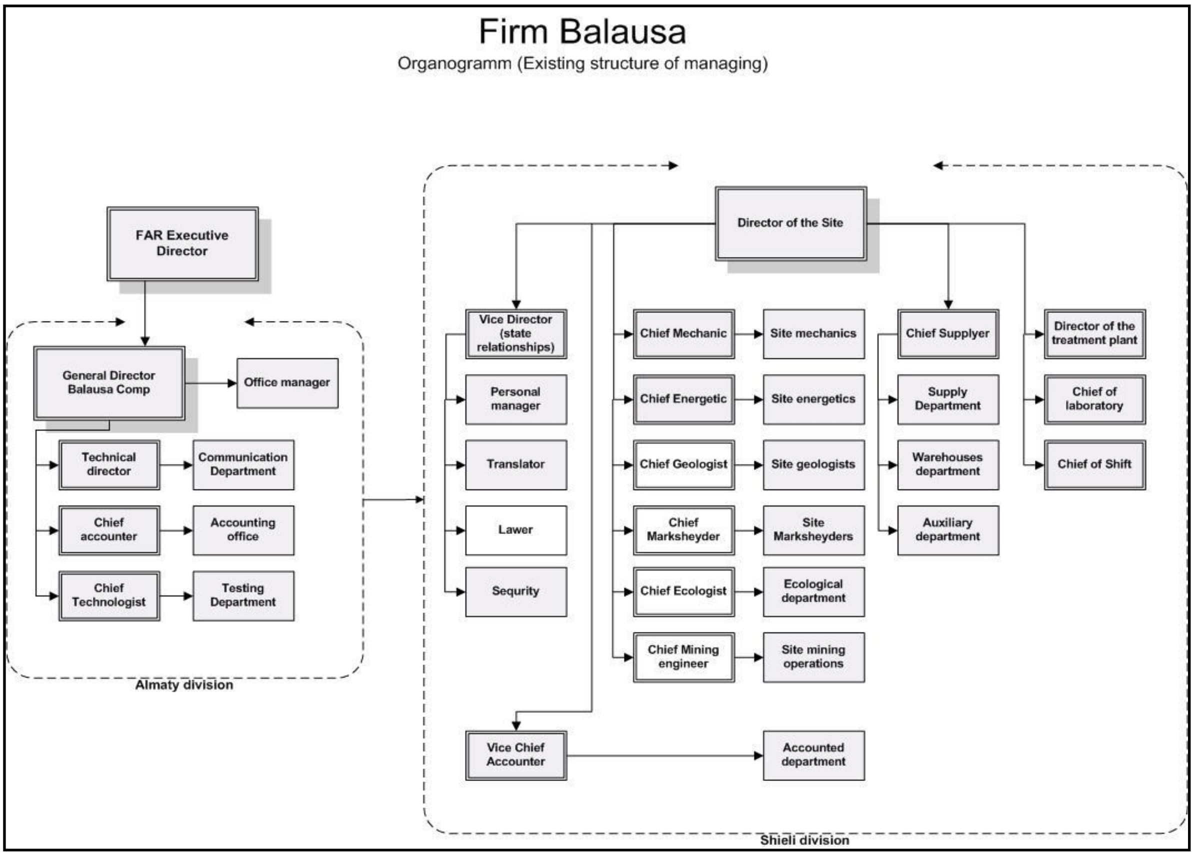


Figure 2-2: Existing Management Structure

2.2 NON-TECHNICAL SUMMARY

The Balasausqandiq deposit is large, with the centre of the estimated range for the resource being around 126 million tonnes, equivalent to over 840,000 tonnes of V_2O_5 and equal to nearly six times the annual world production of 2017. It is likely that the resource estimate will increase after further exploration. If there were no other considerations, the NPV of such a project would normally be optimised by aiming to deplete the resource over 15 – 20 years, indicating an ore treatment rate of around 8 Mtpa, producing around 44,000 tonnes per annum of vanadium pentoxide or equivalent vanadium products.

The Company's first step was to build a 15,000 tonnes per annum pilot plant to test the proposed treatment process for the Company's own mined ore. After completion of the test-programme, this plant was adapted to treat purchased concentrates, thereby increasing its output to a semi-commercial scale. The operation of first the pilot plant and subsequently the adapted plant has enabled the company to refine many of the processes that it will be using in the larger plant, to train the workforce, and to understand the cost structure of the business. These lessons will be invaluable and will substantially de-risk the expansion programme.

Although the deposit is large, there are several reasons why a more gradual development plan is being adopted. Firstly, the world market demand for vanadium (pentoxide equivalent) is currently only around 153,000 tonnes per year, so there would be a concern that if FAR produced at its optimised level the market could be oversupplied in the short term. Secondly, such a large production facility would be technologically risky. It is less risky to start up with a more manageable size and expand later when the experience of construction and operation of a smaller treatment plant can be used to make improvements to the later, larger facility. Thirdly, a smaller amount of money has to be raised to build a smaller plant and the earnings from that plant can be used to pay for, or underpin borrowing for, the next much larger plant, thus considerably reducing shareholder dilution.

The directors of FAR therefore decided on a three-step development plan. The first stage of development, which has already begun, is to extend the existing plant and adapt it to enable it to treat a wide range of much higher grade secondary vanadium-containing raw materials including catalysts from the demetallisation of oil and boiler ashes, enabling output to be increased to some 1,500 tonnes of vanadium pentoxide (equivalent) per year. This expanded plant will include equipment to convert the primary product, ammonium metavanadate, into vanadium pentoxide in powder form. Provision may also be made, subject to market demand, to make electrolyte for batteries. In parallel with the further expansion of the existing plant, detailed engineering will begin on the first of two phases, Phases 1 & 2, of the Company's own mine and processing plant which will be entirely separate from and operate independently of the expanded existing processing plant.

The throughput from Phase 1 of the Company's own mine will be 1 million tonnes per annum ("Mtpa") of ore, producing 5,600 tonnes of vanadium pentoxide (or equivalent in other forms) per year, which is a level that is much lower than the expected annual increase in the market size and should be easily absorbed by the world market. Immediately after commissioning of Phase 1, approximately two years later, work will begin on Phase 2 which is planned to increase throughput to 4Mtpa, producing a total of over 22,000 tonnes per year.

Assuming an initial capital raise of US$35m (US$33m net of expenses), the phased approach will enable the earnings of the expanded current processing operation to fund the remaining equity requirement of Phase 1 of the Balasausqandiq mine (together with approximately $58 million of debt or a bond issue), and the earnings of the two initial operations, to fund the entire requirement of Phase 2. Equity dilution after the first raise will therefore be minimised or eliminated.

The Phase 1 and 2 developments to mine and treat the Company's own ore will also produce several valuable by-products. These, in order of importance, are a valuable form of carbon – which comprises some 18 % of the tailings (together with the associated silica), potassium alum (which can be further refined into alumina and potassium and ammonium sulphate fertilizers), uranium and molybdenum. In addition, up to 330 tonnes per year of a mixed rare earth concentrate will be produced but owing to current low prices and uncertainty over the market demand for this material, no value is being ascribed to it.

The carbon-silica tailings have a number of uses, principally as a feed in the making of ferro-silicon or as filler in the making of rubber as a substitute for carbon black and silica. Both uses have been tested with successful results. The suitability of the material for making ferro-silicon gives the company the potential to build its own ferro-silicon plant which would avoid having to transport the material and has several other competitive advantages. At this stage, only a conceptual study has been carried out, based on analogous costs of another operation adjusted for local costs. The reason why this project is attractive is that all the main production costs, power, labour, carbon and silica, are extremely low cost. The silica and 50 % of the carbon would come from the briquetted tailings of the vanadium operation and power would be generated from locally procured natural gas. Labour costs in this region of Kazakhstan are also very low. Not only would the input costs be very low, but the tailings from the vanadium operation have already had their metallic impurities leached out of them, meaning that the resulting ferro-silicon would be of high purity and low in aluminium, giving a premium price. Initial indications are that such an operation could produce ferro-silicon at an exceptionally low cost, perhaps even the lowest of any in the world. The conceptual capacity is around 150,000 tonnes per year initially, which would use approximately 450,000 tonnes per year – more than half the output of carbon-silica tailings from the vanadium treatment plant. This project would be capital intensive and requires know-how of a new industry. The directors would therefore like to develop it in conjunction with an established ferro-silicon producer who knows the technology and can assist with the marketing. This ferro-silicon project is separate from the vanadium project and is not included in the financial analysis in this Competent Person’s Report or any cash flows mentioned.

There are several reasons for anticipating large annual increases in the world demand for vanadium but two are worth noting. Firstly, there is expected to be an increase in the number of countries requiring the use of vanadium-bearing steel for construction purposes – China started to implement it several years ago but further changes to regulations starting in November 2018 are expected to increase demand significantly. India and the CIS countries are expected to follow soon. Secondly, vanadium is already being used for batteries for electrical storage and this use is expected to grow markedly over the coming years. For these reasons, the directors consider that if demand for vanadium increases as some forecasters foresee, particularly if demand for vanadium for flow batteries increases, there is a strong likelihood that a further expansion after Phase 2 of up to around 8Mtpa may be considered.

It is anticipated that planning for the Phase 2 expansion of the Balasausqandiq mine will start as soon as Phase 1 is in operation. Allowing two years for design and construction (with construction to begin after six months of detail engineering), and making allowance for other potential delays, it is expected to come on stream around in the second half of 2023.

GBM concurs with this phased approach and agrees that a phased approach to project development can reduce project risks.

Table 2-1 shows the anticipated timing of the proposed expansions on the assumption that finance is available in a timely fashion.

Table 2-1: Approximate Expansion Timings

Development	Timing
Current processing operations	
Around 144 tpa of vanadium pentoxide equivalent	Already operating
Expansion to 1,500 tpa vanadium pentoxide equivalent	Increasing over the remainder of 2018 until the end of the first quarter of 2020
Phase1 – (1Mtpa)	
Detailed engineering and other preparatory work	Remainder of 2018 and early 2019
Construction	2019 – 2020
Commissioning starts	Late 2020
Phase 2 – (4Mtpa)	
Detailed engineering and other preparatory work	2021
Construction	2022 – 2023
Commissioning	Second half of 2023

2.3 COMPETENT PERSONS

The Competent Person for this report is Tim Daffern B Eng (Mining), MBA, FIMMM, FAusIMM, MCIM, MSME. Tim Daffern is responsible for the entire report. The GBM Competent Person is Tim Daffern and the GMR Competent Person is Roger Rhodes BSc, MSc, MIMMM. Tim Daffern authored Sections 1, 2, 3, 5, 6, 7, 8, 9 and 10. Roger Rhodes authored Section 4. Both contributing Competent Persons have sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity they are undertaking to qualify as Competent Persons as defined in the ”AiM Note for Mining and Oil and Gas Companies (June 2009)”.

Mr Daffern is a mining engineer of approximately 30 years international experience. This experience spans five continents in mining engineering, mine management, as a Director of numerous public and private mining companies and a corporate level technical consultant. Mr Daffern has co-authored over 65 CPR and NI 43 101 documents for IPO, M&A and Due diligence purposes.

Mr Rhodes is an economic geologist in earth resource management, with over 45+ years of mining industry experience world-wide. Since 1988, he has been an independent consultant, and has worked on numerous large-scale projects, for major mining companies, private investors and government agencies, where his expertise in resource modelling and grade estimations have been used for bankable feasibility studies and JORC-based resource evaluations. He also has extensive experience, in auditing and due diligence of mineral resources, QA/QC assessment of sampling procedures and laboratory sample preparation and analyses. Also, he has notable experience in the field of geophysics and geochemistry for mineral exploration. He has written many Competent Person reports for clients and, also, in conjunction with international consultancies.

The Competent Person Tim Daffern confirms he has taken all reasonable care to ensure that the information contained in this Competent Persons' Report is, to the best of their knowledge, in accordance with the facts and contains no omission likely to affect its import.

This report was prepared for Ferro Alloy Resources Limited by GBM Minerals Engineering Consultants Limited (GBM) and Geo Mineral Resources Limited (GMR) in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the "JORC Code"), 2012 Edition.

This CPR is compliant with the ESMA update of the CESR recommendations – The consistent implementation of the Commission Regulation (EC) No 809/2004 implementing the Prospectus Directive.

The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in GBM and GMR's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report.

This report is intended for use by Ferro Alloy Resources Limited for a prospectus which needs to be approved by the United Kingdom Listing Authority ("UKLA") in connection with a listing on the London Stock Exchange.

The following site visits have been carried out by Roger Rhodes to inspect the project site and verify its characteristics:

- 8 to 11 April 2007: Detailed review of operations with technical staff, including field visits with extensive geological examination of orebody related surface exposures. Collected numerous computerised historical documents and maps, plus translated technical reports. Captured and documented GPS locations of the plant and o/p operations.
- 4 to 7 June 2009: Reviewed open pit field operations, especially grade control channel sampling and results of analyses in relation to the exposed vanadium layers and geological sample maps. Visited numerous historical sample locations from the 1940s. Established analytical requirements for the FAR exploration programme.
- 8 to 11 January 2010: Reviewed progress with technical staff, about diamond drilling, logging, mapping and sampling procedures. Site visits included the new sample preparation laboratory.

- 14 to 20 August 2010: Numerous field visits to review the progress of the FAR exploration diamond drilling programme resulting in upgrading a number of procedures to facilitate quality control. Established a requirement for local co-ordinate system to be very accurately correlated with the historical grid and the UMT (WGS84) system.
- 8 to 11 December 2010: Continued to monitor the exploration programme and to check that the core sampling and sample preparation procedures are to acceptable international standards.

2.3.1 FINANCIAL INTEREST DISCLAIMER

Neither GBM, GMR, nor any of their consultants employed in the preparation of this report, have any beneficial interest in the assets of FAR.

GBM and GMR have been paid fees and will continue to be paid fees for this work in accordance with normal professional consulting practices.

2.3.2 RELIANCE ON OTHERS

The Competent Persons have relied on expert opinions and information provided by FAR pertaining to environmental considerations, taxation matters and legal matters including mineral tenure, surface rights and material contracts.

In addition, the Competent Persons have relied on property ownership data provided by FAR. This information is believed to be essentially complete and correct to the best of the Competent Persons' knowledge and no information has been intentionally withheld that would affect the conclusions made herein. The Competent Persons have not researched the property title or mineral rights for the project and express no legal opinion as to the ownership status of the property.

The Competent Persons have relied on information pertaining to product markets and prices provided by FAR. The Competent Persons have reviewed the information provided by FAR and believes this information to be correct and adequate for use in this report.

For the purposes of Section 9 (Environment) of this report the Competent Persons have relied on information provided by FAR. The Competent Persons have reviewed the information provided by FAR and believe this information to be correct and adequate for use in this report.

For the purposes of Section 10 (Financial Analysis) of this report the Competent Persons have relied on information provided by FAR pertaining to taxation. The Competent Persons have reviewed the taxation information provided and believes it to be correct and adequate for use in this report.

SECTION 3 PROJECT DETAILS

3.1 LOCATION

The Balasausqandiq deposit is located in Kazakhstan on the north-eastern slope of a ridge forming the north-western part of the Great Karatau Range. The current processing operation and associated facilities are located on a gently inclined alluvial plain to the south of the ridge.

Administratively the site is situated in the Schieliiskiy Region of the Kyzylorda Oblast of southern Kazakhstan, as shown in Figure 3-1.

Figure 3-1: Location of the Balasausqandiq Deposit in Southern Kazakhstan

The nearest centre of population is the Aksumbe village situated approximately 18 km from the proposed mine (23 km from the current operations) and within the neighbouring Oblast (see Figure 3-2).

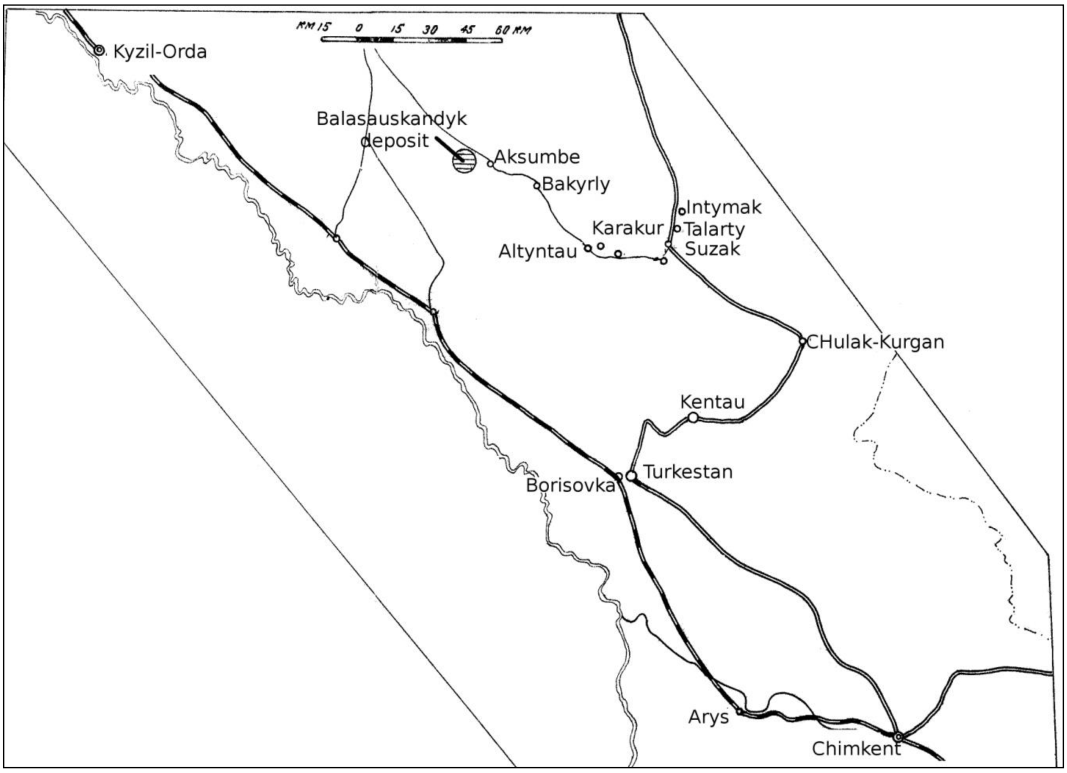


Figure 3-2: **Location of Balasausqandiq in the Schieliiskiy Region of the Kyzylorda Oblast**

3.2 ACCESS

Access to the site is via an 800-metre graded and gravelled track, from a larger surfaced road maintained by KazAtomProm to Shieli (refer to Figure 3-3) approximately 70 km from the project site. Shieli is approximately 120 km from Kyzylorda along the main Kyzylorda-Shymkent highway. Total driving time from Shieli to the deposit is around one hour, depending on weather conditions. Shieli is the nearest railway station to the site. The nearest domestic airport is located in Kyzylorda, while the main international airports are in Almaty and Astana, approximately 850 km and 1,000 km respectively from Shieli.

Figure 3-3: Access Road from Shieli

3.3 TOPOGRAPHY AND CLIMATE

The Karatau ridge extends in the north-western direction. Its general structure is apparently asymmetric with a steeper north-eastern and flatter south-western slope. Figure 3-4 shows the typical topography and scenery adjacent to the mine site.

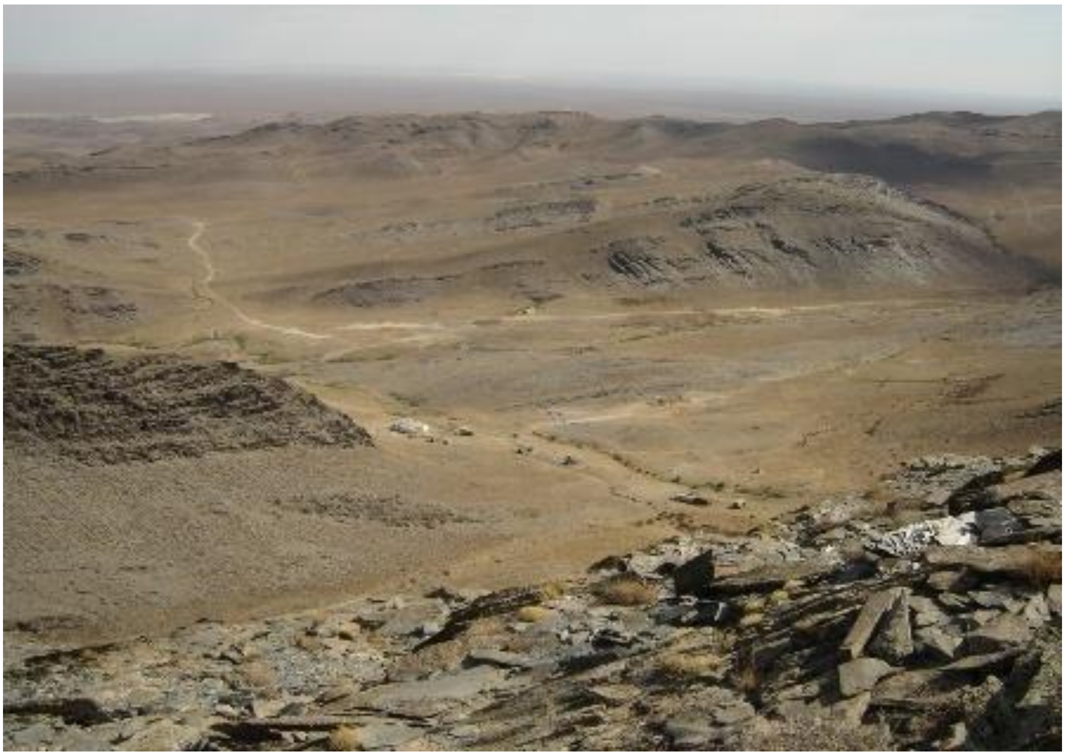


Figure 3-4: Typical Topography at the Current Operation

The Karatau mountain range, situated north of the central course of the 2,200 km Syr Darya river, is a tectonic uplift to 1,500 m that occurred 1.5-1 million years ago. The deposit, including FAR's open pit operations, is located within a ridge of hills (Karatau mountains). In this part of the mountain range, elevations vary from 400 m to 700 m and are composed of a series of SE-NW trending ridges which are interspersed by cross-cutting valleys with dominant drainage patterns to a low lying basin in the NE (Chu Sarysu Basin).

The climate in the area is typically sharp continental, with a hot, dry and prolonged summer with temperatures commonly exceeding 40 °C. The average summer temperature is approximately 36 °C. Daily temperature fluctuations of 16 – 18°C are common.

The winter is relatively short with little snow. Snow cover can be observed from late November until late March. Ground frost permeation during the winter is approximately 0.3 to 0.4 m, the coldest month being January, with an average temperature of -6.9°C, but capable of reaching as low as -25°C.

The site is accessible year round, although it is reported that on certain days when the prevailing wind blow is high, there is snow on the ground and the temperature is at the minimum, access is limited. Such days are rare, at less than 1 in 100 likelihood.

The winds in the area are strong and frequent with a prevailing north, northeast and northwest direction. According to the data provided by the Shieli weather station, the total number of days per year with no discernible wind amount to only 14.3 %. The average wind speed is 3.2 m per second, but can reach 20 – 25 m per second.

Annual precipitation in the northern slopes of the Great Karatau Range totals approximately 151 mm, according to the data provided by the Suzak weather station. The annual precipitation in the southern area of the range is around 417 mm a year, according to the Shieli weather station. Actual precipitation or evaporation rates for the site are not known.

3.4 INFRASTRUCTURE

The current mine has infrastructure sufficient for operating the existing processing plant, which comprises small-scale processing, including temporary sedimentation and water storage facilities, and the crushing of mine waste to a marketable aggregate product. Output from current small-scale mining activities is stockpiled for future mineral processing. The current infrastructure includes the administrative office, assay laboratory, security building and nearby mine camp accommodation facilities.

The infrastructure required for the proposed expansion is described in Section 8.

3.5 MINERAL RIGHTS AND PERMITTING

Mining operations are permitted under the provisions of a ‘Subsoil Usage’ Licence (MG No. 1278 D), which encompasses an allocated area of 1.176 km^2. This licence was granted by the Government of the Republic of Kazakhstan in 1998 and allowed for a mining extraction rate of 500,000tpa. It was initially valid for 25 years but changes in law followed by appropriate permitting obtained by TFB have had the effect of extending this to at least 2043 and it can be further extended when required. A new Subsoil Code came into effect in July 2018 and under the terms of this code an application for further changes has been made with a revised schedule of extraction of 1Mtpa until 2043. This has been approved by the Expert Commission set up by the Ministry of Investment & Development and a Protocol of the Working Group of the same ministry has made recommendations and comments, all of which the Company has complied with. The Company is now awaiting the final Protocol of the Working Group whereupon the Ministry of Investment & Development will sign the appropriate addendum to the subsoil use agreement,

The site boundary of the concession area where extraction of the vanadium ores will be allowed covers an area of about 54 km^2.

The coordinates of the site boundary are detailed in Table 3-1, and are depicted in yellow on the Google Earth Map in Figure 3-5. It covers most of the known vanadium mineralisation in the immediate area, but from historical mapping and sampling, the OB5 vanadium mineralisation continues beyond the boundary area to the north-west, along the geological strike. However, it is understood that for any future mining of this orebody, an extension of this exploitation licence is allowable using the well-known principle of ‘extralateral rights’.

Table 3-1: Site Boundary Coordinates

Point #	Northing	Easting
1	44º31’30”	67º20’10”
2	44º32’40”	67º22’30”
3	44º31’00”	67º26’30”
4	44º29’00”	67º29’00”
5	44º27’20”	67º25’00”

Note: This site boundary area does not include the pilot plant and ancillary infrastructures, but these are covered by individual land title certificates, including the administration offices at Shieli (65 km SW of Balausa) and were issued between 2006 and 2007.

The mining allotment covers approximately 19.2 km^2 and gives the right to mine 500,000tpa according to the approved plan for the period up to 2043 within the coordinates listed in Table 3-2 and as depicted in red on the Google Earth Map in Figure 3-5. The mining rate will be increased to 1 million tpa upon signing of the addendum described above and in 2021 FAR plans to make an application to increase the mining rate further to 4 million tpa.

Table 3-2: New Mining Allotment Coordinates

Point #	Northing	Easting
1	44°32’45”	67°19’30”
2	44°32’5”	67°21’2”
3	44°32’26”	67°21’31”
4	44°32’14”	67°21’49”
5	44°31’54”	67°22’22”
6	44°31’28”	67°23’4”
7	44°31’8”	67°23’33”
8	44°30’46”	67°24’7”
9	44°30’24”	67°24’35”
10	44°29’48”	67°25’18”
11	44°29’30”	67°25’36”
12	44°29’2”	67°24’28”
13	44°26’40”	67°25’38”
14	44°26’32”	67°25’14”
15	44°29’26”	67°23’35”

Point #	Northing	Easting
16	44°30’28”	67°22’18”
17	44°31’7”	67°21’50”
18	44°32’34”	67°19’15”

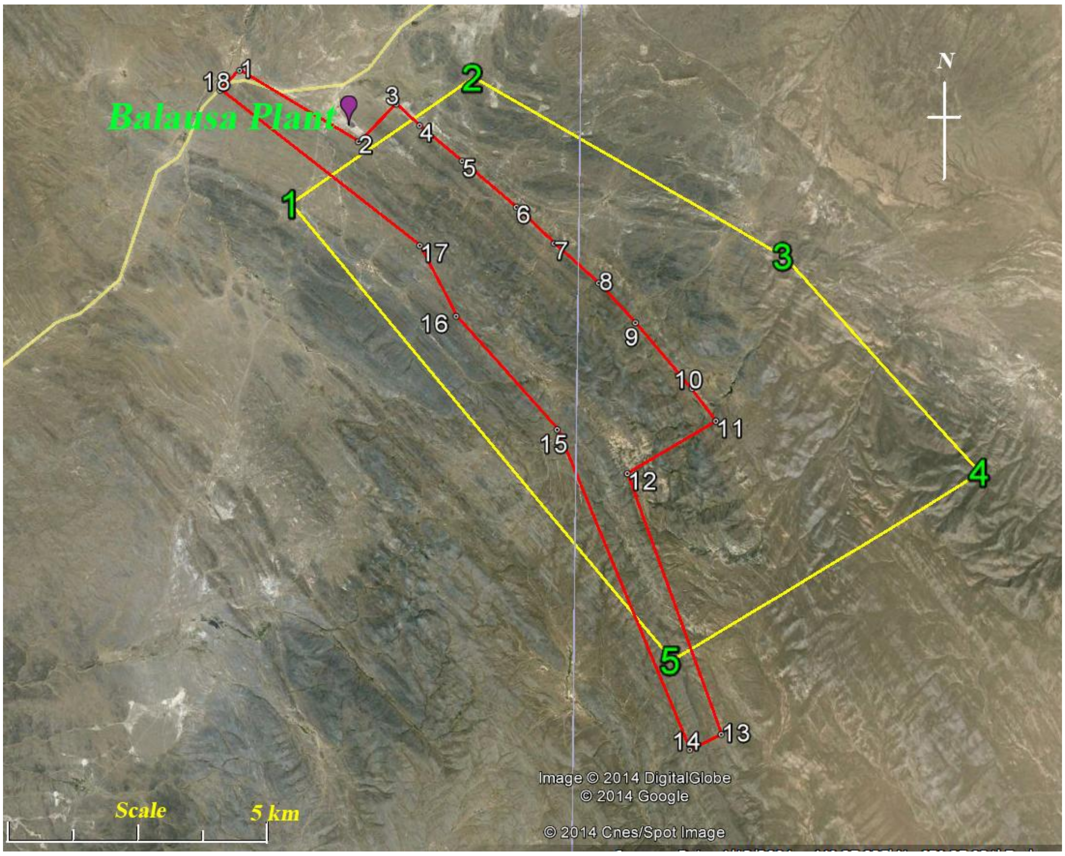


Figure 3-5: Aerial Image of the Site Boundary (yellow) and Mining Allotment

SECTION 4 MINERAL RESOURCE ESTIMATION

4.1 INTRODUCTION

GMR was commissioned by FAR in 2007 to provide on-going technical expertise in mineral resource evaluation, including geological and exploration support, to enable proper assessment of the Balasausqandiq deposit according to acceptable international standards. This diligent approach has consequently allowed the reporting of resources in accordance with the internationally recognised JORC (2012) Code guidelines. In addition, FAR is obliged to report reserves under the officially approved GKZ (Kazakhstan's State Commission on Mineral Reserves).

4.1.1 JORC MINERAL RESOURCES

FAR's drillhole exploration programme, 2010 to 2011, of the primary mineralization at depth was designed to allow JORC "Indicated" resources within ore bodies OB1 and OB2, with an additional option to evaluate OB3 and OB4 for an "Inferred" JORC resource category. However, only sufficient drilling was completed by FAR for orebody OB1 to be classified a JORC-(2012) based resource. Thus this JORC-defined resource estimation is confined only to OB1, where the total resource is smaller than the official GKZ reserves which include variably defined ore portions from ore bodies OB1 to OB5. However, under the JORC (2012) Code, it has been possible to apply JORC-based exploration resource target figures for OB2 to OB5, where additional drilling will allow upgrading to "Indicated" and "Inferred" accordingly. Note: metal equivalents were generated for OB1, as based on the contribution from the by-products of C, MoO_3, U_3O_8 within the primary resource. The method of calculation was to take the sales value derived in the financial model from each product and ascribe to it a vanadium pentoxide equivalent grade equal to the proportion that sales revenue bears to the revenue from vanadium pentoxide. All products are derived from the same process so no allowance was made for differential processing costs.

GKZ Mineral Reserves

There is an approximate relationship between JORC (2012) and the GKZ resources, where JORC "Indicated" is approximate to the GKZ "C1" category and JORC "Inferred" (± JORC exploration) is equivalent to the GKZ "C2" category.

Note that the GKZ reserve (Table 4-68) is historically based and confined to a limited depth of mineralisation with no differentiation between the surface oxide zone and the deeper primary ores. This depth limitation is the main reason for the smaller GKZ resource estimate when compared with the combined JORC "Indicated" (OB1) and JORC based exploration targets (OB2 to OB5). It is therefore a reasonable expectation that the size of the GKZ resource will increase when these ore bodies have been explored to full depth. Test-work has indicated that primary and oxide ores are amenable to the same treatment process, so the lack of a distinction on the GKZ basis does not present any difficulty in the Kazakhstan regulatory environment.

4.1.2 RESOURCE ESTIMATION TARGET

The specific target for this JORC (2012) mineral resource estimate has been confined to OB1, which is one of a number of extensive but separate synclinal-type folded vanadium orebody structures. Historically, these target zones were split and classified according to seven exploration blocks in the 1940s. The historical documentation continually refers to these exploration blocks, plus or minus the actual designated number for each orebody:

- Exploration blocks 1 and 2 refer to orebody 1 (OB1)
- Exploration block 3 refers to orebody 2 (OB2)
- Exploration block 4 refers to orebody 3 (OB3)
- Exploration block 5 refers to orebody 4 (OB4)
- Exploration block 6 refers to orebody 5 (OB5)

With reference to exploration block 7, there appears to be uncertainty as to its location, and therefore this mineralised zone could not be assessed by GMR in determining the JORC-(2012) based exploration resource target figures.

4.1.3 SUMMARY TABLES OF JORC-(2012) BASED RESOURCES, EXPLORATION TARGETS AND RESERVES

A summary of the OB1 JORC (2012) mineral resources is provided in Table 4-1, while Table 4-2 and Table 4-3 summarise the JORC-(2012) based exploration target figures for OB2 to OB5 for V_2O_5 and by-products, respectively. Table 4-4 summarises the JORC (2012) reserves.

Table 4-1: Schedule of Mineral Resources, JORC (2012)

JORC Vanadium Resource OB1				By-Products OB1 (primary ore only)						
				JORC indicated		JORC inferred		Total C% Mean	JORC inferred MoO_3 % mean	JORC inferred U_3O_8 % mean
JORC Class	V_2O_5 % Cut-off	V_2O_5 % Mean	Tonnes [m]	C% Mean	Tonnes [m]	C% Mean	Tonnes [m]			
Indicated	0.0	0.67	21.43	14.08	10.68	13.09	10.75	13.59	0.0300	0.0090
Inferred	0.0	0.67	1.56			13.43	1.56	13.43	0.0297	0.0085
Combined	0.0	0.67	22.99					13.58	0.0300	0.0090
Oxide cap inferred	0.0	0.89	1.33							
Total	**0.0**	**0.68**	**24.32**							

Table 4-2: JORC Based Exploration Target (JORC 2012 Guidelines)

Orebodies 2 to 5	Strike Length (km)	Tonnes [m]		V_2O_5 Grade Range [%]	
		From	To	From	To
Primary Zone	20.9	73	98	0.65	0.71
Oxide-Zone		4.25	5.75	0.85	0.98
Combined		77.3	103.8	0.66	0.72

Table 4-3: JORC Based Exploration Target (JORC 2012 Guidelines) - By-Products applied to all Ore Bodies 2 to 5 (Primary Zone Only)

Target	Global Grades based on OB1	Grade Range ± 5 %	
		From	To
Carbon	13.58 %	12.9	14.26
MoO_3	0.030 %	0.029	0.032
U_3O_8	0.009 %	0.009	0.009
REM	335 ppm	318	352
Total Tonnes (millions)		73	98

4.2 EXPLORATION

4.2.1 1940-1947

Since vanadium was first discovered in the north western part of the Karatau ridge in 1940, continual surface prospecting and evaluation activity in the area resulted in a preliminary exploration of the Balasausqandiq deposit in 1942 and detailed systematic exploration of the Balausa region from 1943 to 1947, with a break in 1945.

The 1943 to 1946 exploration episode covered a vast area but specifically focussed at the Balasausqandiq deposit, led by S.G. Ankinovich et al, though it was only confined to the upper horizons of the mineralisation, with sampling of the orebodies restricted to mainly a series of surface trenches (typically spaced from 50 to 100 m along the geological strike, though at 800 m along OB5), with assessment and sampling support from vertical pits, adits and short shafts with underground drives and cross-cuts. Sampling of the mineralisation in underground exposures only reached a maximum depth of 30 m below surface and was generally restricted from 10 m to 20 m. However, two unsuccessful experimental drillholes resulted in low core recoveries of only 10 % to 20 %, and so it was not possible to properly assess the characteristics of the primary mineralisation below the influence of the surface oxidation processes.

A summary of the works conducted by S.G. Ankinovich from 1941 to 1947 are given in Table 4-4.

Table 4-4: Summary of the works conducted by S.G. Ankinovich from 1941 to 1947

Description of Works	Total Volume of Works
Core drilling unsuccessful (only 15 to 20 % core recovery)	158.0 linear metres
Exploration shafts	140.0 linear metres
Adits	212.0 linear metres
Cross-cuts and drives	239.9 linear metres
Shallow pits up to 10 m	334.6 linear metres
Open pit extraction	897.0 m^3

4.2.1.1 SAMPLING OF OXIDE AND PRIMARY ZONES

This early prospecting work on the deposit had exposed and sampled oxide ores in some detail, however at depth, details of grade, thickness of ore zone and amenability to beneficiation of the primary ores, which account for 95 % of all potential resources, remained unknown. At a depth of 30 m below surface, the ore zone had been exposed by only two workings (crosscuts from shafts 2 and 4). The other six shafts and five adits with crosscuts intersected the ore horizon at a depth range of only 15 to 20 m below surface, which clearly lies in the oxide or transition zone only.

Chemical assays performed on the vanadium ore samples revealed minor uranium content with average grade values ranging from 0.006 to 0.015 %. Uranium mineralisation is unevenly distributed, and was shown to be normally associated with the upper part of the oxidation zone within the vanadium-bearing strata. This unit is thought to be enriched in uranium due to its migration from deeper horizons and subsequent adsorption by iron hydroxides.

4.2.2 1949-1969 AKINTOVITCH

Further work was conducted on the deposit from 1949 and included the following:

- E.A. Ankinovich described 148 minerals, including a few previously unknown species.
- The same year a geologic map of Karatau was issued (Scale 1:200,000), summarising many years of geological exploration.
- In 1952-1953 surveys of the vanadium-bearing strata were resumed.
- In 1961 S.G.Ankinovich published a comprehensive geological analysis of north-western Karatau.
- In 1959-1961 surveys for rare-earth metals (REE) mineralisation was conducted on the vanadium-bearing shales of Karatau. The report was issued in 1963.
- In 1961, the Balasausqandiq deposit was surveyed by three reconnaissance lines with a spacing of 2,050 to 2,650 m. The reconnaissance lines exposed the mid part and the flanks of the deposit. Additional surveys were undertaken on the north-western flank of the deposit (in Block 4) every 3,250 to 3,300 m. In total 15 trenches were cleared and deepened for

sampling including the crosscut running from Adit No. 1. A total of 198 samples were collected from the cleared workings, including 13 samples of primary ores, one sample from the siliceous overburden of the primary ores, 16 samples from the top siliceous section of oxidised ores, 13 samples from the siliceous soil of the oxidised ore and 155 samples from the base of oxidised ore.

- Field surveys were conducted during 1960 and 1962 (Scale 1:50,000 and 1:200,000) to update the geological map of the entire north-western part of the Karatau mountain ridge within the limits of the vanadium-bearing basin.
- Surveys were conducted during 1966-1968 by Kazakh Polytechnic institute, under the guidance of S.G. Ankinovich, which explored underlying rocks beneath the vanadium-bearing horizon where high grades of silicon dioxide have been defined. The results of this work were summarised in 1969 by T.M. Alzhanov.

4.2.3 1972-1973 KOMARNITSKI

4.2.3.1 DRILLING OF PRIMARY MINERALISATION

During 1972 and 1973 15 coring drillholes were concluded, totalling 1,744 linear metres, within the primary mineralised zones beneath the oxide cap, together with an additional 22 new surface trenches and re-sampling of 17 trenches from the 1940s. Table 4-5 summarises these exploration activities.

Table 4-5: Summary of Works 1972 to 1973

Operation	Unit	Scope of Work	
		Designed	Actual
Core Drilling	m	1,630.0	1,744.0
Trenches	m^3	1,370.0	1,411.5

Drillholes Nos. 1 and 7 (OB1) were later re-drilled in 1972, due to poor core recovery, but the re-drill of No. 7 was much lower than that from the initial hole. From this 1972-73 drilling programme, low core recovery was reported and it is a critical quality control issue, but the statistics do not display any obvious bias trends between core loss and vanadium grade levels, and this supports the 1973 Komarnitski report's conclusion, after exhaustive tests on artificially induced core losses. Although this drillhole No. 7 had a core recovery of only 29 %, probably due to very soft amorphous carbon content, the intersected V_2O_5 % grade is relatively high at 0.71 % - FAR's nearest drillhole (B313) is only 25 m away and the intersecting grade is also high at 0.91 % V_2O_5. It is also interesting that there is a core recovery improvement trend with successive drillholes and culminates in acceptable core recoveries at over 90 % for the later drillholes.

Particular emphasis was placed on prospecting the NE limb of OB1, with seven drillholes. Here one drillhole was positioned on each of the five prospecting gridlines (0-4). The SW limb of OB1 was intersected by two drillholes (Nos. 2 and 14) located on survey grid lines 1 and 4.

The large OB2 was intersected on the NE limb by only three drillholes (Nos. 8, 9, 12) located on exploration grid lines 4, 5 and 7, and the SW limb by only two boreholes (Nos. 4 & 10) at exploration grid line 4.

The north-eastern limb of OB3 was intersected at depth by two drillholes (Nos. 6 & 13) located on survey grid lines 4 and 6. No drillholes intersected the SW limb of this synclinal structure.

4.2.3.2 SURFACE OXIDE SAMPLING

Surface trenches were sampled by a continuous channel with a 5 x 10 cm cross-section. In rare cases, when it was necessary to sensitively test smaller intervals (0.2 to 0.30 m) the trench cross-section was increased to 10 x 20 cm.

In total, 1,808 trench and core samples were taken. Core and trench samples were prepared in a similar manner (crushing, grinding and pulverisation) and assayed at the Central Laboratory of the South Kazakhstan Geological Department for the following components: SiO_2; V_2O_5; C; Al_2O_3; P_2O_5: Fe_2O_3; TiO_2: H_2O*.* One part of the sample was additionally analysed and tested for CaO and Mg content*.* In addition, comprehensive spectral semi-quantitative analysis was done for all lab samples.

The total number of samples taken for spectral assays was dependent on the overall thickness of the tested layer; for thicker layers more than one sample was taken. Each of the thin intercalations was defined by one sample only. Chip samples for spectral analysis were taken at the intervals ranging from 10 to 20 cm (depending on the layer thickness) with subsequent compositing of them into one sample over intervals ranging from 0.5 to 5.0 m, but normally 1.5 to 2.0 m.

To test the physical properties of the ores and host rocks, samples were taken to measure the bulk volume density, true density, porosity and water absorption capacity.

4.2.4 1990-1992 TASHKENT INSTITUTE

In 2010, FAR discovered that a major exploration programme was completed during the period from 1990 to 1991, by the Tashkent Research and Development Institute, but no detailed information has been obtained. However, in a 1992 summary report, it stated that 95 diamond drillholes were completed totalling 6,400 linear metres, plus 20 surface trenches. The drilling was designed to target the primary mineralisation at depth: this included 38 drillholes for OB1 and 22 drillholes for OB2 and OB3. The depth of the drillholes ranged from 20 to 205 metres. The ore intersections targeted 10-20 m depth intervals, along the dip of the orebodies, from surface to a depth of 80 m and targeted overall depths from 80-140 m. The NE limb drilling of OB1 targeted the vanadium layer to its deepest levels, based on a grid network of 400x40x20 m. The results essentially confirmed the vanadium pentoxide grade trends for the primary zone, as reported in the 1973 feasibility report by Komarnitski.

4.2.5 2010-2011 FAR'S DIAMOND DRILLING PROGRAMME

GMR prepared a proposed drilling programme for FAR which was submitted in 2009, and included QA/QC issues (discussed in Section 4.5.2). A summary of the proposed drilling programme is shown in Table 4-6.

Table 4-6: Drilling Proposal Summary

Ore Body	Number of Holes	Total m	Average metres per hole	JORC target
1	13	1,621	125	Indicated
2	23	3,727	162	Indicated
3	9	1,209	134	Inferred
4	3	461	154	Inferred
Total	48	7,018	146	

Initially, only alternate exploration profiles were to be drilled and, based on the assessment of the vanadium grades and structural behaviour of the vanadium layer, final decisions were made as to the optimum spacing for achieving the resource objectives.

Ideally, it would have been conducive to have planned the drillhole programme for rigs having the capability of drilling up to -45 degrees, so as to maximise target objectives with the least number of holes and to allow a more orthogonal angle of penetration with the ore layers. Unfortunately, the only drill rigs available had a drilling angle of up to -60 degrees from the horizontal and these steeper angles may induce the drill string to deflect along sub-vertical bedding and therefore be more difficult to intersect the vanadium layer. Additionally, only a double-tube core barrel was available from the contractor but with a guarantee of at least 90 % core recovery within the ore horizons. The minimum core size was NQ (47.3 mm), and HQ (63.5 mm) core size drilling would be particularly used for targeting and collecting geotechnical and metallurgical core samples as part of a later but separate drilling programme.

A summary of the actual drilling programme undertaken in 2010 and 2011 is shown in Table 4-7.

Table 4-7: Summary of Drillholes Completed

Number of Drillholes Completed	Metres Drilled	Orebody #
24	1,945.4	OB1
7	931.6	OB2
3	387.2	OB3
0	0	OB4
Total	3,264.2	

Although only 13 drillholes were proposed initially for OB1, as seven acceptable historical drillholes were already available, the FAR drilling was expanded and resulted in a total of 21 successful drillholes, plus three drillholes which failed to intersect the orebody. The additional drillholes were mainly completed at intermediate drilling profiles, to confirm the continuity, structure and grade tenor of the vanadium layer. The results are shown below in Table 4-8.

Table 4-8: FAR Drilling Summary Results for OB1

Drillhole ID	OB	Profile	EOH Depth [m]	V_2O_5%	Length [m]	True Thick [m]	Comments
B110	1	0	72.8				Missed orebody
B111	1	1	64				Missed orebody
B112	1	2	92.4	0.76	20.7	12	
B112_5	1	2.5	35	0.63	12	7	
B113	1	3	48.1	0.61	6.4	3	
B113_5	1	3.5	105.5	0.66	2.9	2.9	
B114	1	4	62.5	0.73	13.6	10	
B115	1	5	71	0.5	19	?	Nose of syncline
B211	1	1	124.6	0.66	3.5	?	Complex structure
B212	1	2	102.9				Missed orebody
B212_5	1	2.5	73.6	0.67	22.2	10	
B213	1	3	126.2	0.65	25	13	
B213_5	1	3.5	74.1	0.7	13	8	
B214	1	4	76.8	0.71	8	7	
B215	1	5	50	0.68	10.3	10	
B311_bis	1	1	66.2	0.61	11.5	5.7	
B312	1	2	66	0.56	16.5	12	
B312_5	1	2.5	36	0.63	12.5	8.5	
B313	1	3	67.6	0.91	13.2	10	
B313_5	1	3.5	118.8	0.73	24.5	13	
B314	1	4	149	0.68	31.7	30	Syncline axis
B315	1	5	50.8	0.83	29.2	35	Complex syncline nose
B412	1	2	134.3	0.65	10.2	8	SW limb
B412	1	2	134.3	0.7	8.3	8	NE limb
B412_5	1	2.5	77.2	0.63	19.3	11	

For OB2, a total of 23 drillholes were proposed, though only seven holes have been completed at the date of this report, of which two drillholes missed the orebody. These results are shown below in Table 4-9.

Table 4-9: FAR Drilling Summary Results for OB2

Drillhole ID	OB	Profile	EOH Depth [m]	Comments
B124	2	4	73	Missed orebody
B125	2	5	88.4	
B224	2	4	142	Missed orebody
B225	2	5	94.3	
B324	2	4	111.2	
B325	2	5	231	
B425	2	5	191.7	

For OB3, a total of only nine drillholes was proposed, though only three holes have been completed at the date of this report. The results of this FAR drilling for OB3 suggest that structural re-interpretation of the vanadium layer is required, before continuing the exploration programme, and this will also apply for OB2. This will not affect the expected global grades and tonnages, but is necessary to optimise the drilling targets. The results are shown below in Table 4-10.

Table 4-10: FAR Drilling Summary Results for OB3

Drillhole ID	OB	Profile	EOH Depth [m]	Comments
B135	3	5	93.0	Intersected ore zones need further structural interpretation
B235	3	5	114.0	
B335	3	5	180.2	

The next phase of the FAR drilling will specifically target OB3 and OB4, with OB2 being deferred to a later phase.

4.3 HISTORICAL RESOURCES AND RESERVES

4.3.1 1947 RESERVES (GKZ DEFINITION)

After its discovery in 1941, the Kazakhstan Geological Survey undertook extensive high quality resource appraisal between 1942 and 1947. From extensive sampling along a 40 km strike length of the mineralised structure, Soviet-type reserves were estimated in 1943, 1945 and 1947, into B, C1 and C2 categories. This also included comprehensive large bulk sampling in the 1950s and they reached the following conclusions:

- Consistent chemical composition in the ore zones.
- Vanadium grades are regular in the ore horizons and this also applies to uranium and molybdenum grades.
- Carbon may vary in grade more frequently and depends on the type of ore horizon. Carbon grades average 2.69 % in the oxide layer (mean depth 8 to 10 m) and 4.40 % in the primary horizons.
- Additional bulk sampling in the 1950s show that carbon grades can reach levels of 25 % within the ore layers.
- Carbon morphologically is represented by lenses, pockets and irregular shapes, and therefore tends to be more unequally distributed than other commercial minerals.
- Higher vanadium grades tend to have higher carbon concentrations.
- In the oxide zone, carbon concentrations are directly dependent on the degree of oxidation.

The Balausa vanadium GKZ reserves based on the 1947 published former Soviet-era type resources are displayed in Table 4-11.

Table 4-11: Official 1947 Reserve (GKZ) Summary

Category	Reserve [1000 t]	Mean grade V_2O_5 [%]	Contained V_2O_5 [t]
B	5,700	1.09	62,992
C1	13,500	1.08	146,076
C2	53,800	1.04	558,715
B+C1+C2	73,000	1.05	766,783

Under the former Soviet system, the deposit was declared a Group II (geologically uncomplicated) and for this relatively low complexity, it would be required that at least 20 % of the resource must be category B and 80 % as category C1, for mineral exploitation. Under these guidelines, it was assessed by Ankinovich et al, that for the vanadium classification, cross-section profiles for sampling should be spaced at 50 m to 100 m along strike (but ≤50 m for uranium) for each synclinal limb, and for the C1 category 400 m spacing would be required. In addition, for any declared C2 resource category, it would be sufficient to have an average spacing of 800 m along strike.

An example of a 1:500 scale drawing from the 1945 estimation is shown in Figure 4-1. A number of profile blocks for the NE limb of OB1 are displayed, and the construction of the B, C1 and C2 classification blocks in relation to surface trenches and underground exposures can be seen. The format basically allowed the projection of B category reserves on each profile from surface to a depth of the underground ore exposures and, where only trench data available, this was projected to the depth of the nearest adjacent underground level. The C1 category zones were projected as contiguous extensions downdip from the B blocks to a vertical depth of 20 m below and, underneath, the C2 blocks projected to depths ranging from another 20 m to 40 m, below the C1 zones. The outlined blocks were projected half-way to adjacent profiles, for volume determinations. For the 1947 reserves, it appears that the projection of the C2 resource at depth was much greater than in 1945, ranging from 200 to 400 m.

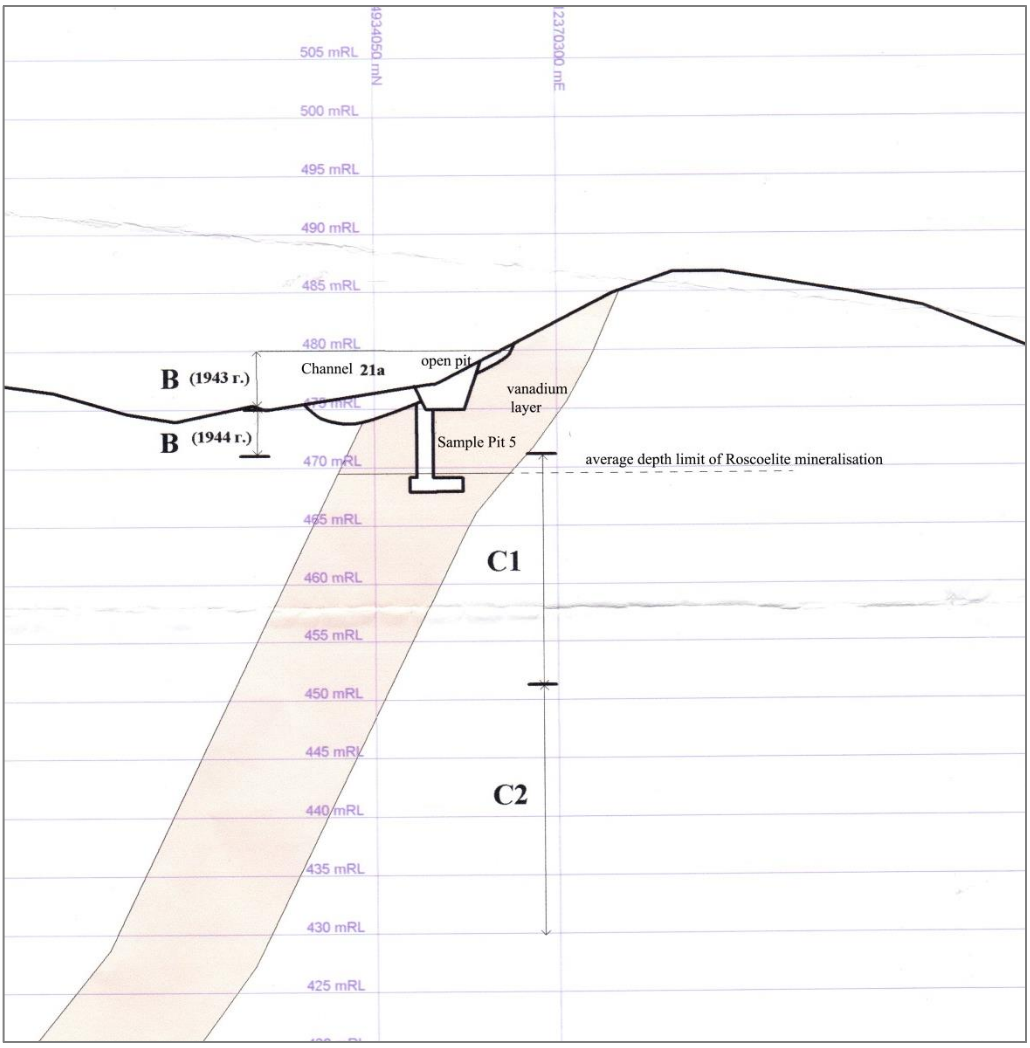


Figure 4-1: Example of a Reserve Block for NE Limb OB 1 - 1945 (Cross-Section Between Profiles 1 and 2).

Table 4-12 summarises the 1947 vanadium reserves. Although uranium reserves were also produced, the grades were low and considered as sub-economic in 1946 and have not been specifically assessed for this review. The former Soviet Union used very low uranium cut-off grades at 0.03 % U and 0.01 % for in situ leaching. It should be noted that uranium, molybdenum, rare earths and organic carbon are expected to be produced as by-products from the processing plant. The ore types are split into oxide and primary horizons and generally this appears to be based on some assumed level below the surface at which the oxide mineralisation becomes primary, as indicated by the decreasing abundance of the vanadium roscoelite mineral with depth. Ankinovich observed that roscoelite is abundant in the underground exposures, and this contrasts with the core material from two failed drillholes which contained insignificant amounts at depths at 80 m to 90 m. However, as an example, OB4 mineralisation was exposed mostly by surface trenches and only one shallow pit, and it is unclear as to how the oxide and primary zones could be confidently delineated for ore reserve estimations.

Table 4-12: 1947 Vanadium Pentoxide Ore Body Reserve Summary

	Category B		Category C1		Category C2		Category B+C1+C2	
	Tonnes	V_2O_5%	Tonnes	V_2O_5%	Tonnes	V_2O_5%	Tonnes	V_2O_5%
OB1								
Oxide	258,212	0.98	148,194.6	0.97	40,953	0.81	447,359	0.96
Primary	465,669	1.01	749,437	0.98	891,912	0.98	2,107,018	0.98
Total	723,881	1.00	897,631.6	0.97	932,865	0.97	2,554,378	0.98
OB2								
Oxide	581,100	1.10	0	0.00	0	0.00	581,100	1.10
Primary	1,655,688	1.11	5,425,901	1.10	6,504,416	1.10	13,586,005	1.10
Total	2,236,789	1.11	5,425,901	1.10	6,504,416	1.10	14,167,105	1.10
OB3								
Oxide	369,330	1.04	210,956	1.08	354,195	1.07	934,481	1.06
Primary	532,033	1.03	2,653,774	1.05	3,222,492	1.05	6,408,299	1.05
Total	901,363	1.03	2,864,730	1.05	3,576,687	1.05	7,342,780	1.05
OB4								
Oxide	324,452	1.18	0	0.00	0	0.00	324,452	1.18
Primary	705,416	1.21	2,186,809	1.13	1,203,846	1.11	4,096,072	1.14
Total	1,029,868	1.20	2,363,519	1.13	1,683,314	1.12	5,076,701	1.14
Totals								
Oxide	1,533,095	1.09	359,150.2	1.03	395,148	1.04	2,287,393	1.07
Primary	3,358,806	1.10	11,015,922	1.09	11,822,666	1.08	26,197,394	1.08
Total	4,891,901	1.10	11,375,072	1.09	12,217,814	1.08	28,484,786	1.08

Note: this OB1 reserve tonnage is only about 10 % of the JORC (2012) resource model, though the V_2O_5 % grade is higher, because of scant knowledge of grades within the primary zone.

Table 4-12 above is based on detailed historical estimations for individual blocks. Surprisingly, the reported overall global resource tonnage, as based on the extent of exploration for the whole of the Balasausqandiq deposit (see Table 4-13), as submitted to the GKZ (State Reserves Committee) in 1947, totalled 73 million tonnes at a 1.05 % V_2O_5 grade, though GMR has not seen any detailed orebody resource inventory to substantiate these figures. However, these reserves were declared as “off-balance” (i.e. not ready for exploitation but potentially economic) because of a shortfall in the understanding of the deposit, especially due to inadequate exploration within the primary zone of mineralisation at depth. This same reserve figure of 73 million tonnes, was re-confirmed by the GKZ on 1 January 1997: (see Table 4-14) but most of this reserve V_2O_5 grade was based on the oxide surface trench sampling and shallow underground exposures, and it is this vanadium grade tenor, within the oxide/transition zone, which had also been applied to the primary zone mineralisation at depth, because erroneously Ankinovich assumed that the oxide and primary zones had similar vanadium grades.

Table 4-13: Reserves as of 01.01.1947

Category	Ore Type	Ore Reserves Ktonnes	Ore grade (%)		Metal reserves		Remarks
			V_2O_5	Uranium	V_2O_5 Tonnes	Uranium tonnes	
B	OO	1,676.7	1.09	0.0012	18,335.7	193.3	OO = oxidized ore
	PO	4,010.7	1.09	0.0011	43,782.4	456.0	PO = primary ore
	Total	5,687.7	1.09	0.0011	62,118.1	649.3	Note: a conversion factor was used for the uranium grade in the ore zones
C1	OO	87.5	0.98	0.0011	853.9	9.5	
	PO	13,438.1	1.08	0.0011	144,788.1	1,416.6	
	Total	13,525.6	1.08	0.0011	145,642.0	1,426.1	
B+ C1		19,213.0	1.08	0.0011	207,760.1	2,075.4	
C2		53,814.8	1.04	0.008	557,247.7	4,557.2	
B+ C1+ C2		73,027.8	1.05	0.009	765,007.6	6,632.6	

Note: For uranium in terms of $U_3O_{8,}$ the grade is 0.011 %, which is about 20 % higher than the OB1 JORC (2012) resource.

Table 4-14: GKZ Confirmation of Off-Balance Reserves 1 January 1997

Deposit name and types of minerals	Units of measurement	Reserves category (in accordance with the extent of their exploration)			
Balasausqandiq deposit		**In-balance reserves**		**Off-balance reserves**	
Vanadium-bearing shales		**C1**	**C2**	**C1**	**C2**
Ore	tonnes				73,000,000
V_2O_5*	tonnes				764,900
V_2O_5 % grade	tonnes				1.05%
Ore	tonnes				73,028,000
Molybdenum **	tonnes				15,336

Note:

** these figures confirm a weighted mean V_2O_5 grade of 1.05 %,*

*** defined within the estimation boundaries of the vanadium and uranium reserve – grade of Mo is 0.021 % (or in terms of MoO_3 = 0.032 %, which is similar to the JORC OB1 grade.*

4.3.2 1973 RESERVES

The GKZ approved further exploration drilling to produce C2 category reserves by targeting the primary vanadium at depth. Drillhole spacing along strike was nominally set at 800 m. From 1971 to 1972, 14 exploration drillholes were designed to intersect at depth the primary mineralised vanadium zone along the limbs of the synclinal folds, for OB1, OB2 and OB3. The primary focus was based on drilling five holes along the NE limb of OB1, plus two drillholes along the SW limb of the syncline. Three drillholes intersected the NE limb of OB2, but only one drillhole intersected mineralisation on the SW limb, as the other drillhole did not intersect the target zone due to structural complications. For OB3, two drillholes intersected the NE limb of the syncline but the SW limb was not drilled. Overall, the drillhole spacing along strike was based on 800 m, but because of terrain accessibility, actual profile spacing ranged from 560 m to 910 m. This spacing would allow the resource to be defined within the C2 category. Additionally, 22 new trenches were excavated for oxide ore zone channel sampling, plus 18 old trenches from the 1940s were deepened and re-sampled, but the assay results from only four of these trenches were received by GMR. The comparison between the 1947 and 1973 results for these four OB1 trenches were inconclusive, with two trenches being quite similar in grade, but the 1947 oxide grades for the other two trenches were higher than the 1973 results. A single bulk volume density figure for this resource was set at 2.5, with no adjustment for a lower density in the oxide zone, as it was considered that the influence of the oxide material was too insignificant. The depth of oxidation was set at 7 m below the surface for the vanadium layer and 4 m for the low-grade siliceous layer which is stratigraphically lower but contiguous to the vanadium layer. Table 4-15 summarises the 1973 reserves.

Table 4-15: 1973 Summary Reserves

Ore Body	Fold Limb	Tonnes	SiO_2%	V_2O_5%	C%	P_2O_5%
1	NE	4,149,007	67.94	0.72	12.41	0.48
1	SW	6,263,651	59.33	0.7	12.65	0.69
Total 1		10,412,658	62.76	0.71	12.55	0.61
2	NE	4,840,687	65	0.68	12.53	0.62
3	NE	8,111,080	68.42	0.63	11.83	0.72
Totals		23,364,425	65.19	0.68	12.30	0.65

Note: single bulk density at 2.5 and the oxide and primary grade results were combined.

It is understood that these reserves were NOT submitted to GKZ for approval.

Figure 4-2 below shows the reserve blocks projected onto a longitudinal section for the NE limb of OB1. It includes the drillholes involved in the estimation at each exploration profile line and also surface trenches used in estimating the oxide zone. The blocks are split into vanadium and siliceous layers. The calculated volumes and tonnages for each block are also shown. Because the grades within the primary zone were found to be so consistent, all blocks received the same average grade for each specific synclinal limb. This average grade approach was also applied to the oxide zone.

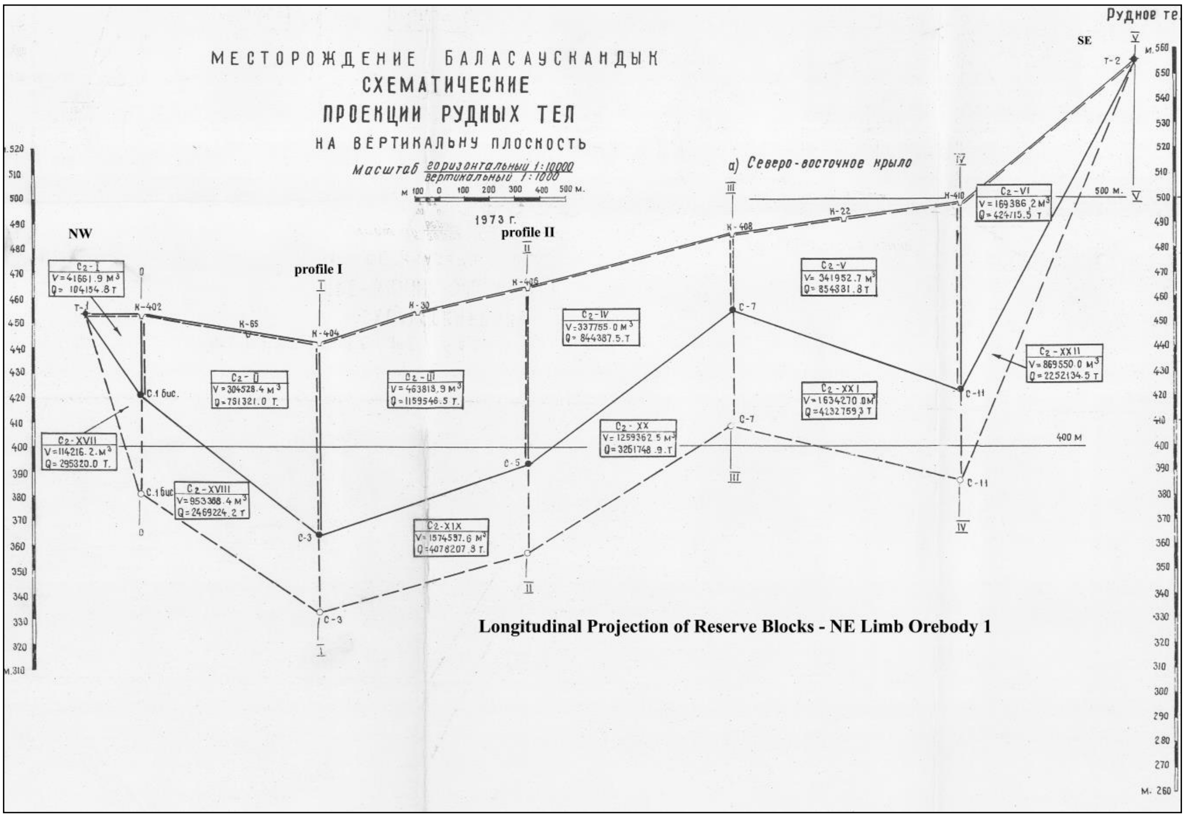


Figure 4-2: Showing Reserve Blocks for NE Limb of OB1 - 1973

4.3.3 DISCUSSION ON 1947 & 1973 RESERVES

The statistics confirmed the consistent grade distributions of the V_2O_5 levels within the vanadium layer, as reported in the historical documentation. Even though the oxide zone exhibited higher grades, there is still uniformity in the percentage level of vanadium content. This degree of homogeneity in the grade distributions allowed the recent FAR exploration drillholes to have a wider spacing than expected and experimental variogram ranges along strike also provides weight for a wider drillhole spacing pattern.

Comparison between the 1947 and 1973 reserves show a dichotomy in the reported vanadium grades. In the 1947 Ankinovich report, it was concluded that both the oxide and primary zones are similar in grade at about 1 % V_2O_5. In the 1973 Komarnitski report, the conclusion was that the vanadium grade is higher in the oxide, and their results basically confirmed the 1947 oxide vanadium grade levels. Most of the 1947 samples were from the oxides (and transition zone) from surface trenches and some shallow underground exposures <30 m: core drilling to intersect the primary mineralisation at depth was unsuccessful. Two shallow shafts reportedly exposed primary vanadium, shafts #2 and #4 in OB1, and the overall weighted grade was 0.93 % V_2O_5 (0.73 % and 1.4 % respectively). The 1973 results were mostly based on drill core sampling but the core recovery is low, so although there is no statistical grade bias, there is a need to generate reserves with good core recoveries. Fundamentally, the 1940s and 1970s reserves are based on two different types of samples, with the earlier results constrained to the upper oxide zone of the deposit, containing only an estimated 5 % of the total resource, and the latter results, based on the primary mineralised core samples at depth where core loss was high. However, there is unanimity about the average V_2O_5 grade within the oxide zone and this has been further confirmed from the results of 55 samples sent to the Intertek laboratory in Perth, Australia in 2007.

4.3.4 1990 - 1992 RESERVES

As a result of a major exploration programme in 1990 to 1991, by the Tashkent Research and Development Institute, reserves were updated (though declared as unofficial), but only a 1992 summary has been obtained by FAR. The results basically confirmed the vanadium pentoxide grade trends for the primary zone, as reported in 1973 by Komarnitski. It also reported that the B and C1 reserve categories have increased significantly, by the upgrading from lower categories.

Main points extracted from 1992 summary report:

- Classification of the reserves and their tonnages remain unchanged since the last reserve work was completed by the Geological Survey of South Kazakhstan in 1973 (Komarnitski). It also re-confirmed a general reduction in the vanadium grade from the oxide/transition surface layer to the primary mineralised vanadium layer at depth.
- In the years 1990-91 the exploration Geological Company #6 (now Kazatomprom) by the Vostokredmet State Enterprise, conducted additional geological investigation on OB1, to qualitatively define the grade of ore at depth, as the earlier work in 1947 was mainly related to the surface and subsurface part of the deposit.
- Exploration of this third phase (1990 - 1991) and, from the drilling results of the second phase (1972), resulted in the OB1 C2 category reserves being upgraded to categories B+C1 - however these reserves were not official.
- The grade of the vanadium pentoxide content from the later surface exploration phases, confirmed the oxide zone results of 1942-1947: the difference on average for OB1 was only 1 %. However, because of the grade dichotomy from oxide to the primary zone, changes in the content of vanadium pentoxide at different depths from the surface were recorded and, results presented in Table 4-17. Here the data were derived from 89 complete drillhole intersections of the vanadium ore horizon and split according to depth intervals. This showed that the 1942-47 primary ore grades were exaggerated, and re-confirmed the primary grade estimates from the 1971-1972 exploration results.

Table 4-16: Reserve Update – 1947 & 1991 Comparison

Ore Bodies, Blocks, Categories of Reserves	Categories of Reserves		Type of Ore – Oxide or Primary	1947 Reserves		1991 Reserves	
	1947	1991		Ore [Kt]	V_2O_5 [%]	Ore [Kt]	V_2O_5 [%]
OB1							
Category B	B	B	O	258.2	0.98	258.2	0.98
Blocks 1947	B	B	P	465.7	1.01	465.7	0.75
Total	B	B	O + P	723.9	1.00	723.9	0.83
Category C1	C1	C1	O	65.2	0.99	65.2	0.99
Blocks 1947	C1	B+C1	P	1,090.2	0.96	1,090.2	0.72
Total	C1	B+C1	O + P	1,155.4	0.96	1,155.4	0.74
Category C2							
Blocks 1947	C2	C2+C1	P	382.9	0.95	382.9	0.71
Blocks 1991	C2+P1	B+C1	O	406.4	0.96	406.7	0.96
Blocks 1991	P1	B+C1	P	2,942.4	0.73	2,942.4	0.73
Total	C2+P1	B+C1+C2	O+P	3,731.7	0.78	3,731.7	0.76
Total OB1		B+C1+C2	O	729.8	0.97	729.8	0.97
Total OB1		B+C1+C2	P	4,881.2	0.83	4,881.2	0.73
Total OB1		B+C1+C2	O+P	5,611	0.85	5,611	0.76
OB4							
Category B	B	B	O	324.5	1.18	324.4	1.18
Blocks 1947	B	B	P	705.4	1.2	705.4	0.9
Total	B	B	O+P	1,029.9	1.19	1,029.8	0.99
Category C1(all blocks)	C1	C1	P	2,363.5	1.13	2,363.5	0.85
Category C2							
Blocks 1947	C2	C2	P	586.8	1.12	586.8	0.83
Blocks 1991	P1	C2+P1	P	425.6	1.08	425.6	0.8
Total	C2+P1	C2+P1	P	1,012.4	1.1	1,012.4	0.82
Total OB4	B+C1+C2	B+C1+C2	O	324.5	1.18	324.5	1.18
Total OB4	B+C1+C2	B+C1+C2	P	4,081.3	1.14	4,081.3	0.85
Total OB4	B+C1+C2	B+C1+C2	O+P	4,405.8	1.14	4,405.8	0.87
OB2							
Category B	B	B	O	377.9	1.12	377.9	1.12
Category B	B	B	P	1,164.6	1.14	1,164.6	0.85
Category c1	C1	C1	P	2,934.7	1.12	2,934.7	0.85
Total OB2	B+C1	B+C1	O+P	4,477.2	1.13	4,477.2	0.87
Total OB 1,2,4				14,494	1.02	14,494	0.83

Table 4-16 shows a comparison between the official GKZ 1947 reserves and the 1991 reserves, and fundamentally this exploration work confirmed the 1947 oxide grades, but shows a basic downgrading of the primary ore grades, as based on the drillhole core sample results. There were no reserve tonnage changes.

Table 4-17: Vanadium grade at depth intervals 1990-1992 (OB1)

#	Depth intervals (m)	Distance (m)	The number of complete intersections of the ore horizon	Grade of vanadium pentoxide	
				Average (%)	Range (%)
1	10-25	15	17	0.79	0.66-0.99
2	25-40	15	13	0.73	0.67-0.86
3	40-60	20	17	0.72	0.53-0.83
4	60-80	20	11	0.73	0.60-0.83
5	80-140	60	8	0.74	0.64-0.93
Weighted average				**0.74**	

Table 4-17 shows that, starting below the depth interval of 10-25 m, the ore is primary in nature with a related drop in V_2O_5 grades. This substantiated the 1973 Komarnitski report, where an average grade of 0.63 % V_2O_5 was recorded for 14 drillholes within the primary zone. Note however, that the 1972 former Soviet-era drillholes which were selected for the current FAR JORC resource estimate, OB1, averaged 0.67 % V_2O_5, which is similar to the global grade from the FAR drilling results.

For OB1, most of the C2 category had been upgraded to B + C1 and the reserves for these orebodies were calculated by the typical former Soviet-era method of blocks projected onto orebody slices. These historic updated results are presented in Table 4-18 below.

Company #6 conducted studies of a large number of samples for the content of Rare Earth Elements (REE). The results for OB1, 2, and 3 are listed in Table 4-34.

Table 4-18: Example of Reserve Blocks 1990–1991 OB1

Reserves of blocks explored in 1990-91 (OB1), reserve blocks 17 and 19, calculated within pit contours.

Block #	Category of reserves		Characteristics of block (m)			Amount of ore [m^3]	Unit weight	Reserves		
	1947	1991	Distance [m]	Depth	Capacity			Ore [t]	Grade [%]	V_2O_5 [t]
16	C2	B+C1	1085	135	8	1171.8	2.00	2344	0.78	18280
	Oxidized					67.7	1.73	117.2	0.98	1148
17	P1	C1	400	73	8	249.6	2.00	499.2	0.78	3894
	Oxidized					28.9	1.73	50	0.98	490
18	P1	C1	900	147	7	926.1	2.00	1852	0.79	14632
	Oxidized					42.8	1.73	74	0.95	703
19	P1	C2	1100	61.2	7	471.2	2.00	937.8	0.73	6919
	Oxidized					95.5	1.73	165.2	0.96	1578
20	P1	C2	380	70	8	212.8	2.00	425.6	1.08	4596
								Average Grade	**0.80**	

4.4 GEOLOGY AND MINERALISATION

4.4.1 GEOTECTONIC SETTING

Understanding the tectonic phases is important in helping to crystallise the somewhat complex structural features found at the deposit, and this understanding can help improve structural interpretation for realistic modelling of the vanadium ore layers. Here the Karatau mountain range has been affected by tectonic events over a lengthy geological time-frame. Regionally, the Karatau Fault System; forms a crustal-scale zone of strike-slip dominated transpressional tectonics, having undergone multiple phases and styles of deformation during a protracted history of reactivation from the Neoproterozoic to the Cenozoic.

GMR reason that the initial geological history appears to be related to the break-up of the Rodinia supercontinent at >825 Ma and the subsequent amalgamation of the supercontinent of Gondwana during the Cambrian: Neoproterozoic-Cambrian Transition. At that time, the Balausa area was centred at a primordial rift system, which developed along old suture(s) which formed when ancient continental masses collided billions of years ago; this rifting occurred within the Gondwana craton, where the ancient suture zone channelled and focussed super-plume activity from deep within the earth’s mantle. The formation of the deposit, within such a supercontinent, appears to straddle the time-frame from late Precambrian to mid-Cambrian age, as supported by the occurrence of Neoproterozoic inliers within the ore field (see geological map Figure 4-9). This also appears to be similar to the situation in the Yangtze Platform of South China, where the Balasausqandiq deposit has been shown to be associated both spatially and geologically: the Balausa district would therefore have inherent deep-seated fault structures that became reactivated after continental accretion within the South Kazakhstania domain, and which eventually resulted in uplift during the Caledonian-Himalayan orogeny, forming the Karatau mountain range in the Cenozoic era (Late Pliocene epoch at about 2.5 Ma).

4.4.1.1 GLOBAL TECTONICS

Balausa is tectonically situated in the western part of the Central Asian Orogenic (Tectonic) Belt (CAOB), and centred between the East European (Baltica), Siberian and Tarim cratons (see Figure 4-3 - tectonic map of Kazakhstan showing Siberian and Tarim cratons) and has a complex tectonic history, as indicated from the early–middle Palaeozoic palaeogeography of Kazakhstan, on the basis of Ordovician and Devonian palaeomagnetic results: Mikhail L. Bazhenov et al 2012 (1). Such a geodynamic evolution of the western CAOB is often described as a tectonic mosaic generated by complex interaction of ocean basins, island arcs and microcontinents (microplates) with a Precambrian basement. The Karatau mountains have been profoundly affected by tectonic events over a protracted geological time-frame and, especially more recently, since the Oligocene (28 Ma), following collision of both the Indian and Arabian with the Eurasian continents of the Alpine-Himalayan orogeny; when dramatic reversion tectonics occurred causing uplift and formation of the Karatau mountain range.

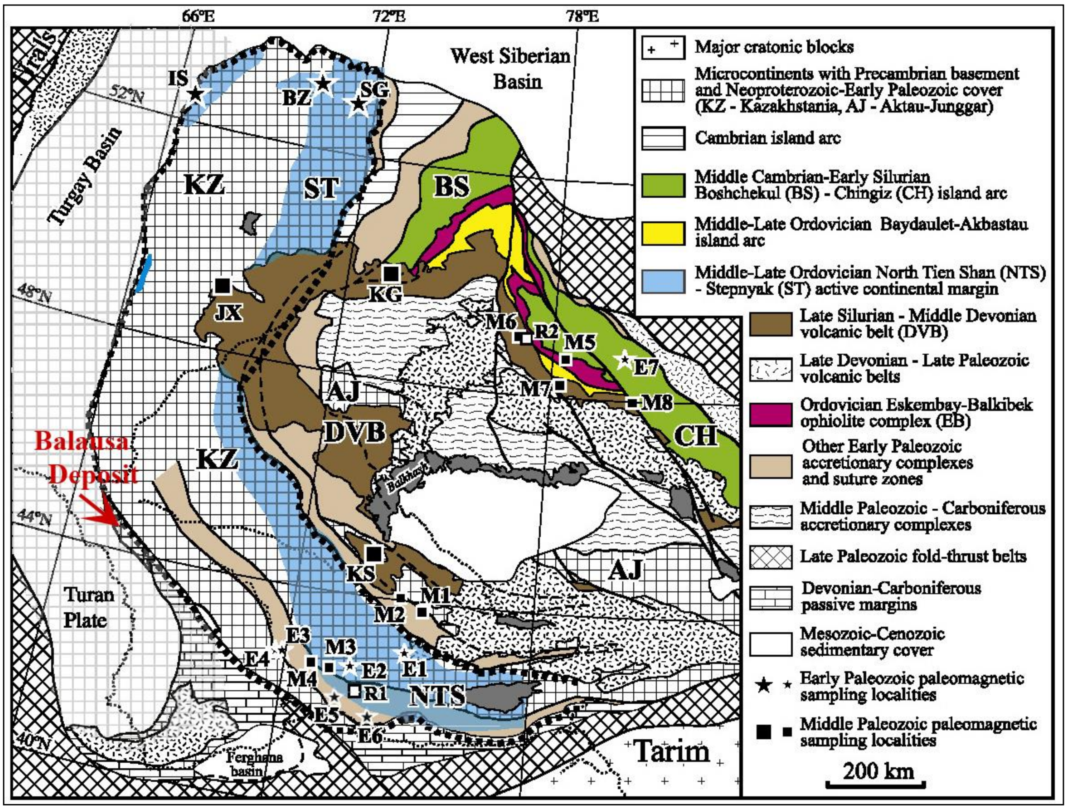


Figure 4-3: Tectonic Map of Kazakhstan

Note: Thick solid lines denote major faults (dashed where inferred). Thick dotted line denotes the exposed parts of Kazakhstania; its western extension under the Mesozoic-Cenozoic cover is in faint grey pattern.

During the Neoproterozoic-Cambrian, which saw the break-up of the supercontinent Rodinia (>825 Ma) and the subsequent amalgamation of the supercontinent Gondwana, Lawrence Och (PhD thesis - about the Neoproterozoic-Cambrian transition in China) (2) stated that the well preserved sedimentary succession from the Precambrian-Cambrian-Transition (Pc-C-T) represents a unique archive of ancient geochemical conditions on the Earth's surface, based on a significant increase of Mo, V and U enrichment in black shales across this Pc-C boundary. The evidence for predominantly anoxic-ferruginous and even intermittently euxinic conditions in the water column across the Pc-C-T boundary and, significant regional variations in geochemical parameters, unravel complex interactions between ocean chemistry, platformal configuration and palaeontology.

Based on both palaeobiogeography and palaeomagnetism, it has been possible to reassemble the spatial positions of the orogenic terranes of the Early Palaeozoic; and this is specifically related to the Kazkhstanian terranes, which forms part of the CAOB area of peri-Gondwana. Three major clusters of early Palaeozoic terranes can be recognised within the Kazakhstanian orogen, of which the southern cluster includes the Chu-lli, North Tyan Shan and Karatau-Naryn terranes and which were amalgamated together by the late Silurian (Popov et al, 2009 (3) : see Figure 4-4 below). The Karatau-Naryn terrane appears to be unique within the Kazkhstanian terranes in showing clear distinct biogeographic affiliation with South China (Yangste Block), North China, Baltica, Avalonia and Laurentia during late Cambrian (Javier Alvaro 2012 (4)). The Karatau–Naryn terrane was indubitably a part of South China at the beginning of the Palaeozoic, and probably rifted shortly before the mid-Cambrian (Holmer et al 2001 (5)). GMR submit that this provides additional cogent evidence that explains the geological environmental similarity with the vanadium deposits in South China.

GMR conclude that such rifting with graben development, as relatively shallow marine basins, close to continental Proterozoic margins, is supported by the presence of underlying basal conglomerates (Cm2bK: see stratigraphic legend in Table 4-19) to the carbonaceous vanadium layer at the Balasausqandiq deposit and is compatible with the findings of local and regional faunal marine fossils and, in addition, supports geochemical modelling for vanadium deposition in such marine environments.

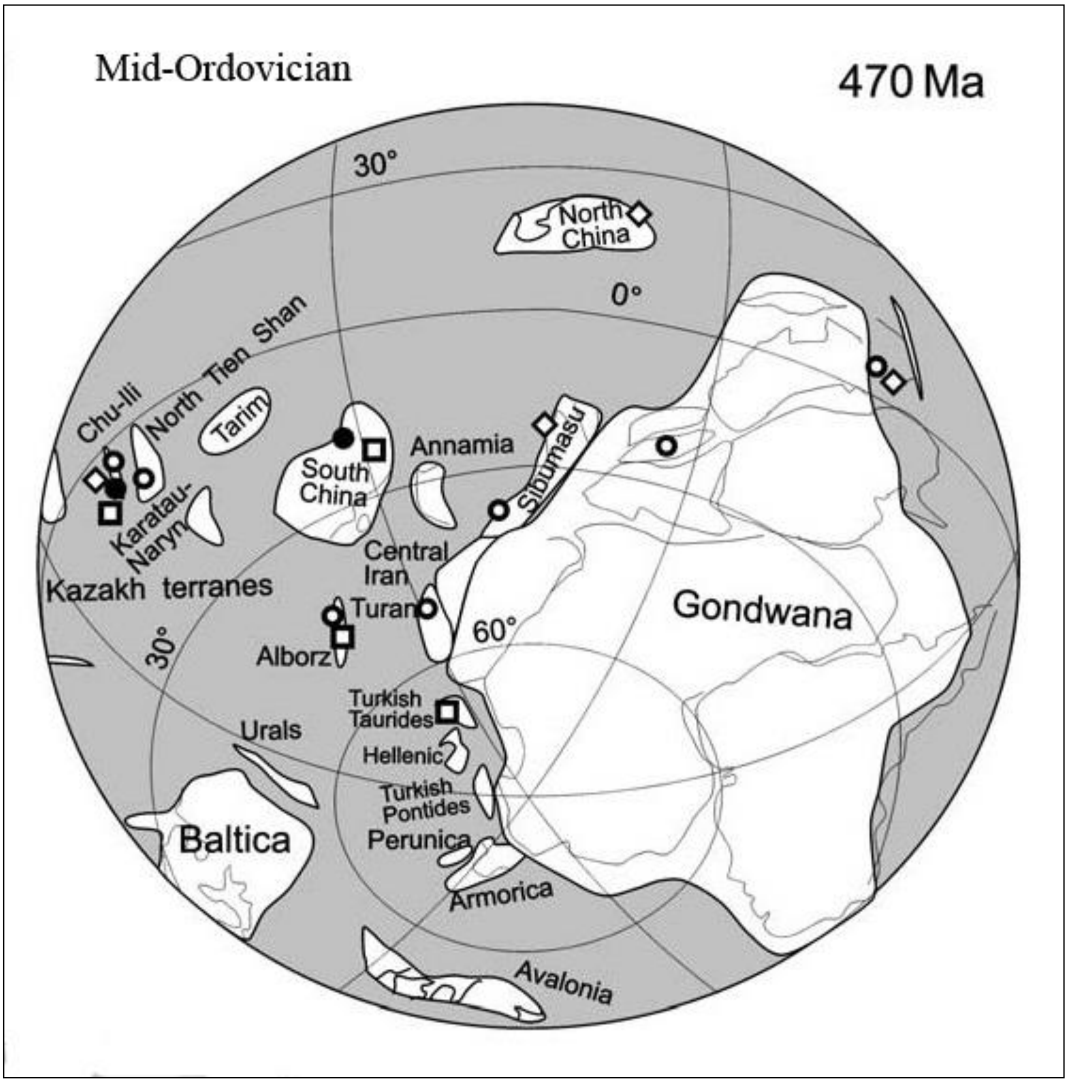


Figure 4-4: Global Reconstruction of Mid-Ordovician Continents, as Based on Distribution of some Characteristic Brachiopod Genera by Popov et al 2009 and modified by GMR 2013

Reference to Figure 4-4, three major clusters of early Palaeozoic terranes can be recognised in the Kazakhstanian orogen. The southern cluster includes three major crustal terranes (i.e. Chu-Ili, North Tien Shan and Karatau-Naryn), which were amalgamated together by the Late Silurian (Popov et al., 2009 (3)). Note especially the relatively close spatial proximity of South China with the Kazakh terranes.

According to Thorsvik and Cocks 2009 (6), the Kazakh terrane assemblage did not amalgamate with the larger Kazakhstania terrane until Late Palaeozoic. Many of those terranes have Precambrian cores (e.g. as found at Balausa) and distinctive Lower Palaeozoic successions and faunas.

The overall collective stratigraphic similarities between the Neoproterozoic-Cambrian sequences of these microcontinents were noted by Ankinovich in 1962 (7). Chumakov 2010 (8) also noted the stratigraphic and faunal similarities between many Kazakhstan and Kyrgyz microcontinents and suggests their possible close proximity in Late Neoproterozoic and Cambrian.

4.4.2 REGIONAL GEOLOGY

The Karatau mountain range, composed of the Greater Karatau (location of the Balasausqandiq deposit) and Lesser Karatau mountains, divides a large intermontane basin into the NE Chu Sarysu and SW Syrdarya sub-basins. These basins are filled with Devonian to Tertiary sediments with a thickness of up to 8,000 m, are hydrocarbon rich and also contain notable uranium deposits. Oil and gas is mainly confined to the Palaeozoic formations, but atypical gas has been discovered within highly fractured metamorphosed Proterozoic basement rocks. Figure 4-5 illustrates the location of the Balasausqandiq deposit, Figure 4-6 shows a vertical section of the Chu Sarysu basin, while Figure 4-7 illustrates a diagrammatic cross section of the relationship between the mountains the sub-basins.

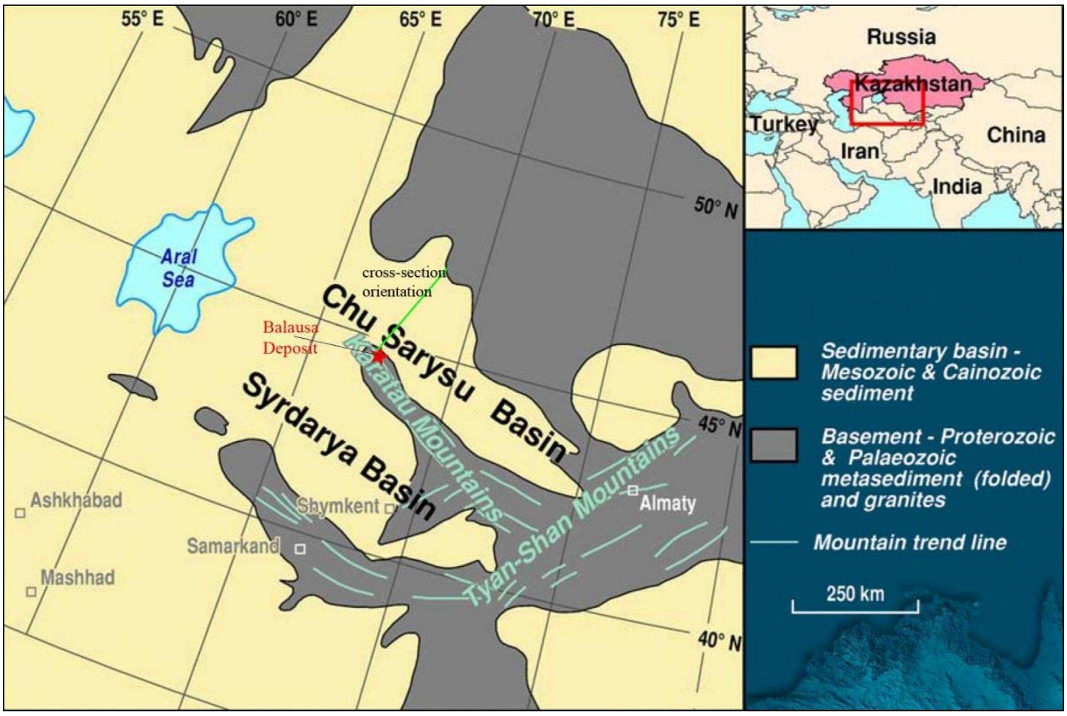


Figure 4-5: Main Structural Elements Showing the Location of the Balasausqandiq Deposit

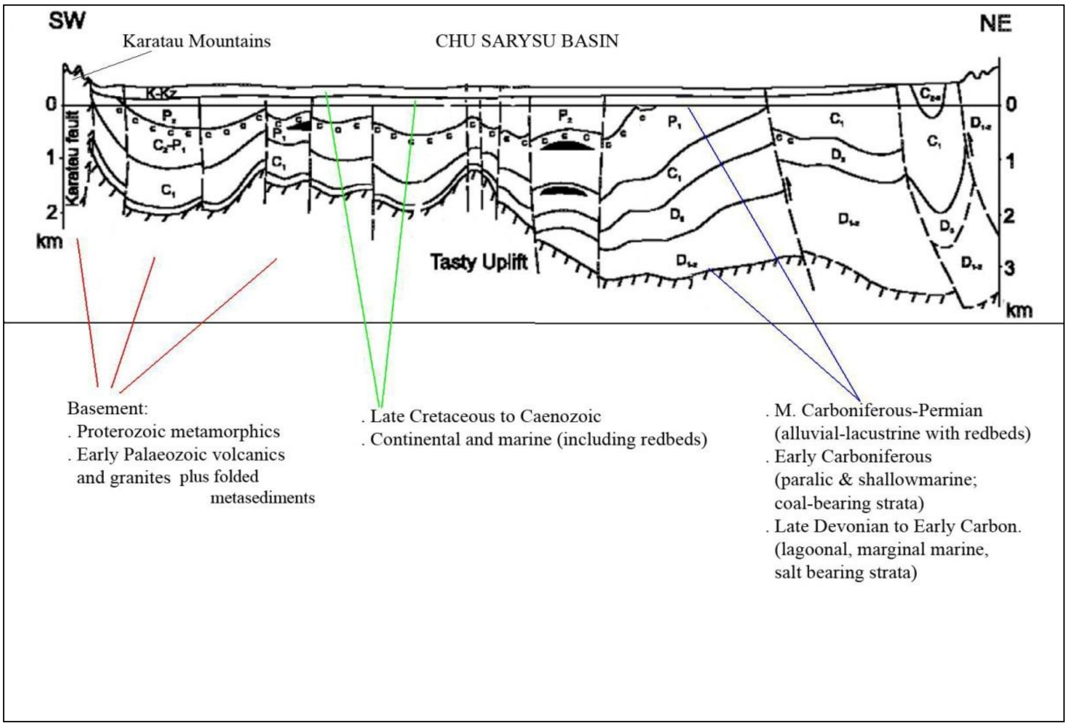


Figure 4-6: Vertical Section of the Chu Sarysu Basin (Figure 4-5 Shows Section Line) of Main Stratigraphic Units

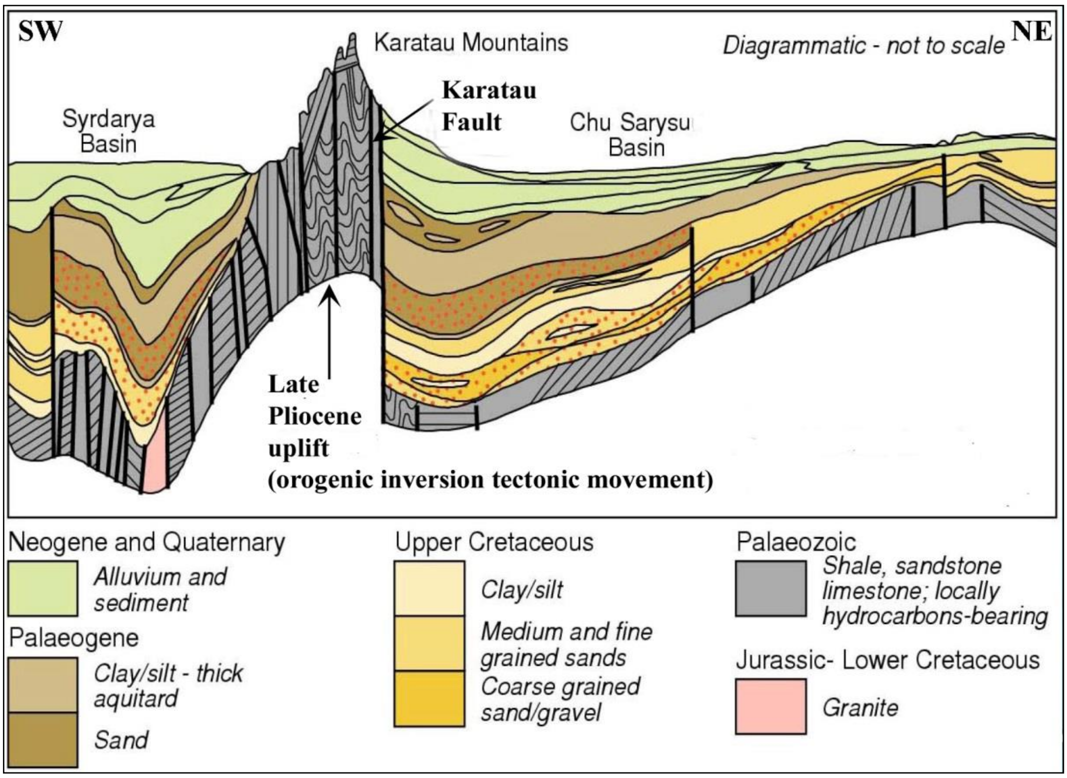


Figure 4-7: Diagrammatic Cross Section showing relationship of Karatau Mountains with adjacent intermontane basins

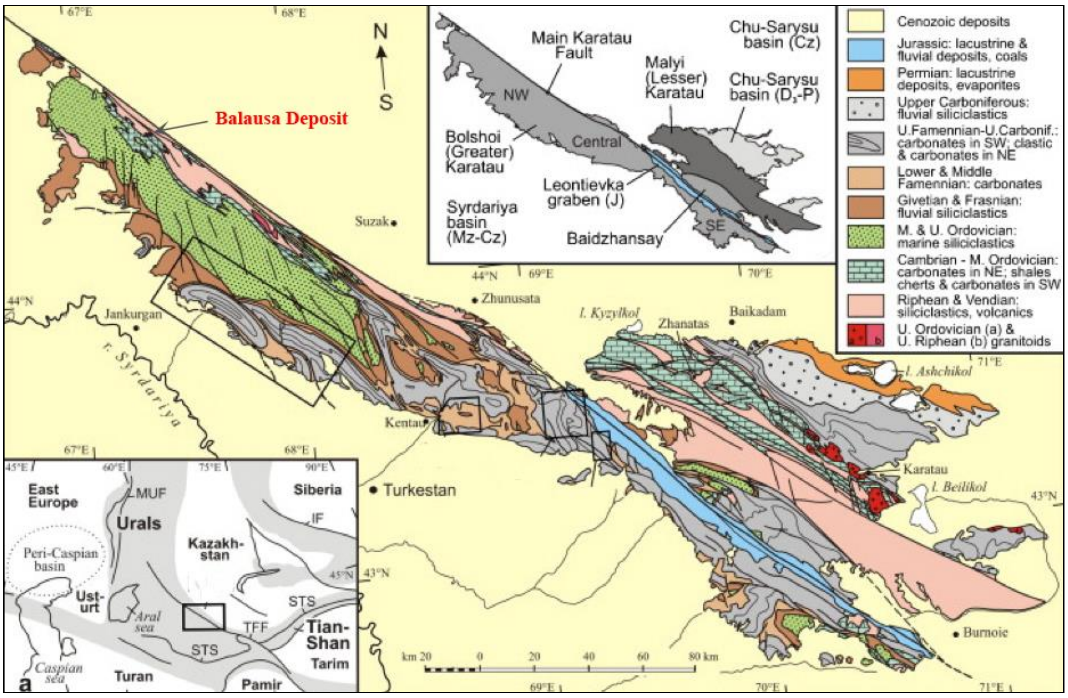


Figure 4-8: Geology of the Karatau Mountains

The geology of the Karatau mountain regions (illustrated in Figure 4-8) clearly shows the dominance of the main Karatau fault, with the juxtaposition of the contrasting Cambrian-Ordovician formations with the Cenozoic deposits to the NE of the fault. The Lesser Karatau region evolved as an isolated carbonate seamount from the Middle Cambrian times (see inset in Figure 4-8 above) and their faunas show remarkable similarity to those of South China until after the Early Ordovician period: there is a Cambrian-Ordovician facies change from carbonates in the Lesser Karatau to dominant shales in the Balausa region of the NE Greater Karatau mountains.

Karatau mountains show polyfolded stratigraphic layers, typical of orogenic belts (Karatau-Naryn) and at the Balasausqandiq deposit, the orebody layers show non-cylindrical elongated synform basin structures; as low order parasitic folds within an anticlinorium.

It is interesting to compare the similarity of the vanadium-rich Cambrian carbonaceous marine sediments of Xiushui County, Jiangxi Province, in south east China, with the Balasausqandiq deposit. Here, both primary and oxide vanadium levels match the Balausa average grades, and, with the mineralisation confined to folded synclines at the margins of regionally extensive graben structures (filled with Cretaceous sediments), also resonate with the Balausa situation. Carbon levels of the primary ores are also similar.

4.4.3 DISTRICT AND DEPOSIT GEOLOGY

The Balasausqandiq deposit is situated within the northwest distal part of the 400 km long Karatau mountain range; with a NW-SE strike. Structurally, the strong mineralogical trend is related to a northwest trending anticlinorium, with numerous subsidiary folding. Middle Cambrian (500 Ma) meta-sediments, which form 11 distinct rhythmic concordant layers, according to lithological signature characteristics, containing the vanadium mineralisation. Specifically, the economically viable vanadium mineralisation is confined to a specific stratiform layer, which typically ranges from 4 m to 14 m in total thickness. The deposit has been exposed along an overall strike length of 10 km and, the total aggregated strike-length of the individual vanadium layers within the Balausa area, is about 40 km.

Structurally the deposit comprises a number of large stratiform vanadium orebodies that have been deformed into extensive asymmetric non-cylindrical folds, with steep sub-parallel axial planes. In surface plan view they exhibit synclinal fold closures and apparent variable wavelength: the orebodies are restricted to these synclines, though an extensive atypical occurrence has been explored at the SW limb of the anticlinorium, where surface exposed vanadium mineralisation can be found as a single unfolded extensive layer and apparently isolated to the normal synclinal folding. The disharmonic fold deformation of these mineralised synclines appears related to the deep-seated Karatau Fault system, which became reactivated during the Late Pliocene (2.5 Ma) when orogenic tectonic inversion uplift generated the Karatau mountain range, with subsequent continual tectonic adjustments and rapid denudation of the mountain range (see Figure 4-9).

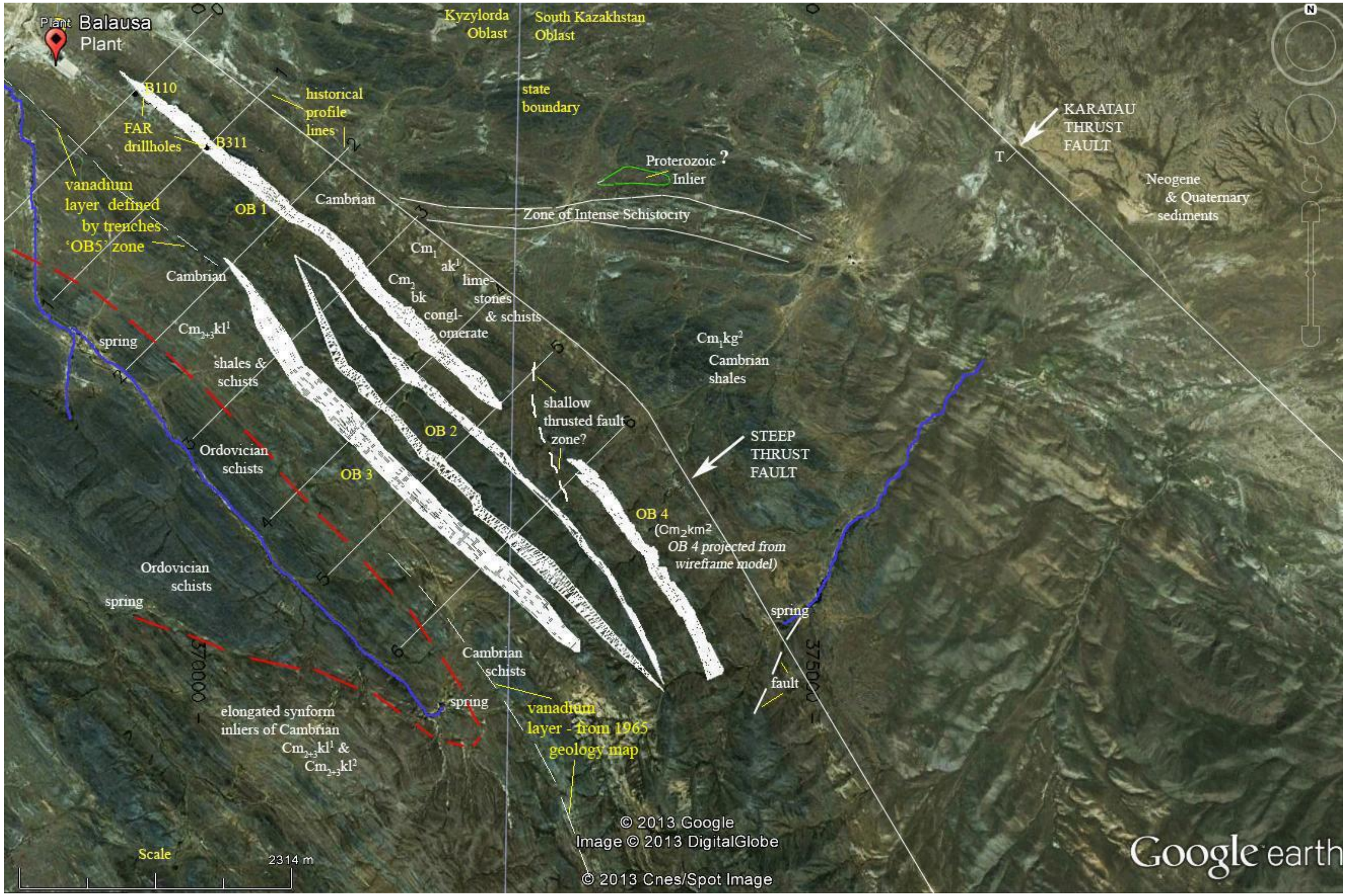


Figure 4-9: Balausa geological features overlaid on Google earth map

4.4.4 STRATIGRAPHY

The stratigraphic record indicates an extremely long geological time span, from Proterozoic to Quaternary formations (600 Ma to 0.01 Ma) and the stratigraphic legend detailed in Table 4-19 summarises the main stratigraphic formations which are associated with the mineralised zones at the Balausa project. This information is based on the evaluation work by the Kazakh Ministry of Geology (B.E. Komarnitski et al) in 1971-1972, and which also takes cognisance of the pivotal exploration work undertaken from 1942 to 1947.

The geological map shown in Figure 4-9, also utilizes the map unit symbols, as defined in the stratigraphic legend detailed in Table 4-19 below.

Table 4-19: Stratigraphic Legend Table

Period	Map Unit Symbol	Stratigraphic Unit Name and Description
Quaternary	Qm	Sediments: Alternating sand, gravel and pebble layers
Ordovician	O_3bs	Besharyk Suite: Greyish-green polymictic sandstones, siltstones, argillaceous slate,
	O_2sn	Suindik Suite: Argillaceous chloritic shales, quartz phyllitic schists,
	O,ks^2	Koskul Suite: The member within the suite is comprised of siliceous schists
	O,ks^1	Koskul Suite: Quartziferous-chloritic-sericitic schists
Mid-Cambrian	$Cm_2+_3kl^2$	Kulantauski Suite: Upper member. Thinly laminated flag-like limestones
	$Cm_2+_3kl^1$	Kulantauski Suite: Lower member. Carbonaceous and argillaceous shales, Carbonaceous-siliceous schists with concretions of phosphorite rocks
	Cm_2km^2	Kurumsak Suite: Vanadium-bearing ore horizon
	Cm_2km^1	Kurumsak Suite: Siliceous schists layer with low-grade vanadium (narrow dark grey dolomite seam at base of this layer “upper dolomites”)
	Cm_2bk	Baikonur Suite: Tillite-like conglomerate
	Cm_1ak^2	Aksumbinski Suite: Upper member. Polymictic sandstones.
	Cm_1ak^1	Aksumbinski Suite: Lower member. Sericitic-chloritic schists. Interbedding of marmorized limestones and carbonate- chloritic shales.
	$Cm_{1,}kg^2$	Karagur Suite: Upper member. Aleuritic (silty) shales
	Cm_1,kg^1	Karagur Suite. Lower member. Siltstones
	Cm_1,rn^2	Rangski Suite: Upper member. Polymictic sandstones, seams of gravelstone. Chlorite and argillaceous shales, carbonaceous-argillaceous shales
	Cm_1,rn^1	Rangski Suite: Lower member. Polymictic conglomerates. Sandstones
Proterozoic	$Ptkn^2$	Kainar Suite: Upper member. Persilicic volcanic rocks and their pyroclastic varieties (e.g. rhyolites, dacites & tuffs)

Period	Map Unit Symbol	Stratigraphic Unit Name and Description
	Ptkn^1	Kainar Suite: Lower member. Basic volcanic rocks and their pyroclastic varieties (e.g. basalts)

4.4.4.1 DESCRIPTION OF THE STRATIGRAPHY

4.4.4.1.1 UPPER PROTEROZOIC

Proterozoic strata within the limits of the Balasausqandiq deposit have been subdivided into two series of formations: lower Kainar Suite (Ptkn1) and upper Kainar Suite (Ptkn2). Due to apparent lack of reliable palaeontological documentation, the stratigraphic sequence still remains a matter of dispute among scientists; therefore the position of the mentioned suite is identified by their structural and tectonic features, which are variously interpreted by different experts.

Lower subsuite in the Kainar strata (Ptkn^1) Rocks in the subsuite include a sequence of effusive mafic formations featuring diabase porphyrite and diabase, less frequent diorite and andesite porphyry, tuffs and tuff breccias of the major or (less frequent) medium age. Sometimes the effusive rocks feature thin (less than 3-10 cm) intercalations and lenses of dark grey limestones, siltstones and siliceous shales (chert). To a different extent, the rocks are schistose and sometimes to the extent of becoming metamorphic shales. Prevailing colour is greyish-green.

The thickness of the strata which form the subsuite reaches 100 to 120 m.

Upper subsuite in the Kainar strata (Ptkn^2) is mainly formed by persilicic effusive rocks and, to a lesser degree by their tuffs. The effusive rocks here include albitophyres, quartz albitophyre, porphyries of light grey and yellowish-greenish-grey. Tuffs and tuff breccias persilicic are represented by crystalline and lithic variations. In certain places the rocks are intensively schistose and turned to various shales.

The thickness of these rocks ranges from 80 to 100 m.

4.4.4.1.2 LOWER PALAEOZOIC - CAMBRIAN AND ORDOVICIAN

The strata of the Lower Palaeozoic contain the exploitable mineralised target zones at Balausa. The characteristics of these strata are also similar for other, but separate, vanadium areas within this part of the Karatau mountain range.

4.4.4.1.3 MID-LOWER CAMBRIAN STRATA (CM$_1$)

According to structural and tectonic features, lithologic composition and, other distinctive signatures, the Mid-Lower Cambrian strata (joined into the Ulutau series) comprise three suites: Rangski (Cm1rn), Karagurski (Cm1kg) and Aksumbinski (Cm1ak).

Strata within the Rangski suites can be subdivided into two subsuites: lower subsuite of basal conglomerates and upper subsuite of coarse-grained sandstones.

Lower Rangski suite (Cm1,rn1) This horizon lies unconformably on the upper Proterozoic strata.

It is represented by basal conglomerates, with the composition of the pebbles reflecting rocks of the upper Proterozoic series. The sizes of the clasts range from a few centimetres up to 0.2 - 0.3 metres, sometimes up to 0.6 - 0.8 metres. Pebbles and boulders are well rounded. Colour of the rocks varies between greenish-grey, light grey and dark grey.

Upper Rangski suite (Cm_1,rn^2) Upwards in the stratigraphic profile, the underlying basal conglomerates develop into intercalations of siltstone shales, sandstones, limestones and dolomites, in tandem to a gradual decrease in grain sizes, and eventual merging with the overlying seams of sandstones. The upper contact of this subsuite can be identified by a bed of black dolomites.

The thickness of the subsuites of basal conglomerates is variable, from a few metres to 300-350 metres. Within the limits of Balasausqandiq deposit, the average thickness of basal conglomerates is 100 to 120 m.

Karagurian Strata

The thickness of this strata ranges from 50 to 55 metres.

Strata of the Karagurian suites can be subdivided into two subsuites: lower section is a siltstone and upper subsuite of violet and green silty shales.

Lower Karagurian (Cm_1,kg^1) strata show conformity in stratification with a gradual transition from the underlying Upper Ranskian, although the borderline between them can be clearly seen in the abrupt changes of colour. This lower subsuite is formed by siltstones with intercalations of silty shales. Rocks have greenish-grey, yellowish-grey frabic, with very infrequent violet colouration. Typically, uneven distributions of pyrite blebs are found. The thickness of the strata ranges from 60 to 100 metres.

Upper Karagurian (Cm_1,kg^2) The subsuite is distinct from other surrounding strata due to its colouration. It is predominated by crimson, lilac and violet argillaceous sericitic-hematite shales, which make this subsuite very unusual. Less frequent are the argillaceous and chloritic, siliceous-chloritic and siliceous-sericitic-chloritic shales brilliant green and emerald-green colour. A distinctive feature of the lilac shales is their thin lamination, which at times has a form of seasonal bands (banded structure), which is determined by variations in the colour of the laminations and less frequently by the grain size characteristics. The thickness of some single layers ranges from microns to one centimetre.

Shales in the top section of the profile gradually become arenaceous with thin interlayers of marmorized dolomitic limestones of cream-coloured, greenish-grey and white in colour. Upwards in the section (profile) the total number of limestone interlayers increases, which determines the gradual transition to the overlying strata.

The thickness of strata in the upper subsuites varies from 100 to 150 m.

Aksumbinski suite crowns the Mid-Lower Cambrian strata in the profile. Lithologically this suite of strata has been subdivided into two sub-suites: lower subsuite of carbonate and chloritic stratum of limestones and upper – arenaceous and shaly subsuite.

Lower Aksumbinski subsuite (Cm1ak1) shows a gradual transition to the associated underlying bedrock. Subsuite formed by evenly alternating interlayers (rhythmic stratification) of dolomitic and marmorized limestones (white and pale) and carbonate-sericitic schists (light-greenish), sometimes grey. Limestones either prevail or occur in different proportions with schists. The thicknesses of certain layers vary from microns up to 5 cm, less frequently - thicker.

Rock sub-suites frequently feature inharmonious folding or wrinkling.

The thickness of strata ranges from 100 to 170 m.

Upper Aksumbinski subsuite (Cm1ak2) conformable with the underlying subsuite, but with a distinct contact. The lower part of the suite is formed by greyish and brownish silty polymictic sandstones, in the upper section – by dark-green and dark grey to black [sic] “carbonaceous” shales with argillaceous-sericitic and argillic-chlorite-sericitic composition. A distinctive feature of the sub-suite is its impregnated pyrite, dolomite and ferrous carbonate.

Layers of yellowish-grey and light-yellow dolomites can be traced all the way through the sub-suite (total thickness of each layer up to four metres).

The thickness of single strata in the suite ranges from 10 to 60 m.

4.4.4.1.4 MID-CAMBRIAN STRATA

The designated mid-Cambrian strata can be subdivided into two suites: the lower – Baikonur suite and overlying - Kurumsakski suite (which defines the vanadium enriched horizons).

Baikonur suite (Cm_2bk) is a typical formation of the Cambrian strata. Strata of the Baikonur suite overlay and overlap with various horizons of eroded surfaces of the Aksumbinski suites with minor angular unconformity with the cross bedding structures.

Overall, the rock mass appears as a tillite-like conglomerate, but no consensus has been established.

The strata of the Baikonur suite in most cases are structureless, sometimes very schistose, greyish-green, dark grey or brownish rocks looking like sandstones or schists with some impregnations of clastic rocks of various sizes, shapes, variable spherical forms and lithologic composition, scattered chaotically in the main rock mass.

Vanadium enriched strata – Kuramsak Suite

The strata of Kurumsakski suite (Cm2km) which form the ore field comprise two constituent units: the lower Siliceous horizon (Cm2km1), which contains subeconomic vanadium grades, and the overlying vanadium horizon (Cm2km2), which is the target for this JORC resource estimation.

Siliceous Horizon (Cm2km1)

The siliceous horizon consists of dark grey rocks, which in the weathered zone are brownish-grey with a platy structure and granular texture. Normally, they are interstratified with thin layers of quartz. At the lower contact, there is a dolomite seam, the so-called “upper dolomites”, lying directly on the tillite rocks of the Baikonur suite: the thickness of this dolomite bed is not uniform, ranging from 1 to 5 m. These barren dolomites serve as a good marker for the siliceous bed and allows for clear tracing of the footwall contact. Where there are no dolomites, the lower border of the orebody can still be traced clearly by the abrupt changes of facies colour in the rock.

The dolomite horizon was intersected by only two drillholes - 11 and 12. Its absence in other boreholes can be attributed to either pinching out of the structure or poor core recovery. In surface trenches, the dolomites were revealed more frequently.

According to Alzhanov (9), the Siliceous Horizon, can be divided into five sub-layers and, from the bottom upwards, can be summarized as follows:

- Sericitic-carbonaceous-quartz schists
- Carbonaceous-siliceous schists
- Lyddites exhibiting as dense greyish-black to black rocks with a conchoidal fracture.
- Sericitic-quartz schist.
- Lyddites and quartziferous rocks.

In detail they a have variegated composition and composed of micro-quartz rocks, lyddites, carbonaceous-quartz schists with their dependant carbonaceous-sericitic and quartz schists and lentils of carbonaceous-siliceous- and argillaceous vanadium-bearing and carbonaceous-quartzitic silty schists, dolomites and radiolarian chert. At the macroscopic level the rocks are greyish-black in colour, sometimes ranging up to dark grey. Lyddites and radiolarian cherts form a massive solid rock. Various schists are clearly schistose, often crenulated. In the lower part of the layer there are intercalations of dark grey dolomites. The rocks of this layer have very minor content of carbon. Due to their massiveness, they predominate on the ridges and crest of the hills.

In general, the thickness of the siliceous horizon within the limits of the Balasausqandiq deposit range from 15 to 23.6 m.

4.4.4.1.5 VANADIUM HORIZON (CM_2KM^2)

The stratum of the vanadium horizon forms a conformable and natural continuation of the underlying siliceous strata.

Within the weathered surface zone of the vanadium-bearing horizon, 14 lithological interlayers have been defined. However, at depth within the primary rocks, it is not possible to distinguish such layering in the drillcore.

1. Rn-ore lower bench – coarsely laminated siliceous schists alternating with carbonaceous and combined argillaceous-carbonaceous shales with the former prevailing. Average thickness 40 cm.
2. Rs-ore schistose bench – alternation of carbonaceous-argillaceous, argillaceous-sericitic and silty schists with intercalation of argillaceous-siliceous, carbonaceous and siliceous schists with characteristic quartzitic interlayers with siliceous and roscoelite ore. Average thickness is 1.0 m.
3. Rpf-ore sub-phosphate bench – regular [parallel] intercalation of black siliceous schists with ore-containing carbonaceous-argillaceous interlayers. The thickness of the siliceous schist interlayers ranges from 3 to 20 cm, argillaceous-carbonaceous shales – from 8 cm to 0.64 cm.
4. Sf-schistose phosphate seam – argillaceous, siliceous-argillaceous silty schists black in colour with abundant phosphate and siliceous-phosphatic concentrations. The thickness of the seam is variable and it ranges from 5 cm to 25 cm.
5. Rf-ore phosphate bench – consists of ore-bearing argillaceous and carbonaceous shales with siliceous schists. Abundance of phosphate concentrations (0.65 m).
6. Gn-argillaceous seam, lower – thin seam of argillaceous, argillaceous-carbonate schists with pyrite and pyrite pseudomorphing marcasite. Average thickness from 6 to 20 cm.
7. Rpd – ore sub-dolomite bench: in its lower part it shows intercalation of siliceous and argillaceous-carbonate shales, and in upper part – intercalation of argillaceous and argillaceous-sericitic shales. Average thickness is 92 cm.
8. D – seam of dolomites – dark-grey medium-grained dolomites. Average thickness is 40 cm.
9. S – shaley seam above dolomites – lower part mainly consists of argillaceous-siliceous, sericitic-siliceous and siliceous schists, with thin laminations.

   Upper part of the bench is formed by argillaceous, chlorite-argillaceous and siliceous-argillaceous schists.
10. Rnd – ore seam above dolomites – thin alternation of argillaceous-carbonaceous and siliceous schists with thicknesses of their interlayers not exceeding 5-7 cm. In the central part of the seam there are obvious thin intercalations of argillaceous schists. Average thickness is 1.98 m.

11. Gs – mid argillaceous seam – the same argillaceous schists as in the seam Gn. Average thickness is 10 cm.
12. Rp – continuous ore-containing layer – regular rhythmical alternation of siliceous and argillaceous-carbonaceous shales, equally distributed, with some minor phosphate and siliceous- phosphate concentrations. Average thickness is 1.3 m.
13. Gv– upper argillaceous seam – the same argillaceous schists as in the interlayers Gn and Gs. Average thickness is 10 cm.
14. Rv – ore upper bench – argillaceous-carbonaceous vanadium-bearing schists, in which some lenses of siliceous schists are unevenly distributed. Average thickness is 82 cm.

The total thickness of these strata in the vanadium-bearing horizon range from 4.4 m to 12.0 m.

Kulantauski Suite – lower member (Cm_2+$_3kl^1$)

The strata of the lower suite of the Kulantauski formation occur directly above the vanadium horizon. According to their composition, chemistry and rock fabric, they are identical to the Siliceous formation which directly underlies the vanadium horizon, and signifies that their depositional environment must also be similar.

4.4.5 STRUCTURAL GEOLOGY OF THE ORE BODIES

The Balasausqandiq deposit comprises a number of large stratiform ore bodies that have been deformed into extensive asymmetric non-cylindrical folds, with steep sub-parallel axial planes. In surface plan view they exhibit synclinal fold closures and apparent variable wavelength: the ore bodies are restricted only to these synclines. This apparent disharmonic fold deformation appears related to the deep-seated Karatau Fault, which became reactivated during the Late Pliocene orogenic uplift (2.5 Ma) with continual tectonic adjustments. In plan, OB2 appears anomalously large in outline, and GMR believe that such an apparent disharmonic fold, located spatially between OB1 and OB3, could be the result of thrust displacements along strike orientated faults, whereby two normal synclines have been “joined” by displacement of the vanadium layer: such evidence can be seen from historical surface mapping and trenching results of the vanadium mineralisation, at the NW synclinal closure area, where there is a confusing split distribution of the ore layers at surface, plus a recent FAR drillhole intersection, where unexpectedly ore was encountered within the central area of this syncline. Note: the continuation of the FAR exploration drilling programme will help resolve this apparent structural anomaly: see Figure 4-16.

The Balasausqandiq deposit was initially split and classified by S.G.Ankinovich into seven exploration blocks. However, exploration blocks 1 and 2 constitute a single (consolidated) closed synclinal structure, cut by a cross-fault approximately in the middle part of the fold. Thus, the previously classified seven exploration blocks have been reclassified into six ore bodies.

Although most of the notable vanadium mineralisation is represented by large folded structures showing synclinal closures and steeply inclined limbs, OB5 appears to be anomalous, being composed of a single limb-type structure but with an extensive strike length of about 8 km.

4.4.5.1 ORE BODY 1

OB1 corresponds to exploratory blocks 1 and 2 described by S.G.Ankinovich. The ore body constitutes a spatially separate asymmetric synclinal fold situated on the north-eastern flank of the ore field. Strike length 4.5 km.

The rocks forming the north-eastern limb dip towards the south-west at approximately 75-78°. The core of the fold is made up of siliceous-carbonaceous rocks of lower-Kulantau subsuite, whilst the limbs are of tillites of the Baikonur suite. The thickness of the vanadium-bearing horizon on the north-eastern limb varies from 5.4 to 8.7 m and up to 11.8 m on the south-western limb. The thickness of the siliceous horizon ranges from 16 to 24 m on the north-eastern limb and from 16 to 18 m on the south-western limb.

4.4.5.2 ORE BODY 2

OB2 corresponds to exploration block 3 described by S.G.Ankinovich. This is the largest in the ore field and occupies its central part. The strike of the syncline fold, which forms OB2 is north-west (azimuth 310º to 315°). The rocks of the north-eastern limb dip to the south-west at 80-85°. The thickness of the ore body varies for the vanadium-bearing horizon from 5 to 12 m, and for the siliceous horizon from 16.5 to 23 m. Strike length 5.5 km.

The south-western limb of OB2 has mainly been explored at outcrop. Only one borehole was drilled to the depth. The south-western limb is flatter, dipping at 55 to 75° to the north-east.

Thus, the fold, which forms two ore bodies, has an asymmetric structure. The south-western flank is broken up by faults, sometimes with complete removal of the vanadium-bearing horizon, the thickness of which is up to 12.1 m. Minimal thickness of siliceous strata reaches 35 m while the minimal thickness of siliceous rocks is unknown.

4.4.5.3 ORE BODY 3

OB3 (exploration block 4 of S.G.Ankinovich) forms a narrow synclinal fold oriented parallel to OB2. Strike length 4.8 km.

The synclinal fold that forms OB3 is comprised of siliceous and carbonaceous rocks of the lower Kulantauski subsuites, and along the fold limbs it includes tillites of the Baikonur suite and (less frequently) by formations of the siliceous horizon.

The north-eastern limb of the fold dips at angles of 80 to 85° to the south-west. However, some areas demonstrate overturned bedding, and the beds in this case dip towards the north-east at the same angle. The rocks of the south-western flank also steeply dip towards the north-east.

The thickness of the vanadium-bearing strata varies from 3.2 to 12.0 m on the north-eastern limb and from 3.0 to 7.8 m on the south-western limb. The strata of the siliceous horizon have a thickness between 14 and 26.6 m on the north-eastern limb. While on the south-western limb the thickness of the siliceous rocks has not been explored.

4.4.5.4 ORE BODY 4

OB4, exploration Block 5 according to S. G Ankinovich, was not explored in great detail during 1971-72. Previous prospecting works only surveyed the oxidised ores by a series of trenches spaced every 100 m and, as yet, no drilling of the primary horizon has been undertaken. Strike length 2.6 km.

Primary ores were surveyed only through one sampling pit (No. 4). OB4 is a natural strike continuation of OB1, separated by tectonic thrusting.

According to S.G.Ankinovich, both the north-eastern and the south-western limbs of this fold are steeply dipping, and its cross-strike profile is close to symmetrical. Both limbs dip from 50 to 75°. The thickness of the vanadium-bearing horizon on the north-eastern limb ranges from 6.45 to 17.1 m, and on the south-western limb from 5.5 to 16.7 m. The rocks of the siliceous horizon were not surveyed.

4.4.5.5 ORE BODY 5

Block 6 (OB5) identified by S.G.Ankinovich was not surveyed during 1971-72. Exploration consisted of using a low-density grid of trenches within the oxidized ores. This ore body lies on the south-western limb of the Balasausqandiq anticline. The thickness of the vanadium-bearing horizon varies from 5.56 to 7.94 m. The dip of the strata is typically steep, towards the south-west, at angles ranging from 60 to 80° striking 310-325°. The siliceous horizon here was not explored. Strike length 8 km.

Figure 4-10: Lower Contact of OB1 (NE Limb OB1) – Tape Extended 1m

Figure 4-10 illustrates an exposure that was seen in a historical trench and clearly shows the brown oxidation of the vanadium layer and the distinct contact with the lower conformable graphitic looking siliceous layer, which contain low grade mineralisation. The upper contact within the same trench is seen in Figure 4-11. This contact is less distinct with the overlying schists to the right.

Figure 4-11: Upper Contact OB1 (NE Limb) Looking SE

4.4.6 ORE GENESIS

The geological history of this deposit in relation to plate tectonics and lithology/geochemistry characteristics, strongly imply that the primary ore layers and, other juxtaposed sediment layers, were deposited in graben structures that were in a marine environment of relatively shallow depth: at Balausa there are indications of conglomerates, deep seated faulting and rifting in common with the South China vanadium deposits, within the Gondwana craton. Such grabens were spatially centred at a reactivated suture zone which had pre-existed in the Precambrian supercontinent of Rodinia, and this developed into an open but relatively shallow marine basin within the supercontinent of Gondwana from Late Proterozoic.

Historically, there were a number of hypotheses on the genesis of the vanadium mineralisation:

- Vanadium is believed to have penetrated into the basin of sedimentation, together with other volcanic products (tephra). This viewpoint is based on the paragenetic relation between carbonaceous-siliceous schists, volcanic formations and higher content of vanadium pentoxide in the effusive rocks of Precambrian and Cambrian strata. However, it was noted that this direct relationship between vanadium concentrations and the sub-synchronous volcanism was not well correlated, as the extent of volcanism was very restricted, yet the vanadium was widely spread in the vanadium-bearing Cambrian strata of Kazakhstan, Tien-Shan and other areas.
- S.G. Ankinovich, E.A. Ankinovich and M. Adyshev attribute the penetration of vanadium into mid-Cambrian ponds, to deep chemical erosion of their adjacent land areas, where the effusive rocks of Precambrian age feature higher vanadium content. As a result of this, primary vanadium accumulations took place in the weathering crust, and thereafter as a result of erosion, moved to the accumulation basin in solution (as dissolved substance) with their subsequent precipitation.
- B.T. Tyurin assumed it possible that chemical sedimentation occurred from regular (molecular) solutions, however, S.G. Ankinovich believes that vanadium precipitation occurred as a result of biochemical processes through digestion by plankton. On death of the plankton, vanadium accumulation built up within sea floor sediments.

Fundamentally however, a primary sedimentary origin has now been accepted for the vanadium mineralisation at the Balasausqandiq deposit, and this has been established for other vanadium deposits occurring in similar mid-Cambrian strata of Kazakhstan. The ore horizon being clearly stratified and can be traced over vast distances.

From a genesis perspective, there are two valid, but probably overlapping scenarios: a biogenic and an abiogenic geochemical process, which were involved in the concentration of the vanadium mineralisation.

4.4.6.1 BIOGENIC CONTRIBUTION

Marine filter feeders are known to concentrate vanadium several (10^6) million times seawater levels. A positive trend between V (IV) and large size phytoplankton biomass suggests that levels of reduced V oxidation states may be influenced by biological activity.

Vanadium is an essential element for many marine phytoplankton species and other organisms. Many enzymes contain V and it may be involved in many metabolic processes such as chlorophyll synthesis, cell division, phosphate uptake kinetics, sulfoxidation in cell motility and photosynthesis. While V (V) is the thermodynamically stable form in oxygenated seawater, V (IV) commonly exists in intra-cellular media V (IV) ions, such as VO_2+, and is generally internalized into the cytoplasm through passive diffusion. Once inside the cell, V (IV) is actively involved in phytoplankton metabolism. Vanadium is relatively abundant in open ocean waters (34–45 nano-metres in oxygenated seawater) with a relatively conservative distribution. Thermodynamically, V (V) is stable in oxidized marine environments, while V (IV) is stable in moderately reducing environments. In general, the solubility of V decreases with decreasing valence, and therefore V (IV) increases proportionally with decreasing total dissolved V pool as the environment becomes progressively more reducing (e.g., Sadiq, 1988 (10)). For example, compared to the Atlantic Ocean, dissolved V concentrations are depleted (by ~60 %) in reducing deep waters of the Black Sea and the Cariaco Basin suggesting that reducing sediments may serve as a major sink for this element (e.g. Emerson and Huested, 1991; Nameroff et al., 2002 (11)).

However, the role of molybdenum as a trace-nutrient is crucial for organisms performing nitrogen fixation, the capability of reducing N_2 to biologically useful ammonia is limited to some bacteria and archea - this observation can be confirmed at OB1, where Mo has a higher correlation with carbon than vanadium with carbon at Balausa.

As for the underlying formation layer of siliceous rocks, which contain low grade vanadium, the precipitation of silicon dioxide apparently took place through chemical processes and partly through a biogenic process, which is testified by numerous remnants of siliceous sponge skeletons in the area of the Lesser Karatau mountains.

4.4.6.2 ABIOGENIC CONTRIBUTION

High concentrations of vanadium (>0.1 % V_2O_5) in carbonaceous shales can be explained by abiogenic geochemical changes that occur during the time of deposition and this environment exhibits distinct signatures:

- High correlation between vanadium and organic carbon
- Close source of rocks enriched in phosphate
- High content of organically based sulphur
- Relatively slow rate of deposition in a marine basin.

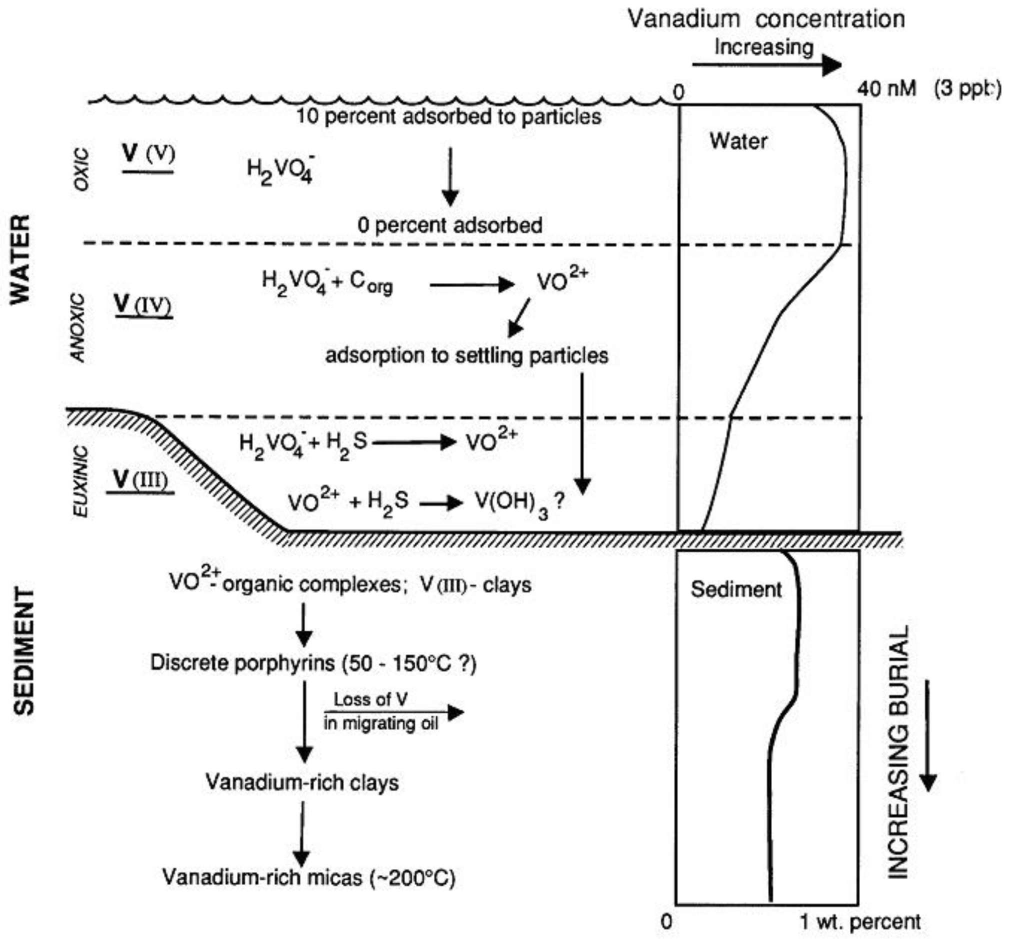
Reduction/oxidation, adsorption and complexation control the mobility and accumulation of vanadium. Vanadate mineral species V (V) dominates the surface aerated surface waters, adsorbs strongly to some mineral surfaces, especially ferric oxides, and limits the concentration of dissolved vanadium at moderate pH levels. Adsorption to biogenic particles promotes depletion in shallow-marine waters but contributes V to the sediments, though in oxic waters these particles will tend to redissolve during re-mineralisation of these particles. In deeper oxic or anoxic waters detrital organic matter and associated vanadium is likely to be preserved. Adsorbed vanadium is likely to increase in anoxic waters because of reduction of vanadate V (V) to vanadyl V (IV) - having a greater tendency to adsorb: see Figure 4-12. Formation of stable organic complexes by the vanadyl ion, relative to many transition metals, explains why vanadium and organic carbon is highly correlated in the primary layer.

The common occurrence of vanadyl-organic species in petroleum attests to the stability of such complexes.

Vanadium-rich carbonaceous rocks are commonly interbedded with transitional rocks enriched in phosphate. The separation of vanadium-rich carbonaceous and phosphatic rocks into distinct sub-units, is due to the contrasting composition of the depositional waters. Phosphate accumulations form when organic matter is partly destroyed under suboxic conditions, whereas carbonaceous matter and associated vanadium are mostly retained by sediments that are in contact with anoxic or euxinic waters. The occurrence of such sub-units can be seen in surface exposures at Balausa.

The presence of organically bound sulphur in vanadium-rich carbonaceous rocks suggests that H_2S was abundant during early diagenesis. H_2S not consumed in pyrite formation reacted with organic matter or diffused upwards, resulting in possible euxinic bottom waters. Under natural conditions, only H_2S is known to reduce V (IV) to V (III), an oxidation state which favours incorporation of vanadium into clay minerals and such enrichment appears to be a characteristic of this deposit. Vanadium enrichment of clays, will tend to be retained during metamorphism, to produce vanadium-rich graphitic schist, though graphite at Balausa has been reported to be <1 % of the total carbon, so the level of metamorphism must be relatively lower than that required to form a general conversion to graphitic schists.

Stratified marine basin sediments require open circulation, in order to provide sufficient vanadium for carbonaceous sediments to have high concentrations of vanadium greater than 0.1 % of V_2O_5.



Geochemical processes that affect the abundance and speciation of dissolved vanadium and the distribution of vanadium in sediments during diagenesis. Vanadium concentration profiles are estimated based on the geochemical processes. Concentrations in water are nanomolar; concentrations in sediment are weight percent on a dry-weight basis.

Figure 4-12: Vanadium Geochemical Enrichment Processes in Marine Basin of Deposition

As an addition to ore genesis, observations and interpretations on a Queensland vanadium deposit provide interesting additional information, which support some of the findings at Balausa (12):

Although V and Mo were concentrated by marine organisms and deposited with organic matter which is a significant component of the Toolebuc Formation, these elements were firstly mobilised into new phases by diagenesis and then again by oxidation during weathering. Diagenesis caused the majority of V (~80 %) to be adsorbed onto clay minerals, although the specific hosts are unresolved. Mo was incorporated into the sulphide component of the fresh rock. Oxidation due to weathering removed TOC as a rock component, with residual enrichment of V and Mo. The majority of the V (~60 %) and Mo (~85 %) contents in fresh rock are remobilised during oxidation/weathering and redeposited in oxide/oxyhydroxide phases.

4.4.7 MINERALISATION

Extensive mineralogical investigation of the vanadium enriched strata was conducted under Ankinovitch in 1943 to 1947. However, most of this research was confined to the depth-limited exposures found at surface, including shallow underground exposures, within the oxide and transition zones. Here, such weathering of the primary ores had changed the chemical and mineralogical character of the vanadium and its associated by-products, so only relatively scant observation of the true nature of the underlying primary zone was achievable; but it was acknowledged that these primary ores comprised an estimated 95 % of the total resource potential and this is confirmed by GMR from the OB1 resource modelling. It was not until the 1970s (Komarnitski) that core was successfully recovered from the primary ores at depth, and then a better understanding of the mineralogy was possible.

The primary ore stratum (Cm_2km^2) normally starts at a depth of 10 m from the topographic surface, but may outcrop when the carboniferous schists/shales are highly silicified or can be deeply oxidized to a depth of 25 m+ where the carbonaceous shales are more argillaceous and only slightly siliceous.

The stratiform vanadium layer is the economic target for exploitation, though historically, the underlying and thicker conformable siliceous horizon (Cm_2km^1 stratum), has also been the target for evaluation, but the grade levels in this formation are sub-economic.

4.4.7.1 SURFACE OXIDATION OF PRIMARY VANADIUM HORIZON

Oxidised vanadium ores, which account for only 5 % of the total ore tonnage exhibit significant changes in chemical characteristics. The content of silicon dioxide is typically 10 % higher than in primary ore, whilst the carbon content is 80 % lower but the vanadium pentoxide content is about 55 % higher.

Low levels of uranium have been identified with the deposit strata. Uranium mineralisation is unevenly distributed and normally associated with the upper part of the oxidation zone in the vanadium-bearing strata due to its migration from the deeper parts and subsequent adsorption by iron hydroxides. The uranium content is considered too low for commercial mining, but will be extracted as a by-product. It is understood that a small quantity of vanadium (10 %) is associated with uranium bearing minerals.

Other typical minerals in the oxide zone include halloysite, metahalloysite, allophone-alumophosphates, ferroalumophosphates, alunite, gypsum, jarosite hydrohematite and limonite.

In general terms, loss of organic matter by oxidative weathering takes place across a reaction "front" where organic carbon content decreases sharply toward the land surface along with organic nitrogen and sulphur, but not for organic phosphorus which remains relatively constant over the same depth range. Accompanying the decrease in organic matter is an increase in porosity. The oxygen content of the organic matter increases sharply toward the land surface across a similar carbon oxidation front.

Pyrite in the primary zone decreases toward the surface more extensively than organic matter and at intermediate depths it is essentially absent in the presence of high levels of organic matter.

Weathering of organic carbon in shales is caused by its reaction with gaseous O_2 and O_2 dissolved in groundwater. Once a weathering profile is developed the downward migrating of O_2 reacts first with modern soil organic matter and subsequently with low concentrations of remaining pyrite and ancient organic matter before reaching the "front". Pyrite reacts faster with O_2 than does organic matter (for a given local concentration of oxygen) as evidenced by reduced pyrite concentrations accompanying high organic matter concentrations at the front. Upon further downward migration, lower levels of O_2 react with higher concentrations of pyrite and organic matter.

Table 4-20: Chemical Composition of the Oxide Ore Horizon Ore Body 1

Element Assayed	Oxide Ore		
	Max (%)	Min (%)	Average (%)
Ignition Losses	8.3	2.88	5.57
SiO_2	82.38	62.6	72.8
Al_2O_3	7.09	2.1	5.07
CaO	6	2.13	3.38
MgO	0.65	0.43	0.57
BaO	*	*	*
Fe_2O_3	6.71	4.57	5.51
C	7.3	0.48	2.69
P	0.38	0.23	0.31
Fe	4.7	3.2	3.85
Mn	0.29	0.16	0.19
P_2O_5	0.89	0.53	0.72
MnO	0.67	0.21	0.31
V_2O_5	1.27	0.76	1.1

Based on analysis of 239 core samples from 13 drill intersections, the primary vanadium ores possesses high organic carbon content (see Table 4-20 and Table 4-21) which has a positive impact on using these ores in metallurgical applications along with a relatively high content of silicon dioxide and low content of phosphorous pentoxide and aluminium oxide. The titanium oxide content is also very low with higher titanium values directly correlated to the amount of pyritisation present, which itself is associated with increased content of aluminium oxide.

Overall this will contribute to the level of iron in the ferro-alloys. The level in the vanadium ore being directly correlated with the amount of argillaceous material present as intercalations within the bed.

Table 4-21: Typical Composition of Vanadium Ore

Component	Range [%]	
SiO_2	59.2	74.8
V_2O_5	0.53	0.83
C	9.96	13.61
P_2O_5	0.41	0.75
Al_2O_3	3.01	6.07
Fe_2O_3	2.56	5.01
CaO	0.60	4.11
MgO-	0.34	1.93

4.4.7.2 MINERALISATION OF VANADIUM HORIZON

In explaining the occurrence of different vanadium minerals, within both the primary ore and the relatively minor secondary oxide ore zone, it would be informative to see this in the context of the geochemical changes that are associated with the type of carbonaceous black shales found at Balausa. Such changes can be explained by abiogenic geochemical changes that occur during the time of deposition and this environment exhibits distinct signatures: refer to Section 4.4.6.2.

Figure 4-13: Drillhole Core from B213 at 121m

Figure 4-13 shows characteristic primary ore formation rock, containing 0.84 % V_2O_5 and 14.73 % carbon: schisty carbonaceous siliceous vanadium shale showing a mesh of quartziferous veinlets.

4.4.7.2.1 VANADIUM

At Balausa, vanadium has been found in 117 minerals, but the bulk of the vanadium is confined to a few dominate minerals, and, as vanadium is found in a number of oxidation states (II, III, IV, V) the vanadium within the various minerals are characterised by specific oxidation states. As a general trend, vanadium in sulphides show low oxidation states (II and III), as found in the primary carbonaceous zone, intermediate oxidation states (III and IV) are seen in micas within the zones of weathering and primary, and the highest oxidation state (V) are seen in vanadate compounds (V_2O_5) within the oxide surface zones. This scenario also mirrors the geochemical changes that occur during enrichment processes within a marine basin (Figure 4-12 above) and also reflects the degree of difficulty at which vanadium can be extracted from the ore-bearing minerals during ore processing: it is more intricate to recover vanadium from the sulphides, easier from micas and easiest from vanadates.

4.4.7.2.2 MAIN VANADIUM MINERALS IN THE PRIMARY ZONE:

- **Patronite**: is an impure vanadium sulphide, and the V_2S_3 form is the most stable – it is the main vanadium contributor at an estimated 50 % of the ore resource.
- **Sulvanite**: is a primary ore sulphide of copper and vanadium, CU_3VS_4, and contributes 5 % of the vanadium resource.
- **Phengite**: it is micaceous vanadium mineral that is similar to muscovite, high in silica, either as a primary mineral or as a detrital mineral which can survive low grade metamorphism. It has been spilt into two types, according to the vanadium oxidation state:
- **Phengite**: where the oxidation state is V (III), containing BaV_2O_4 and contributes a massive 35 % of the vanadium resource. The higher amount of vanadium in this phengite type is because vanadium as V (III) is much more easily incorporated into clay minerals (from which phengite is derived), during deposition and diagenesis, than V (IV).
- **Phengite**: where the oxidation state is V (IV), but its vanadium contribution is relatively minor.

4.4.7.2.3 VANADIUM SECONDARY MINERALS FORMED BY OXIDATION PROCESSES

These remaining minerals contribute about only 10 % of the total vanadium resource. The list is limited, but contains the most important vanadium containing minerals within the oxide zone:

- **Roscoelite**: a vanadium bearing mica found where there is intense weathering of the primary vanadium ores and more typical and dominant in this type of arid climate.

- **Carnotite**: is a uranium-vanadium oxide mineral, $K_2(UO_2)_2(V_2O_8)\cdot H_2O$, typically formed through the action of meteoric waters on primary uranium and vanadium minerals. It is also similar to tyuyamunite, $Ca_2(UO_2)_2(V_2O_8)\cdot H_2O$, and these two minerals are found together within the zone of oxidation.
- **Nolanite**: is a vandiferous iron oxide formed under oxidizing conditions, and contains both V (IV) and V (V) oxidation states.
- **Chernikovite**: is a vanadium bearing mica and may have variable composition.
- **Muscovite**: although a widely spread rock-forming mineral, this mica can contain vanadium and survive within the zone of oxidation.
- **Montrozeite**: VO(OH) – note that when the water decreases it can alter to VO_2. It is normally found in the more relatively unoxidized parts of the weathered zone and considered to be a primary mineral, though GMR believe that its existence within the oxide zone at Balausa could be related to the presence of reported weathered anthraxolite bitumen (occurring in veins or disseminated), as with similar vanadium deposits in China, where anthraxolite has been reported to contain the mineral montrozeite. This explains why this primary mineral remains stable, when complexed with bitumen, in the zone of oxidation.
- **Wakefieldite**: is an uncommon rare earth metal vanadate mineral, tending to have lighter REMs. It is derived by the natural leaching of primary vanadium-yttrium minerals, such as hollandite.

V- porphrins: ($C_{32}H_{38}N_4V$) these are vandiferous organic compounds, probably derived from primitive plant forms at the time of deposition. In the oxide zone these are most likely associated with bitumens. Their contribution to the vanadium resource is estimated at 1 %.

4.4.7.2.4 CARBON

Carbon, as a by-product, is a major economic contributor to the resource. The carbonaceous primary shales and schists, are rich in carbon, containing an estimated average 14 %, and it is relatively straightforward to extract, with very high recoveries from the ore processing. The form of the elemental carbon can range from graphite, to partial graphite and amorphous carbon. It has been reported that the graphitic form of the carbon is very minor at <1 %, and this is indicative of a relatively low level of metamorphism to which the vanadium layer has been subjected, vis-à-vis graphitic schists where metamorphism is much more pronounced. Note: the open pit area exposed the nose of the OB1 syncline, and because here the vanadium horizon has been atypically highly compressed, contorted and silicified, the carbon is more likely to be converted to graphite, vis-à-vis limbs of the fold. Although the carbon can be considered to be in an elemental form, it has obviously been derived from the organic carbon that existed during the time of deposition. In the zone of oxidation, it is easily leached from the rock strata, whereby the rock fabric is converted to a light brown colour.

Note: at the time of writing this report, there is very little information on the molybdenum and uranium containing minerals.

4.5 DRILLING, SAMPLING AND ASSAYING

4.5.1 FORMER SOVIET-ERA

4.5.1.1 SAMPLING OF THE OXIDE ZONE 1942 – 1947

Although extensive sampling campaigns were undertaken in the 1940s (Ankinovich) this was broadly restricted to the oxidized surface layers, by mostly cross-strike trenching and minor underground exposures, via shafts and galleries to a maximum depth of 30 m. Underground sampling intersected both oxides and transition ores but only very minor primary unoxidized layers were mined. Based on the analytical results from the 1940s exploration sampling, it was erroneously concluded that the V_2O_5 grades were similar for both the oxidized and deeper primary zones, due to scant knowledge on the vanadium grade distributions at depth, beyond the influence of surface oxidation processes. This sampling bias was confirmed from the core drilling campaign, by Komarnitski (1970 to 1971), which sampled primary unoxidized ores at depth: results clearly showed that grades were lower in the primary ores vis-à-vis oxide zone. However, the total reported organic carbon values, by Komarnitski, appeared far too low for the primary ore zone, due to the analytical method employed: note that FAR’s total carbon analyses from the primary zone averaged 14 %, as based on Australian laboratory results in 2011.

The extensive historical sampling of the oxide layer, via trenches, pits, shafts, crosscuts and adits, proved extremely valuable in determining the surface continuity and geometry of the ore bodies, and this defined surface expression of the vanadium layers was used in helping to generate 3D models using Datamine.

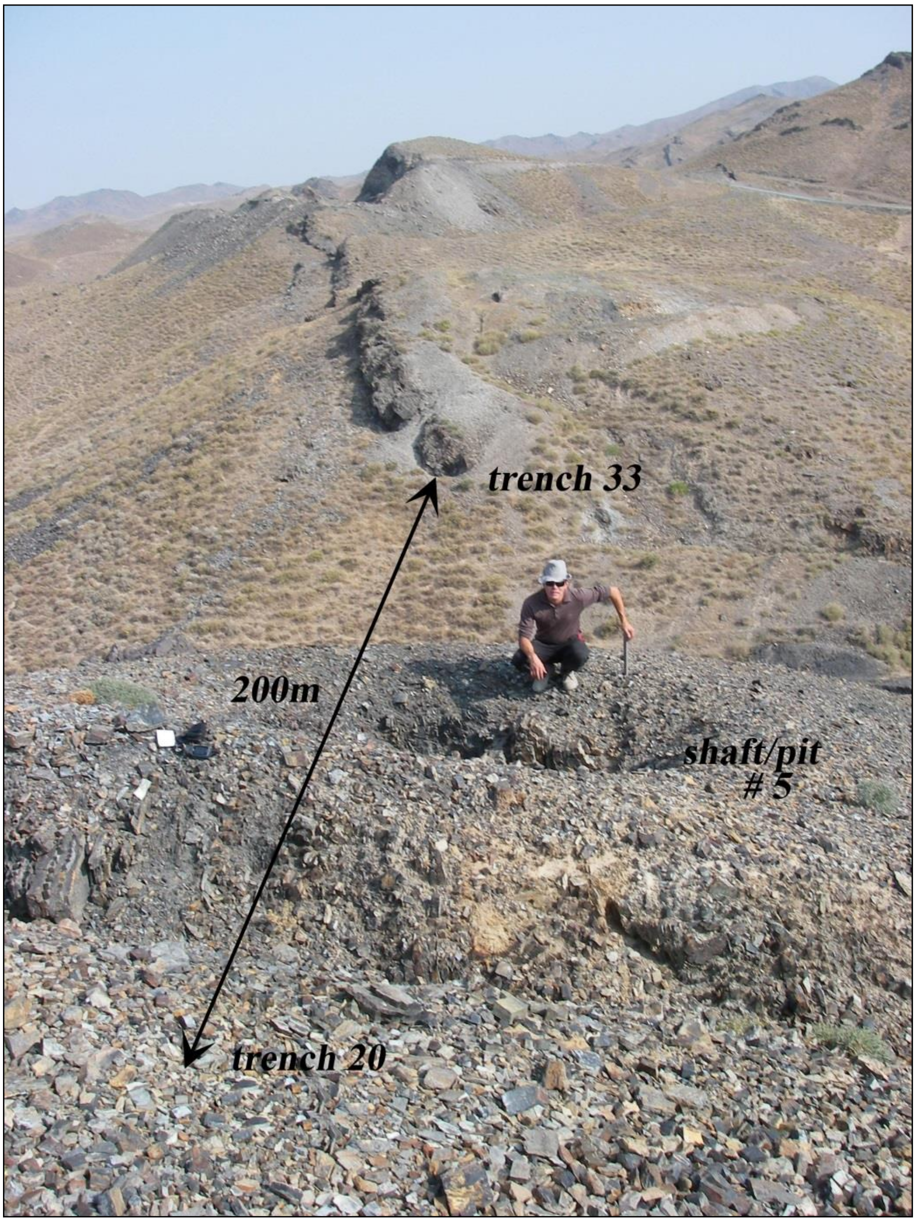


Figure 4-14: Photo at profile 1 looking SE along NE limb of Orebody 1

shows the NE limb as a conspicuous and contiguous surface expression, where the stratigraphically and conformable underlying sub-vertical siliceous horizon to the vanadium stratum, is more highly silicified than that of the lower elevated SW limb to the right**.** It also clearly demonstrates the continuity of the ore horizon along strike and emphasised by the historical sampling sites at trenches and pits. shows the same view from the 3D Datamine model.

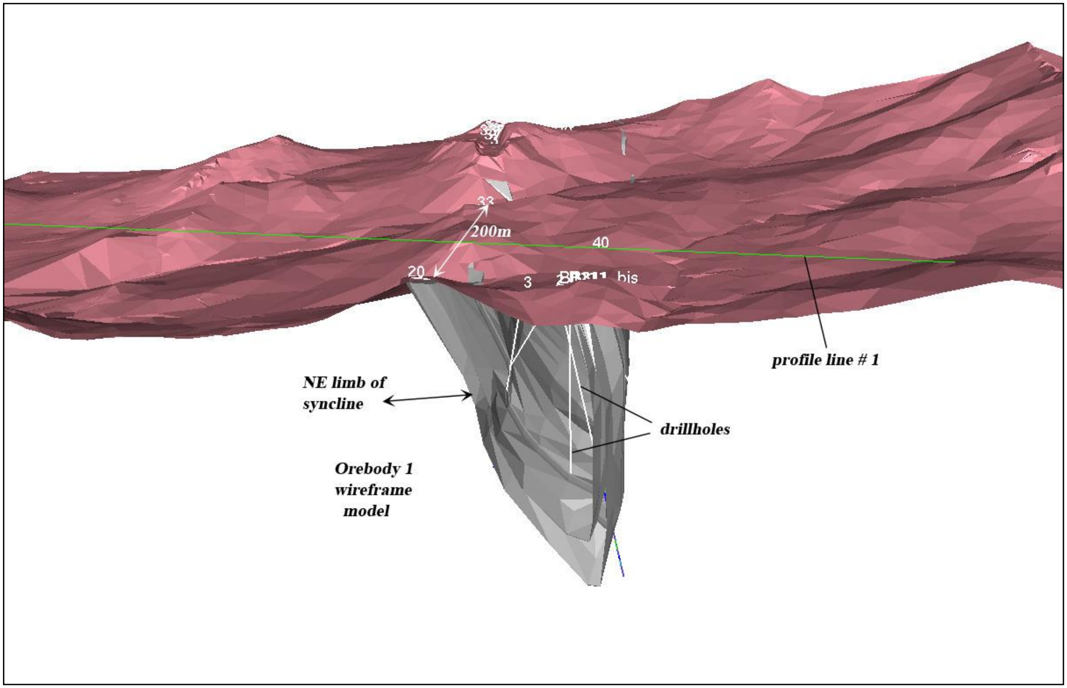


Figure 4-15: 3D DM Model – Same View as with Trench 20 and 33 at Surface

The comprehensive trench sampling results, from this period, were used to estimate the global resource for the surface oxide zone at OB1 (Inferred JORC). However, very little reliable information was available for the primary mineralisation at depth, from this period, and no suitable assay results were accessible for resource estimations.

4.5.1.2 1943-1944 FIRST CORE DRILLING ATTEMPTS

The first recorded core drilling was conducted during 1943 and 1944. Two boreholes were drilled for a total length of 158 m (true vertical depth). Core recovery in the ore zone was very low, being in the order of 15 to 20 %, which clearly is unrepresentative of the ore horizon. No more core drilling was attempted until the 1970s.

4.5.1.3 PHASE 2 EXPLORATION DRILLING (KOMARNITSKI 1972-1973)

During this period 15 coring boreholes were drilled totalling 1,744.0 m, together with an additional 22 new trenches and clearance, deepening (by 0.2-0.5 m) and re-sampling of 17 old trenches (total volume 1411.5 m3), from which 1,808 samples were taken (see Table 4-22).

Table 4-22: Summary of Works 1972 to 1973

Operation	Unit	Scope of Work	
		Designed	Actual
Core Drilling	m	1,630	1,744.0
Trenches	m^3	1,370	1,411.5

Borehole Nos. 2 to 8, No. 9 - down to 74.7 m and borehole No. 10 - down to 31.8 m were drilled with a double ejector pellet borer (borehole size 110 mm), known as “Shot drilling”.

Borehole Nos. 1, 11, 12, 13, 14 and 1-bis, together with extension to borehole Nos. 9 and 10 were drilled with a single-barrel core drilling machine with diamond drill bits (bore size 76 mm). Diamond drilling however was not “successful” because low levels of core recovery were recorded. However, the 1990-1991 drilling campaign (see Section 4.5.1.4 below) confirmed the soundness of the V_2O_5 grades and the FAR drilling results also supported the credibility of the sample analyses.

Borehole Nos. 1 and 7 drilled and sampled in 1971 were later (in 1972) re-drilled due to poor core recovery. Borehole No.1 was also twin drilled and named “1-bis”. This borehole produced better core recovery than borehole No. 1. Although borehole No. 7 was re-drilled, it produced no positive results as the core recovery was much lower than that from the initial hole.

Particular emphasis was placed on prospecting the north eastern limb of OB1 (seven boreholes). Here one borehole was positioned on each of the five prospecting gridlines (0-IV). The south-western limb of OB1 was intersected by two boreholes (Nos. 2 and 14) located on survey grid lines I and IV.

The north-eastern limb of OB2 was intersected by three boreholes (Nos. 8, 9, 12) located on survey grid lines IV, V and VII only. OB2 was not surveyed at depth on survey grid lines II, III, VI and VIII. The south-western limb of OB2 was intersected at depth by only two boreholes (Nos. 4 and 10) both being located on survey line IV.

The north-eastern limb of OB3 was intersected at depth by two boreholes (Nos. 6 and 13) located on survey grid lines IV and VI. The south-western limb of this structure was not explored at depth.

During the same period, gamma-logging was also conducted on all boreholes and gamma measurements were taken in old workings and trenches (see summary Table 4-23 below). All of the trenches were covered by gamma-logs. In boreholes, some of the holes were only partially surveyed. Calliper log measurements and neutron-activity logging to reveal vanadium-52 and aluminium-28 isotopes was also undertaken alongside gamma logging. In the course of logging operations, it was discovered that the presence of radium and aluminium interfered with the vanadium identification.

Table 4-23: Summary of Gamma Logging 1972 to 1973

Planned Coverage	Operation	Unit	Scope of work		%
			Designed	Actual	
95 %	Gamma Logging	m	1,548.5	1,474.4	88
100 %	Gamma Measurement	Points	1,370	1,555	110

Samples were taken from all of the vanadium-bearing and siliceous horizons. In addition, two to three samples were taken from the hangingwall rocks (strata of the lower Kulan subsuite) in view of their possible use as siliceous ores. The footwall rocks were typically tested by two to four samples, and less frequently by more than four samples.

For the siliceous horizon the minimum length sample interval was 1.0 m, and typically up to 3.0 m.

Sampling of the vanadium-bearing horizon, which has greater lithological variation and frequent interlayers, was completed over core intervals not in excess of 1.5 m. The minimum sample interval was 0.4 m, whilst the average length was 0.5-0.8 m.

4.5.1.4 DRILLING FEASIBILITY CAMPAGN 1990-1991

In 2010, FAR discovered that a major drilling programme, comprising of 95 drillholes, was completed during the period from 1990 to 1991 by the Tashkent Research and Development Institute. Unfortunately, details of this 1992 drilling programme, plus specific core sample analyses and the generated feasibility report, are not available. However, a summary report of this work, dated 1992, is available and states that there were 95 diamond drillholes totalling 6,400 linear metres, plus 20 surface trenches. The drilling was designed to target the primary mineralisation at depth: this included 38 drillholes for OB1, 22 drillholes for OB3 and 13 drillholes for OB4. The results basically confirmed the vanadium pentoxide grade trends for the primary zone, as reported by Komarnitski in 1973. It also reported that the B and C1 reserve categories have increased significantly.

The 1992 results also cover analyses of rare earths (lanthanides): see Table 4-34.

4.5.2 FAR'S EXPLORATION DRILLING 2010 TO 2011

4.5.2.1 EXPLORATION DRILLING PROPOSAL 2009

An exploration drilling programme was proposed with the cardinal objective to obtain sufficiently detailed information to allow JORC “Indicated” resources within OB1 and OB2, with an additional option to evaluate OB3 and OB4 for an “Inferred” JORC resource category. To help define the exploration programme, appropriate historical archived materials were computerised and Datamine software used for evaluating the data, generating wireframe models of the vanadium layer and designing drillhole layouts for the project.

In order to achieve JORC (2012) classification for the resource, the drilling should establish continuity of the geological synclinal structures and grade continuity. The historical exploration profiles ranged from 650 m to 900 m along strike and GMR's experimental variogram suggested that this nominal historical 800 m spacing along the strike would be sufficient to provide an acceptable confidence level in estimating the primary vanadium resource: this spacing is also supported from the 1973 core drilling results. It was therefore on this principle that the diamond drilling programme was planned, but of course, complexity due to folding and faulting can add problems to the structural interpretation of the deposit, but it is not likely to fundamentally undermine the confidence in the grades and tonnages, especially on a global context. To confirm that the expected grade and structural continuity can be projected between historical profiles, some intermediate profiles would also be drilled for OB1. Table 4-24 summarises the planned drilling programme for a total 7,000 linear metres. For downdip drillhole intersections of the orebody, a target spacing of about 60 m was planned for the JORC "Indicated" category, with a maximum predicted depth of ore intersection at about 200 m from surface.

Table 4-24: FAR's Drilling Programme Summary

Ore Body	Number of Holes	Total m	Ave. metres per hole	JORC target
1	13	1,621	125	Indicated
2	23	3,727	162	Indicated
3	9	1,209	134	Inferred
4	3	461	154	Inferred
Total	48	7,018	146	

Initially, only alternate exploration profiles would be drilled and, based on the assessment of the vanadium grades and structural behaviour of the vanadium layer, final decisions would be made as to the optimum spacing for achieving the resource objectives.

Altyn-Ken (contractors) provided Longyear wireline core drilling, but the refurbished Soviet-era mounting platforms were restricted to angles between 60° to 90° (vertical), though optimally, shallower angles would have provided a greater flexibility for drilling the orebodies orthogonally. Contracted core size was NQ (47.6 mm) through the vanadium layer and reported core recovery was high at 90 %+. HQ (63.5 mm) was the larger core size.

4.5.2.2 LISTS OF PROPOSED DRILLHOLES

The three number drillhole ID (BHID) has a built-in code to allow easy identification of the target and location. The leading digit is the sequential number of the hole for the specific profile number (trailing digit) and the middle digit is the ore body number. Table 4-25 provides details of the proposed drillholes for OB1, Table 4-26 for OB2 and Table 4-27 for OB3 and OB4.

Table 4-25: Proposed Exploration Holes for OB1

ORE BODY	PROFILE	BHID	LENGTH	AZIMUTH	DIP	EASTING	NORTHING	RL
1	0	B110	80	45	60	12369513	4934650	451
1	1	B111	178	45	70	12370055	4934152	394
1	1	B211	206	225	60	12370073	4934169	386
1	2	B112	150	45	60	12370718	4933645	441
1	2	B212	115	225	60	12370724	4933652	444
1	2	B312	76	45	60	12370747	4933675	459
1	3	B113	78	225	60	12371319	4933092	490
1	3	B213	146	225	65	12371341	4933114	462
1	3	B313	104	45	65	12371369	4933142	482
1	4	B114	104	225	60	12371921	4932457	480
1	4	B214	126	45	70	12371975	4932511	468
1	4	B314	159	45	87	12371954	4932490	446
1	5	B115	99	225	60	12372432	4932034	541

Table 4-26: Proposed Exploration Holes for OB2

ORE BODY	PROFILE	BHID	LENGTH	AZIMUTH	DIP	EASTING	NORTHING	RL
2	2	B122	175	45	60	12370545	4933473	464
2	2	B222	198	45	70	12370503	4933429	459
2	2	B322	250	225	65	12370321	4933247	479
2	2	B422	207	45	60	12370303	4933230	526
2	3	B123	100	45	61	12371153	4932927	549
2	3	B223	170	45	70	12371126	4932900	500
2	3	B323	191	225	65	12370840	4932614	547
2	3	B423	106	225	60	12370820	4932593	594
2	4	B124	105	45	60	12371767	4932303	519
2	4	B224	170	45	60	12371727	4932262	499
2	4	B324	74	225	60	12371481	4932018	568
2	4	B424	121	225	60	12371503	4932041	544
2	5	B125	134	45	60	12372266	4931867	545

ORE BODY	PROFILE	BHID	LENGTH	AZIMUTH	DIP	EASTING	NORTHING	RL
2	5	B225	185	45	70	12372226	4931830	520
2	5	B325	225	225	65	12371912	4931515	526
2	5	B425	158	225	60	12371885	4931488	571
2	6	B126	131	45	60	12372851	4931254	603
2	6	B226	200	45	60	12372794	4931199	580
2	6	B326	171	225	60	12372478	4930884	612
2	6	B426	169	45	60	12372436	4930842	626
2	7	B127	101	44	60	12373189	4930876	627
2	7	B227	140	44	80	12373153	4930839	616
2	7	B327	246	224	60	12373037	4930248	697

Table 4-27: Proposed Exploration Holes for OB3 & OB4

ORE BODY	PROFILE	BHID	LENGTH	AZIMUTH	DIP	EASTING	NORTHING	RL
3	2	B132	209	225	60	12370213	4933140	525
3	3	B133	117	45	60	12370757	4932530	592
3	3	B233	109	225	60	12370618	4932391	590
3	4	B134	126	45	60	12371269	4931965	603
3	4	B234	148	225	60	12371109	4931804	606
3	5	B135	136	45	65	12371766	4931371	609
3	5	B235	97	225	60	12371674	4931277	618
3	6	B136	141	225	60	12372179	4930587	666
3	7	B137	126	224	60	12372902	4930106	775
4	6	B146	105	45	60	12373122	4931527	600
4	7	B147	172	224	60	12373454	4931152	553
4	7	B247	184	44	60	12373425	4931122	556

4.5.2.3 CURRENT STATUS OF FAR'S EXPLORATION DRILLING RESULTS

Table 4-28 below shows the current status of the drilling as at the date of this report and, for OB1, the basic primary objective has been achieved with the generation of an acceptable JORC (2012) resource. Intermediate profiles were also drilled at 2.5 and 3.5 metres, and this confirmed that the grade, geometry and continuity of the fold structure could be reasonably well predicted from the basic profile drilling positions. However, nearer the bottom of the syncline (hinge line) where the maximum curvature of folding occurs, the vanadium layers appear much thicker than expected.

OB2 appears to be the largest and possibly the deepest fold structure, but a drillhole located at the apparent centre of the syncline (see Figure 4-16) unexpectedly encountered vanadium at a depth of only 100 m. Thus OB2 appears to be affected by strike thrusting, where the vanadium layer has been displaced to a higher elevation. Only seven out of the planned 23 drillholes have been completed, but two holes missed the orebody.

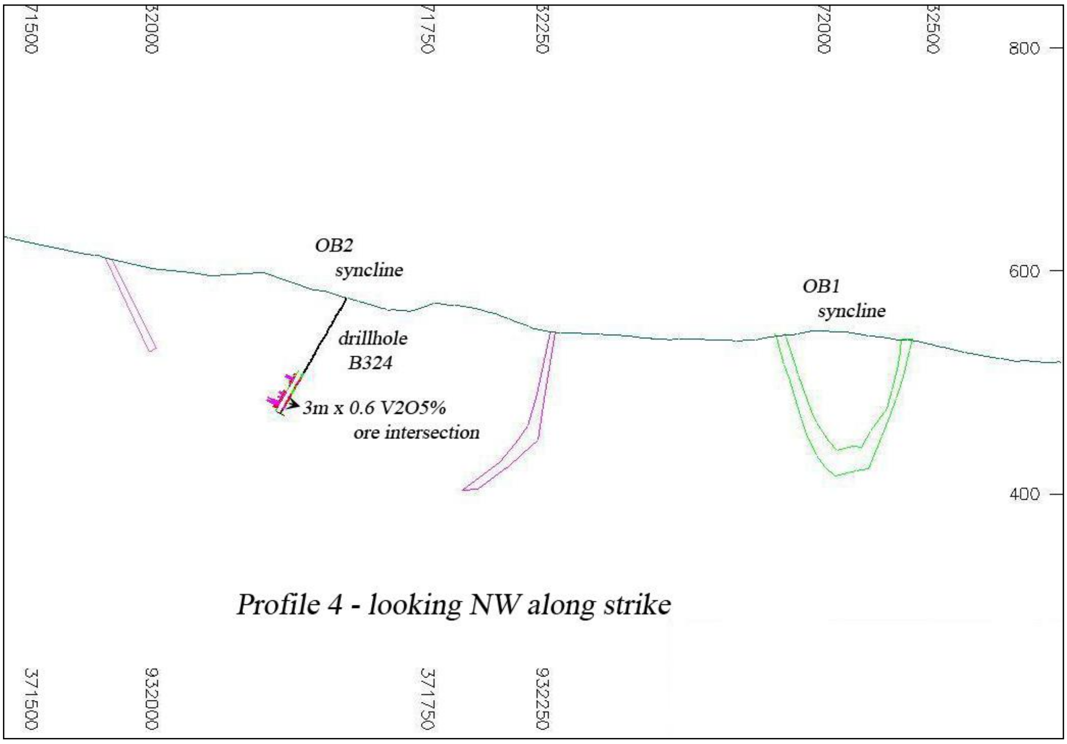


Figure 4-16: Unexpected Vanadium Intersected at 100m Depth - Centre of OB2 Syncline

OB3 fold style appears similar to OB1, though only one third of the drillholes have been completed, but evidence of strike thrusting in the axis of the fold has been seen.

For OB4, three drillholes were proposed, but no drilling has occurred to date.

Table 4-28: FAR Drilling Results – 2010 to 2011

BHID	OB	Profile	EOH Depth in metres	Laboratory V_2O_5 Analysis
B110	1	0	72.8	missed orebody
B111	1	1	64.0	Australia
B112	1	2	92.4	Australia
B112_5	1	2.5	35.0	XRF-FAR
B113	1	3	48.1	Australia
B113_5	1	3.5	105.5	Karaganda
B114	1	4	62.5	Australia

BHID	OB	Profile	EOH Depth in metres	Laboratory V_2O_5 Analysis
B115	1	5	71.0	Australia
B211	1	1	124.6	Australia
B212	1	2	102.9	missed orebody
B212_5	1	2.5	73.6	Australia
B213	1	3	126.2	Australia
B213_5	1	3.5	74.1	Karaganda
B214	1	4	76.8	Australia
B215	1	5	50.0	Australia
B311_bis	1	1	66.2	Australia
B312	1	2	66.0	Australia
B312_5	1	2.5	36.0	Australia
B313	1	3	67.6	XRF-FAR
B313_5	1	3.5	118.8	XRF-FAR
B314	1	4	149.0	Australia
B315	1	5	50.8	XRF-FAR
B412	1	2	134.3	Australia
B412_5	1	2.5	77.2	Australia
B124	2	4	73.0	no assay stopped 1m in OB
B125	2	5	88.4	Karaganda
B224	2	4	142.0	XRF-FAR
B225	2	5	94.3	XRF-FAR
B324	2	4	111.2	XRF-FAR
B325	2	5	231.0	XRF-FAR
B425	2	5	191.7	XRF-FAR
B135	3	5	93.0	Karaganda
B235	3	5	114.0	Karaganda
B335	3	5	180.2	Australia
	Total		3264.2	OB1 1945.4m, OB2 931.6m, OB3 387.2m

4.5.2.4 FAR'S SAMPLING & SAMPLE PREPARATION

High quality sampling and sample preparation are critical for acceptable QA/QC standards, and in 2010 FAR constructed a modern sample preparation laboratory for treating the core samples for their exploration programme. Half-core is cut at the core shed, using a diamond saw, under the supervision of the geologists to ensure that the core cutting produce unbiased samples: see Figure 4-17. At the nearby sample preparation laboratory, the half-core samples are crushed to 2 mm (crushers 1, 2 and

3 are shown in Figure 4-18), mixed and split using a Jones splitter and a 400 g fraction is extracted for pulverization. The rest of the sample is stored as a coarse reject for future additional analyses. The 400 g sample is ground to -200 mesh (74 µm) and 2 x 200 g samples are sent to external accredited laboratories, such as to Intertek and Ultra Trace in Perth, Australia, for analyses. Figure 4-19 illustrates this sample preparation flow diagram. The Perth laboratories endorsed FAR’s prepared pulped samples as being suitable for their analyses.

Figure 4-17: Cutting Core

Figure 4-18: Sample Preparation – Crushing Equipment

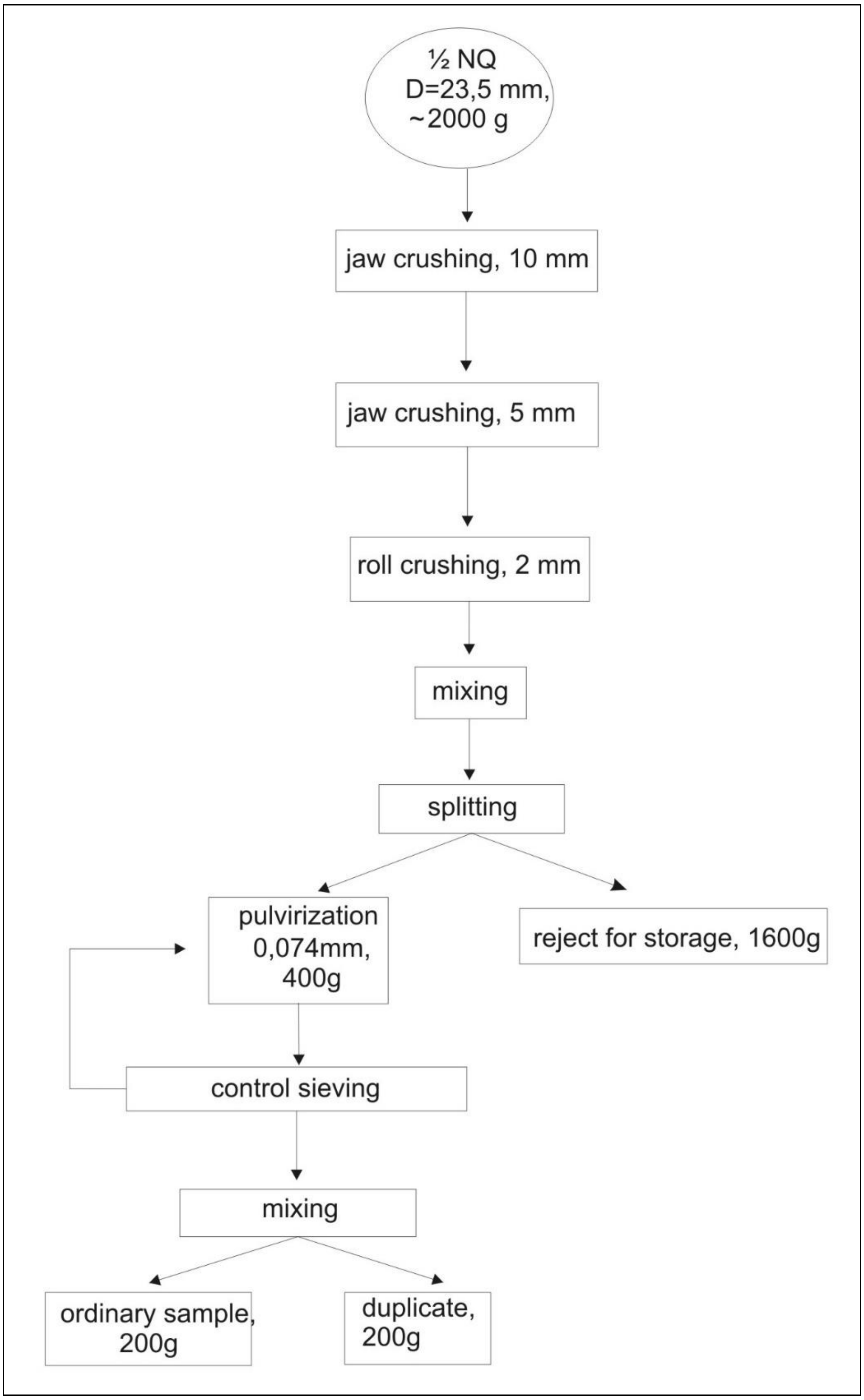


Figure 4-19: Sample Preparation Flow Diagram

4.6 QUALITY ASSURANCE AND QUALITY CONTROL

4.6.1 GEOGRAPHIC SURVEY

Typically, the former Soviet-era maps and plans do not display coordinate data and FAR undertook an extensive topographical re-survey of the Balausa area, which included locations of drillhole collars, sample trenches, adits, shafts and sample pits, and infrastructure. The survey data were compiled in MapInfo, from UMT (WGS84) survey measurements, and from which AutoCAD drawings were generated with a ‘local’ coordinate system that was parallel with the old historic grid (based on the Soviet Gauss Kruger (Pulkovo 42)). These AutoCAD plans were directly imported into DM and used for generating topographic wireframe surfaces (digital terrain models), plus imported AutoCAD drawings of historic maps and plans. There were a number of problems in locating the correct historical drillhole standpipe collars for the 1973 exploration holes during field surveys, because of indecipherable or absent identification numbers and, especially, where there was confusion from unknown drillhole collars related to the extensive 1990s drilling campaign: but after diligent examination of the 1973 archived plans and maps, together with the new DM topographic model, it was possible to locate with reasonable certainty, the actual drillhole locations. Based on GMR’s independent GPS audit measurements of a number of FAR’s surveyed drillhole locations, these were accurate and correlated with the historical plans. However, for a very few drillholes, it was not possible to exactly reconcile the field survey with the historical plans and the topographic surface, and this discrepancy was less than a maximum horizontal displacement of 25 m, which is considered to be within acceptable limits for this resource project.

4.6.2 OPEN PIT SAMPLE ANALYSES

The first QA/QC analytical exercise was to establish if the laboratory at the Balausa mine site can produce acceptably accurate and reproducible V_2O_5 results, when compared with internationally accredited laboratories. However, it must be emphasised that analytical results from the Balausa laboratory have never been used for resource estimations.

In October 2007, shallow rotary drillhole and surface trench samples, totalling 55, were taken within the oxide (+ or – transition) zone of the operating pit and submitted to the internationally accredited Alex Stewart laboratory in Kara Balta, Kyrgyzstan. The samples were re-numbered by a GMR geologist based in Almaty, prior to despatch to Kara Balta, to allow independent QA/QC checks to be made. Samples were pulverised to 85 % passing minus 75 µm (-200 mesh) screen and vanadium determined by Inductively Coupled Plasma Optical Emission Spectrometry. The Balausa methodology for analysis of the vanadium content was by either redox titration or gravimetric analysis. The assay results were compared with the Balausa’s laboratory sample-splits from the same batch of samples sent to Alex Stewart. It is also understood that Alex Stewart pulped the samples and produced splits, prior to despatch to an Australia laboratory, as a second validation check, but GMR do not know if these samples were sent to Australia.

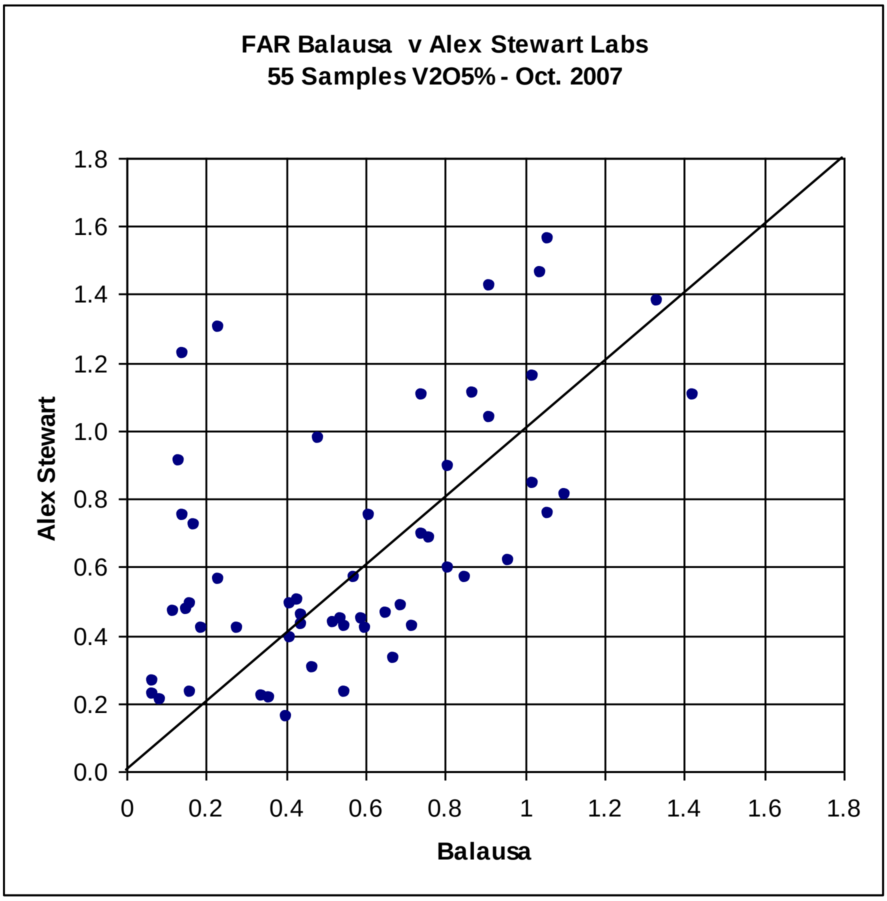


Figure 4-20: FAR v Alex Stewart laboratory V_2O_5 % – oxide open pit samples

The results in Figure 4-20 show a poor correlation between the laboratories. The reasons are unknown, but problems could include poor sample preparation, digestion and analysis, plus mislabelling. Average results were 0.56 % and 0.66 % of V_2O_5 %, between the Balausa and Alex Stewart laboratories. Note that these samples exhibited lower grades than expected for the oxide zone, as a number of samples were taken from adjacent contact waste zones – these samples were derived from 29x rotary holes and 26x channel samples.

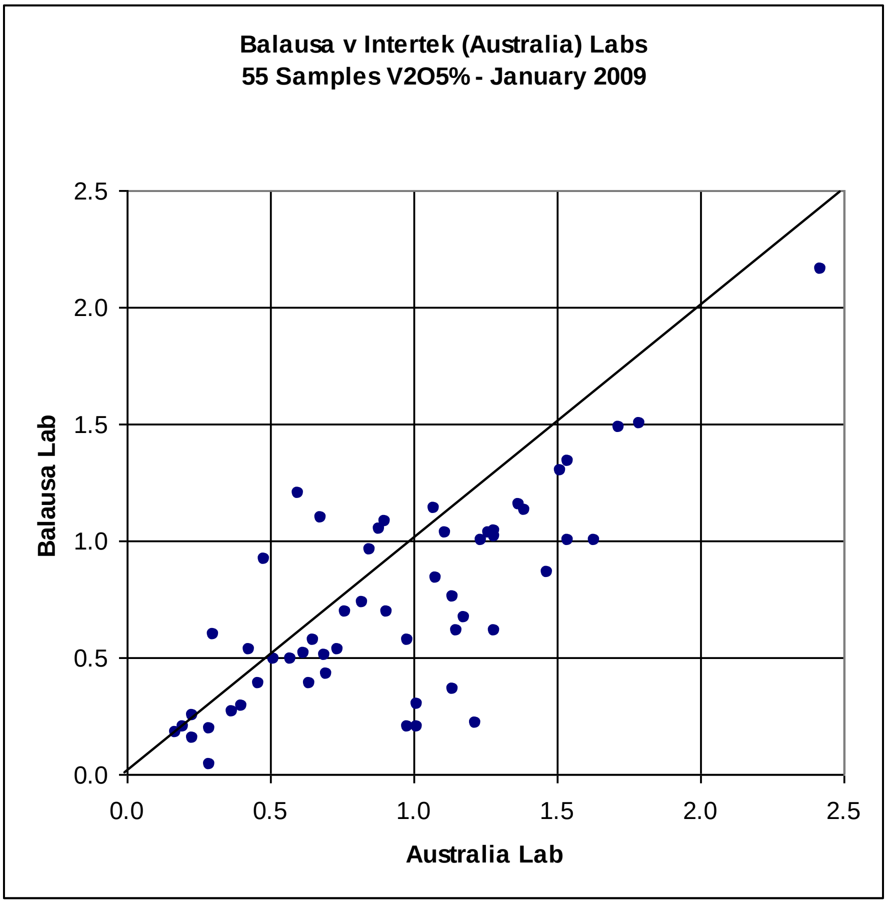


Figure 4-21: Balausa mean V_2O_5 % 0.73 % and Australia mean 0.93 % (oxide)

A second batch of samples was based on channel samples taken from the active open pit bench, located at the nose of the synclinal structure in OB1 in June 2008. These channels were dug by a bulldozer's ripper (10 to 30 cm deep x 1 m length and each sample weighed about 8 to 9 kg) and sample-splits were taken from these samples after homogenising. The results, illustrated in Figure 4-21, are much better than the 2007 results, with a much improved correlation trend between the both laboratories, but the accuracy of the Balausa lab results are not acceptable, as it has a distinct propensity to understate the actual contained vanadium – this lower V_2O_5 grade was also apparent for the 2007 results.

The preliminary comparison of results between the FAR Balausa laboratory for V_2O_5 % and the internationally accredited Australian laboratory (Intertek), showed that it was critical to send the samples for the FAR exploration drilling programme to an internationally accredited laboratory. The Intertek laboratory in Perth, Australia, was selected for this project. Average results (shown in Table 4-29) confirm the reported historical V_2O_5, C, Mo and U grade levels and are typical for the oxide zone, as indicated from the low carbon content and signature vanadium levels vis-à-vis primary zone.

Table 4-29: Quality Control Results Intertek Laboratory

C %	Mo ppm	U ppm	V ppm	V_2O_5%
0.83	149.02	45.45	5,213.02	0.93

Sample preparation and analytical methods at internationally accredited laboratories were undertaken to ensure appropriate analytical techniques. Note however, when FAR's new sample preparation laboratory was completed in 2010, the quality of the pulps was endorsed by the Australian laboratory as being acceptable for their analytical work.

4.6.2.1 ANALYTICAL METHOD - INTERTEK LABORATORY, AUSTRALIA

The analytical methods for the 55 samples sent to Intertek (14/01/2009) were as follows:

- V - Oxidative alkaline fusion using sodium peroxide flux, after roasting (ashing) to remove graphitic and other organic material (some analytes may be lost during roasting). Analysed by Inductively Coupled plasma optical (atomic) emission spectrometry. (50 ppm detection limit) – lab code RD/OES
- C - Digestion by hot acids and infrared analyses using induction furnace. (0.01 % detection limit) – lab code Ind/IR
- Mo - Oxidative alkaline fusion using sodium peroxide flux, after roasting (ashing) to remove graphitic and other organic material (some analytes may be lost during roasting.). Analysed by Inductively Coupled plasma mass spectrometry (1 ppm detection limit) – lab code RD/MS
- U - Oxidative alkaline fusion using sodium peroxide flux, after roasting (ashing) to remove graphitic and other organic material (some analytes may be lost during roasting.). Analysed by Inductively Coupled plasma mass spectrometry (0.1 ppm detection limit) – lab code - RD/MS

4.6.3 FAR'S EXPLORATION CORE SAMPLE ANALYSES

The first batch of core samples from FAR's exploration programme, totalling 110 samples, was sent to Intertek in March 2011, after sample preparation on site, these pulped samples weighed about 200 g each. FAR requested that Intertek test the validity of using a multi-acid digestion, vis-à-vis the more expensive sodium peroxide fusion approach, as used in 2009 (Section 4.6.2.1 above). A comparison of results on 20 test samples confirmed that both methods were similarly accurate in determining the actual contained V, Mo, and U elements, and the four-acid approach was therefore adopted for all FAR's exploration sample analyses. A further two batches of samples were sent to Intertek: giving a total of 370 samples. To confirm the Intertek laboratory results, for external QA/QC assessment, a number of random sample pulp duplicates were re-analysed by the internationally accredited laboratory Ultra Trace, also in Perth, amounting to 14 % (51 samples) of the Intertek samples (see Figure 4-22). Results show excellent precision and accuracy: the analytical technique is identical for both laboratories. Also, see the Intertek assay certificates for the first batch of samples in APPENDIX A.

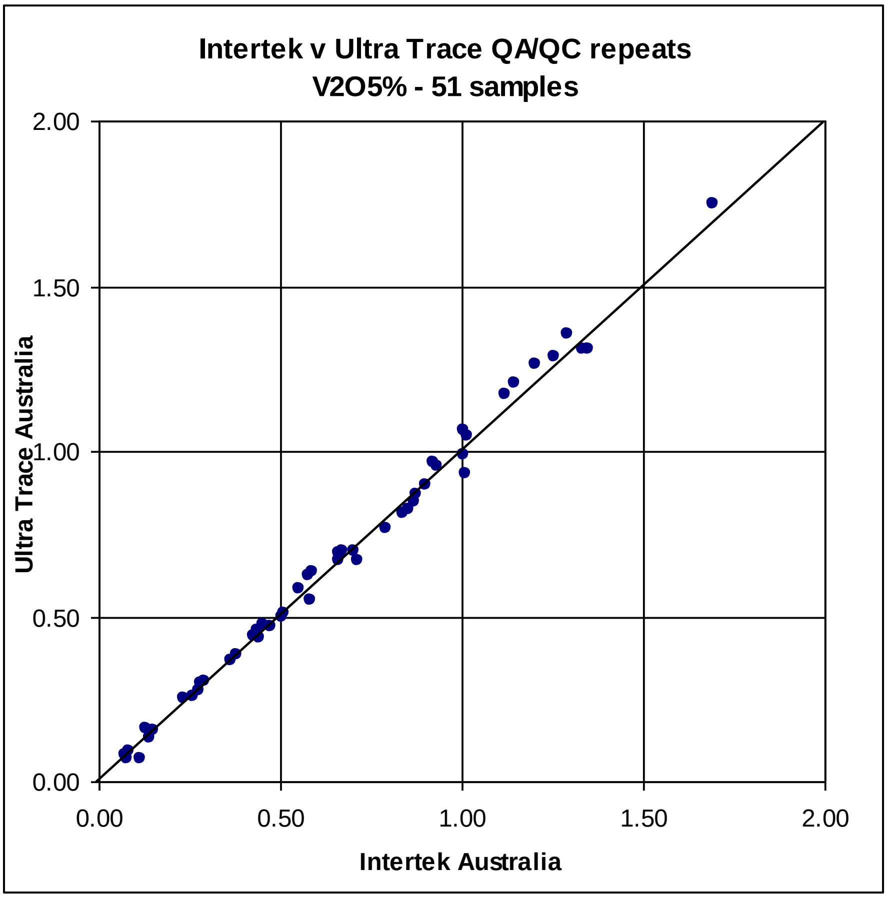


Figure 4-22: V_2O_5 – Ultra Trace (0.67 %) + QA/QC repeats at Intertek (0.67 %)

4.6.4 ANALYSES UNDERTAKEN BY OTHER LABORATORIES – KAZAKHSTAN

After the three main batches of samples were analysed in Australia, FAR decided to send core samples from the exploration programme to the TOO CentreGeoAnalysis Laboratories, in Karaganda, which is approved by the National Centre of Accreditation Kazakhstan. FAR also decided to upgrade its own laboratory by installing a modern XRF facility in Balausa. In addition, there were a number of trial sample splits analysed at the Kozlov laboratory (KazMex) which appears to be a former Soviet-era style laboratory.

4.6.4.1 ANALYSES AT THE KARAGANDA LABORATORY – V_2O_5

GMR assessed the quality of the Karaganda laboratory, by comparing sample splits which were also analysed in Australia. The results are illustrated in Figure 4-23, and it shows a good correlation and good accuracy. On this basis, GMR accepted the use of this laboratory for the resource estimate. Although GMR does not have the details of the analytical method used by Karaganda, FAR stated that it was similar to the method used in Australia. It should be noted that only V_2O_5 was analysed at the Karaganda laboratory, and this limited the final number of samples containing C, U and Mo results for the by-product resource estimations.

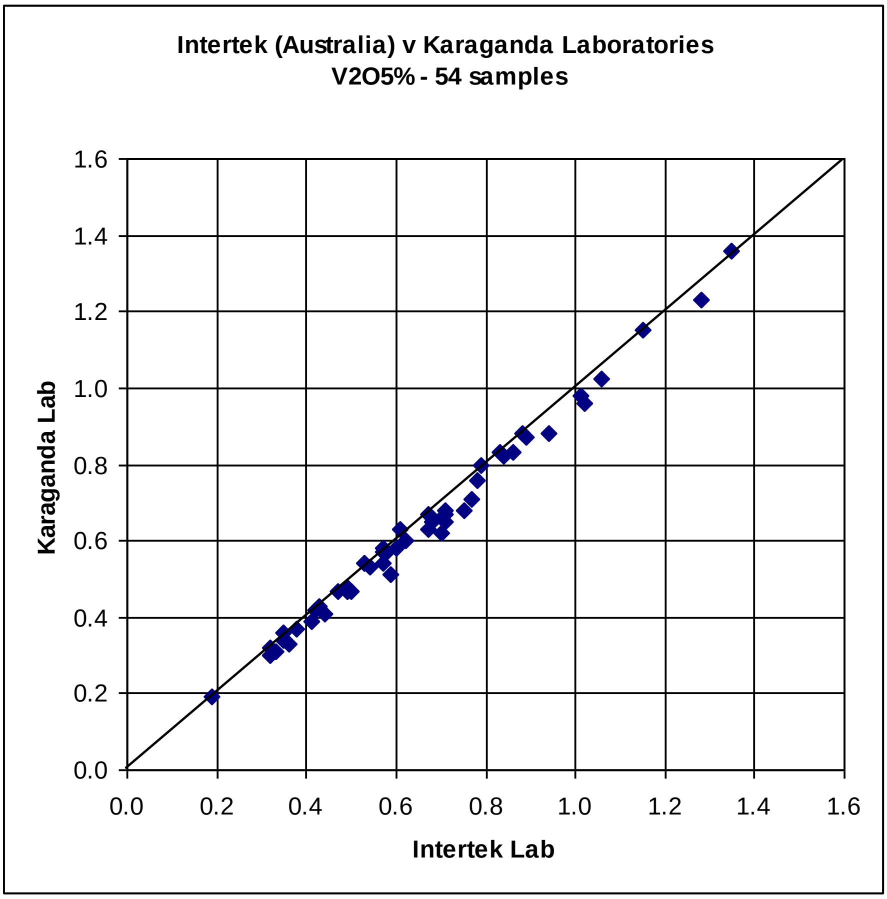


Figure 4-23: Scatterplot V_2O_5 Intertek (0.65 %) v Karaganda lab (0.63 %)

4.6.4.2 BALAUSA XRF LABORATORY ANALYSES OF V_2O_5

FAR had installed modern XRF equipment to specifically analyse elements on site. The XRF system is very accurate in quantifying the contained metals and is the preferred technique for the determination of the major element oxides, as well as some trace elements. However, it needs careful preparation of the samples, plus calibration of standards and matrix corrections and adjustments for drift etc., to avoid false readings. At Balausa, the press powder technique, vis-à-vis fusion technique, is used but is subject to particle size and matrix effects which increase analytical uncertainty.

A comparison between the analytical results from the Karaganda laboratory and the Balausa XRF results were used to determine the acceptability of the XRF method for the resource estimations. The scatterplot in Figure 4-24 shows the results of 117 samples.

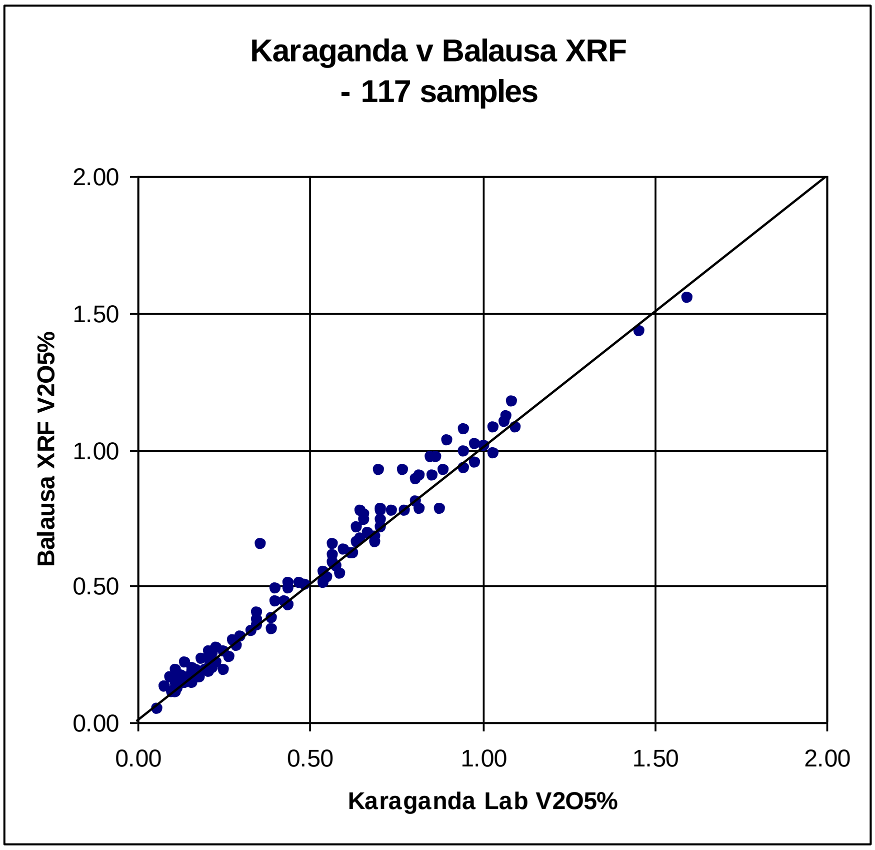


Figure 4-24: Scatterplot Karaganda v Balausa XRF V_2O_5

The average V_2O_5 is 0.48 % at Karaganda and 0.51 % Balausa XRF lab. Overall, the correlation is good but the accuracy slightly lower than expected, with Balausa having 6 % higher grade than Karaganda, which is reasonable and considered to be within acceptable limits and not a fundamental flaw. The data comparison was mostly from OB2 and OB3 samples, and included ore and adjacent low grade zones - however, with the16 samples from OB1 the XRF result was 7.4 % higher than the Karaganda results, though this is still considered by GMR to be within acceptable limits.

Note: when the above problems were recognised, sample homogenization improvements by finer sample grinding, for the press powder technique, has improved the XRF accuracy of the results, according to FAR.

4.6.4.3 ANALYSES AT THE KOZLOV LABORATORY – V_2O_5 & C

The results of the Kozlov laboratory in Kazakhstan were compared with the Balausa XRF and with the Intertek results. Figure 4-25 shows the comparison with the Balausa XRF results for 41 samples and highlights a fundamental flaw with the accuracy of the Kozlov analytical procedure, as the results are far too high, and totally unacceptable for JORC-based resource estimations. Comparisons with other laboratories also show anomalous high-grade trend problems with the Kozlov analyses: see Intertek laboratory results in Figure 4-26.

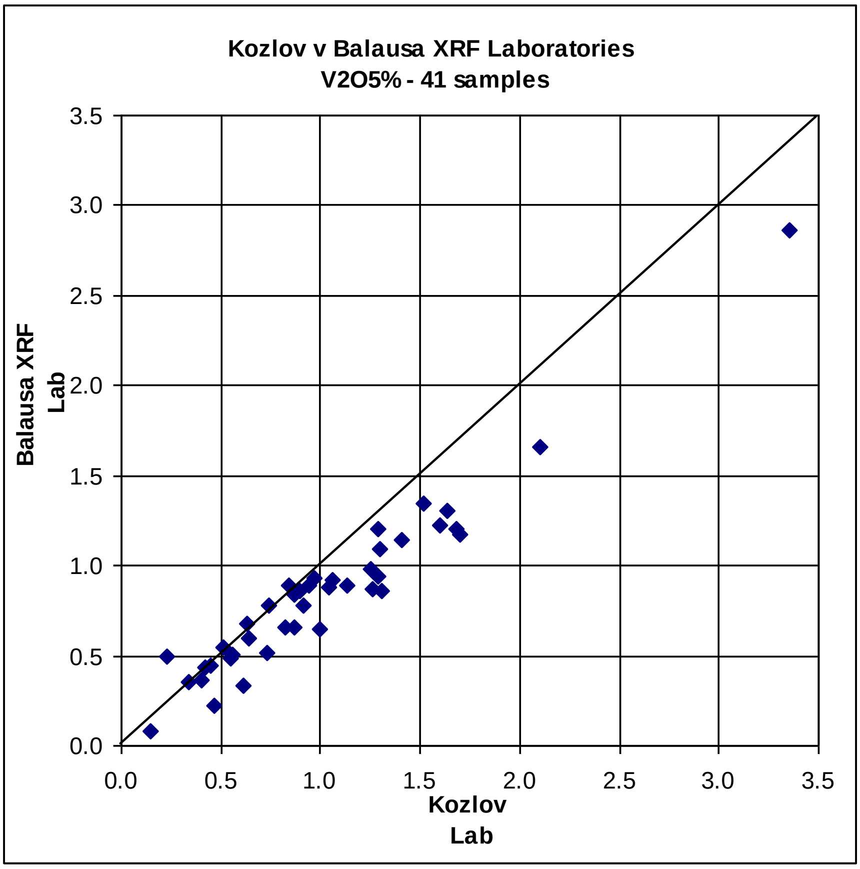


Figure 4-25: Scatterplot V_2O_5 Balausa XRF (0.84 %) v Kozlov (1.01 %)

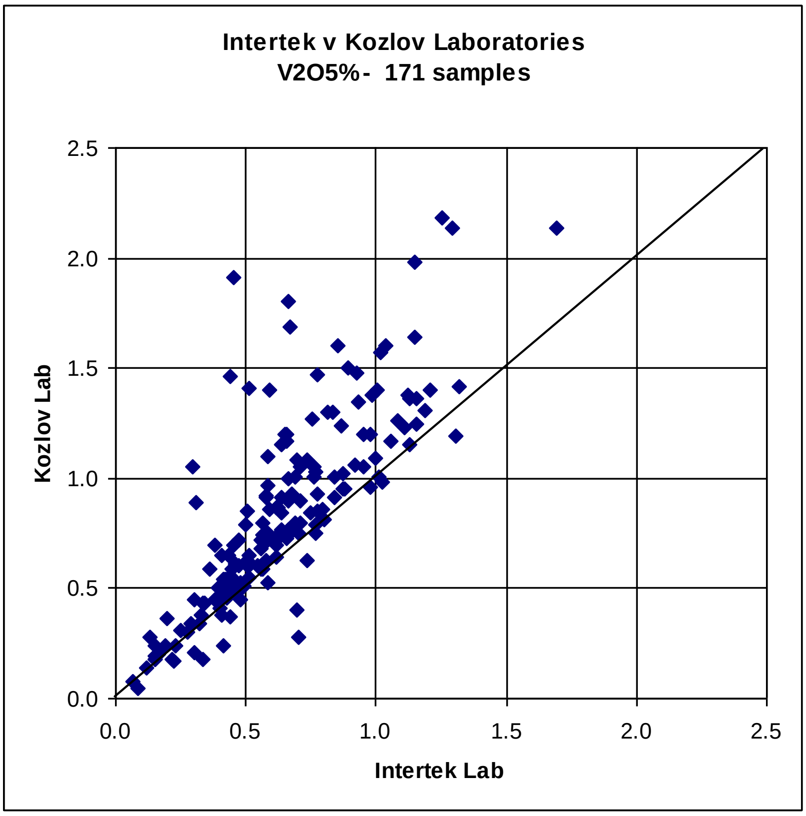


Figure 4-26: Scatterplot V_2O_5 - Intertek (0.64 %) v Kozlov lab (0.85 %)

Similar to the previous example, the 171 Kozlov results show big discrepancies with the Intertek results and, again, are highly exaggerated, especially above 0.5 % V_2O_5.

In addition to the V_2O_5 analyses, it has been possible to assess the Kozlov carbon analyses with the Intertek results for OB1. These results show quite a poor correlation and the average carbon grades are, Kozlov 14.4 % versus Intertek mean of 11.7 % - this again confirms that the Kozlov results do have a high grade bias. See Figure 4-27.

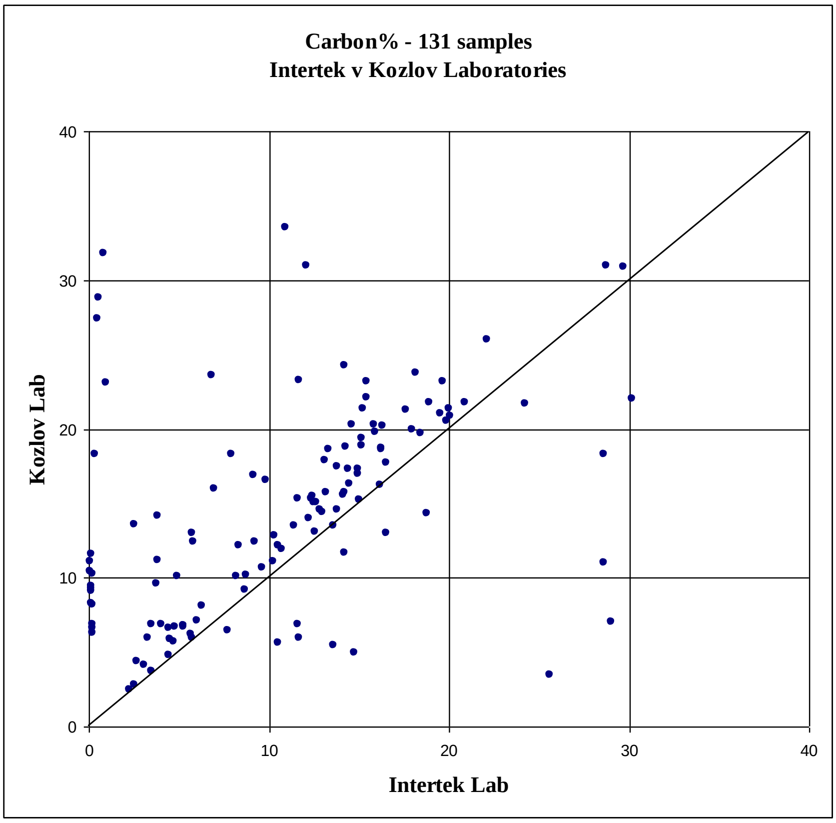


Figure 4-27: Scatterplot Carbon - Intertek (11.73 %) v Kozlov lab (14.36 %)

4.6.4.4 ANALYTICAL V_2O_5 RESULTS FOR BALAUSA LABORATORY (PRE-XRF)

Similar to the findings for the open pit oxide analyses at Balausa in 2007 and 2009, see Section 4.6.2 above, the Balausa laboratory (pre-XRF) results for the FAR exploration samples continue to show the same analytical accuracy problems, especially when compared with the Intertek results, where there is also a low correlation or repeatability. See Figure 4-28 and Figure 4-29.

The pre-XRF Balausa laboratory analyses are therefore not reliable enough to be used for the JORC-based estimations, and neither are the Kozlov analyses.

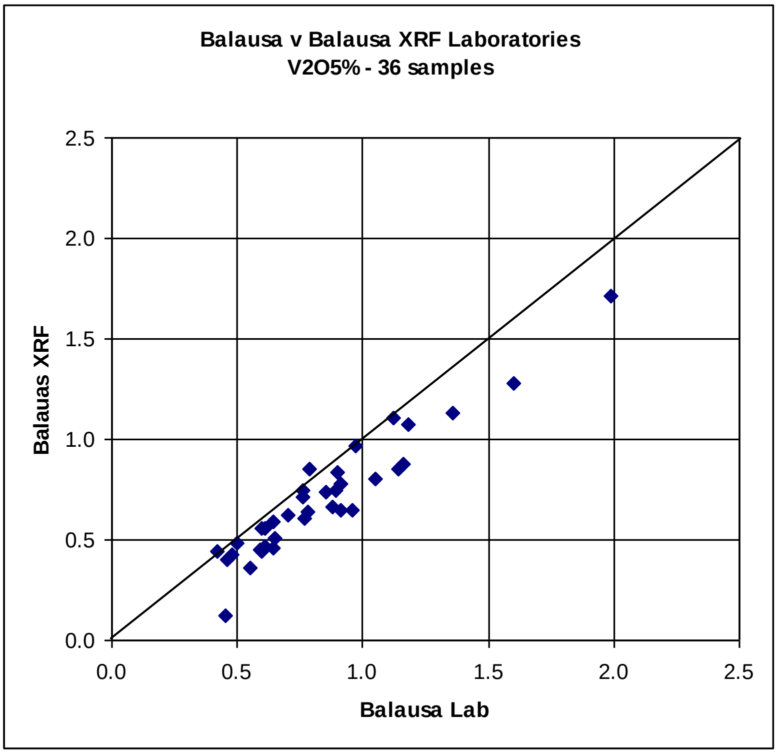


Figure 4-28: Scatterplot V_2O_5 Balausa lab (0.84 %) v Balausa XRF (0.70 %)

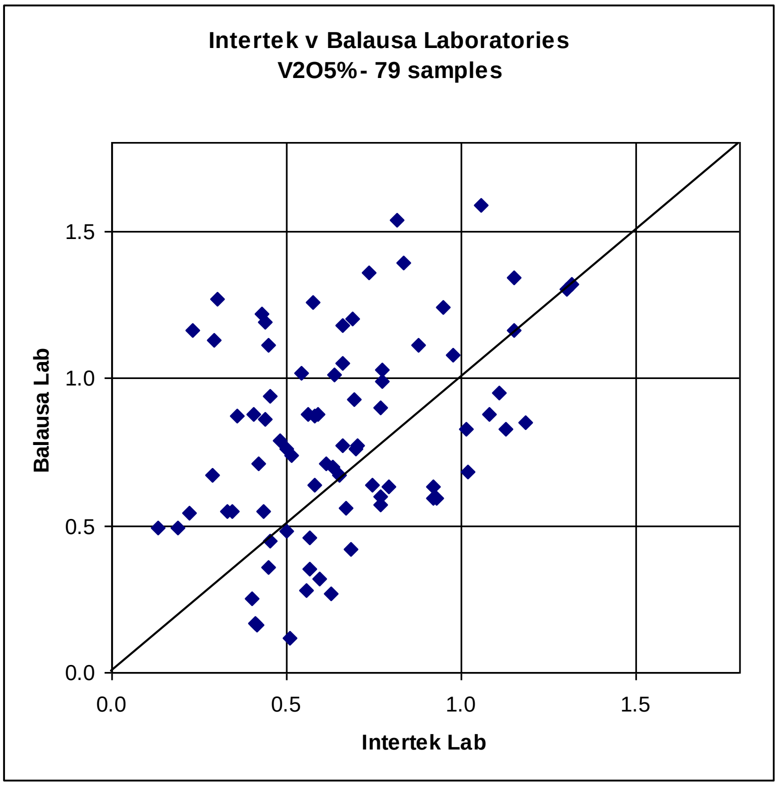


Figure 4-29: Scatterplot V_2O_5 Intertek (0.65 %) v Balausa lab (0.81 %)

4.6.5 QA/QC ANALYTICAL ASSESSMENT

Many FAR core samples had analytical results from more than one laboratory, and GMR generated an algorithm using the DM macro facility, to prioritise the use of V_2O_5 results, according to confidence levels, as ascertained from the QC assessment studies. The confidence levels in the laboratory results, from highest to lowest are as follows:

Australian labs > Karaganda > Balausa XRF

Note that Kozlov and the pre-XRF Balausa laboratories were too unreliable to be used for the grade estimations.

When comparing the recent FAR drilling results to the 1973 Komarnitski Soviet-era drillhole results at similar orebody intersection locations, expected levels of vanadium content were found and therefore these historical former Soviet-era drillhole results were accepted for the purpose of this mineral resource estimate.

4.6.6 REE ANALYSES

4.6.6.1 REE ANALYSES - AUSTRALIA

Rare earth elements can be an important economic contributor to the resource and to make a preliminary quantitative assessment of a selected range of these elements, within the vanadium layer for OB1, a number of FAR's sample pulps at the Ultra Trace Laboratory in Perth were composited into four compound samples, representing specific exploration section lines (1, 2, 3, & 4). Aliquots of these samples were selected from drillholes representing intersections through the primary vanadium layer and composited accordingly. See Table 4-30.

Table 4-30: Sample composites for REE analysis

Profile Line	Drillholes	Sample #'s	Total No. of Pulp Samples	Composite #	V_2O_5%
1	B211 & B311	B211-2 to B211-6 B311-2 to B311-9	13	Comp 1	0.68
2	B412	B412-2 to B412-8 B412-10 to B412-13	11	Comp 2	0.71
3	B113 & B213	B113-4 to B113-10 B213-2 to B213-24	30	Comp 3	0.65
4	B314	B314-2 to B314-17 B314-20 to B314-27	24	Comp 4	0.73

Note: in addition to rare earth elements, gold, platinum and palladium were also analysed. See Table 4-31 for a summary of the Ultra Trace results.

Table 4-31: REE Results Ultra Trace laboratory

Sample	Au1	Pt	Pd	La	Ce	Pr	Nd	Sm	Gd	Ho	Er	Yb	Lu	Y	Sc
	ppb	ppb	Ppb	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Comp 1	15	40	60	39	46	10	43	11	14	3.6	11	10	1.4	127	<
Comp 2	34	40	60	47	58	13	59	15	18	4.0	12	11	1.4	140	<
Comp 3	30	40	60	36	40	8	35	8	8	2.4	7	6	0.8	95	<
Comp 4	36	40	40	45	54	11	47	10	12	3.0	10	8	1	123	<
Mean	**29**	**40**	**55**	**41**	**49**	**10**	**46**	**11**	**13**	**3**	**10**	**9**	**1.2**	**121**	**<**

Repeat checks of these analyses were completed by Intertek and these results are summarised in Table 4-32.

Table 4-32: REE Results Intertek laboratory

Sample	Au1	Pt*	Pd*	La	Ce	Pr	Nd	Sm	Gd	Ho	Er	Yb	Lu	Y	Sc
	ppb	ppb	Ppb	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Comp 1	15	21	42	38	46	10	44	10	13	3.3	11	10	1.4	127	<
Comp 2	25	20	59	44	58	12	56	13	17	3.7	12	11	1.4	134	<
Comp 3	25	20	49	51	75	11	49	10	11	2.5	8	7	1.1	105	<
Comp 4	18	17	40	43	52	11	47	10	13	2.9	10	9	1.3	125	<
Mean	**21**	**19**	**47**	**44**	**58**	**11**	**49**	**11**	**13**	**3**	**10**	**9**	**1.3**	**123**	**<**

** mean of 2 repeats*

The results from both laboratories show acceptable repeatability and accuracy, however, for sample Comp 3 the La and Ce results show a noticeable dichotomy and also a problem with all Pt results. GMR therefore asked Intertek to check their Pt assays, and the Intertek re-analyses confirmed that the Ultra Trace Pt results were “too high” at 40 ppb - Intertek communiqué dated April 2012.

The overall composite REE results show a mean grade of 335 ppm for the four composites and, individually, the average composite REE grades are similar. This suggests a regular grade level distribution throughout OB1.

An initial analysis check by Intertek, using a multi-acid approach, showed REE levels at only about half the level compared to the fusion method, as certain minerals such as cassiterite, rutile, ilmenite, zircon, tantalite-columbite, wolframite, garnets, and xenotime incorporate lots of REE in their structures and these may be the dominant REE carriers in the Balausa vanadium bearing sediments for some of these elements. These minerals will therefore not dissolve completely in four acid digests, hence the acid digestion approach show lower levels than the fusion. This is less noticeable in rocks with higher REE contents, as the REE carriers tend to be more soluble in acids, such as bastnäsite and monazite minerals, although REE locked up in xenotime will not be released fully in most cases. See Intertek REE check result assay certificate in APPENDIX A.

AU, PT, AND PD

The samples have been analysed by fire assay using a 40 g or 25 g (Intertek) portion of the composite sample. This is a classical fire assay process and will give total separation of gold, platinum and palladium in the sample. Analysed by Inductively Coupled Plasma Optical Emission Spectrometry.

LA, CE, PR, ND, SM, GD, HO, ER, YB, LU AND Y

The samples have been fused with sodium peroxide and subsequently the melt has been dissolved in dilute hydrochloric acid for analysis. Due to the high furnace temperatures volatile elements are lost. This procedure is particularly efficient for determination of major element composition (including silica) in the samples or for the determination of refractory mineral species. Analysed by Inductively Coupled Plasma Mass Spectrometry.

SC

Sc has been determined by Inductively Coupled Plasma Optical Emission Spectrometry.

AG (ONLY INTERTEK ANALYSED THIS ELEMENT)

Multi-acid digest including Hydrofluoric, Nitric, Perchloric and Hydrochloric acids in Teflon Tubes. Analysed by Inductively Coupled Plasma Mass Spectrometry.

The final REE analytical methods have now been optimised, in accordance to the Ultra Trace and Intertek laboratory methodology, and this approach provides excellent results for total determinations of contained rare earth metals at Balausa.

4.6.6.2 REE ANALYSES – KARAGANDA LABORATORY

The Karaganda Laboratory analysed 477 drillhole pulp splits for vanadium, from the FAR exploration programme, for mostly OB1, with minor samples from OB2 and OB3. These pulps were split into three class limits, based on V_2O_5% values, into 0.25 to 0.5 %, 0.5 to 1.0 % and 1.0 to 1.5 %. Pulps within each class limit were composited, using about 15 samples per composite. The final 31 composites were assayed for REE, and results are listed in Table 4-33. The V_2O_5% value was based on the average grade of the pulps within each composite.

GMR has no direct confirmation of Karaganda's REE analytical methods, in support of this QA/QC assessment. However, the Karaganda average REE results show a good correspondence with the Australian laboratory results, even though the samples are of different composites. The only exceptions are where the Karaganda's detection limits are not sensitive enough; e.g. for the Ho, Er, and Lu elements.

Table 4-33: Karaganda REE Analyses – Based on Vanadium Pulp Sample Composites

	Lu	Eu	Dy	Tm	Yb	Y	Er	Nd	La	Gd	Ho	Pr	V_2O_5%
	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	
Limit	5	?	5	2			10		20	10	5	10	
		1			14	69		23	24				0.37
		2			14	100		37	37				0.42
		1			11	57		29	27				0.4
		1			14	88		38	36				0.38
		1			15	91		33	29				0.37
		2			18	127		36	34				0.39
		1			16	83		32	26				0.37
		2			18	116		39	38				0.38
					11	53		19					0.35
		1			15	92		31	29				0.37
		2	6		29	162		48	41				0.72
		2	5		29	159		44	41				0.77
		3	7		25	203		68	38	11		11	0.75
		3	6		29	175		63	41	10		11	0.69
		2			27	149		46	38				0.69
		2			25	136		40	36				0.68
		2	5		27	235		48	46				0.74
		3	6		30	189		61	37			11	0.77
		2	6		30	234		51					0.69
		2	5		28	164		50	42				0.73
		3	6		30	178		59	44				0.68
		2	6		30	198		45					0.74
		2			24	143		47	46			10	0.72
		2			20	109		39	33				0.65
		2			23	205		50	43			10	0.73
		2	6		27	150		44	43				0.69
		3	5		42	217		75	58	11		15	1.2

	Lu	Eu	Dy	Tm	Yb	Y	Er	Nd	La	Gd	Ho	Pr	V_2O_5%
	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	
		2	6		35	157		52		10			1.4
		3	7		36	200		67	53	13		12	1.15
		4	10		50	239		87	37	18		14	1.5
		3	6		40	198		75	57	12		15	1.26
Mean		2	6		25	151		48	39	12		12	0.67

Yttrium is widely distributed in this deposit and to illustrate its distribution in relation to vanadium levels, Figure 4-30 shows a distinct broad propensity for the higher yttrium levels to be associated with raised levels of V_2O_5, though the lowest class group at <0.5 V_2O_5%, and the higher class groups above >0.5 %, do not exhibit notable sensitivity trends in relation to changes in the yttrium levels within their respective groups.

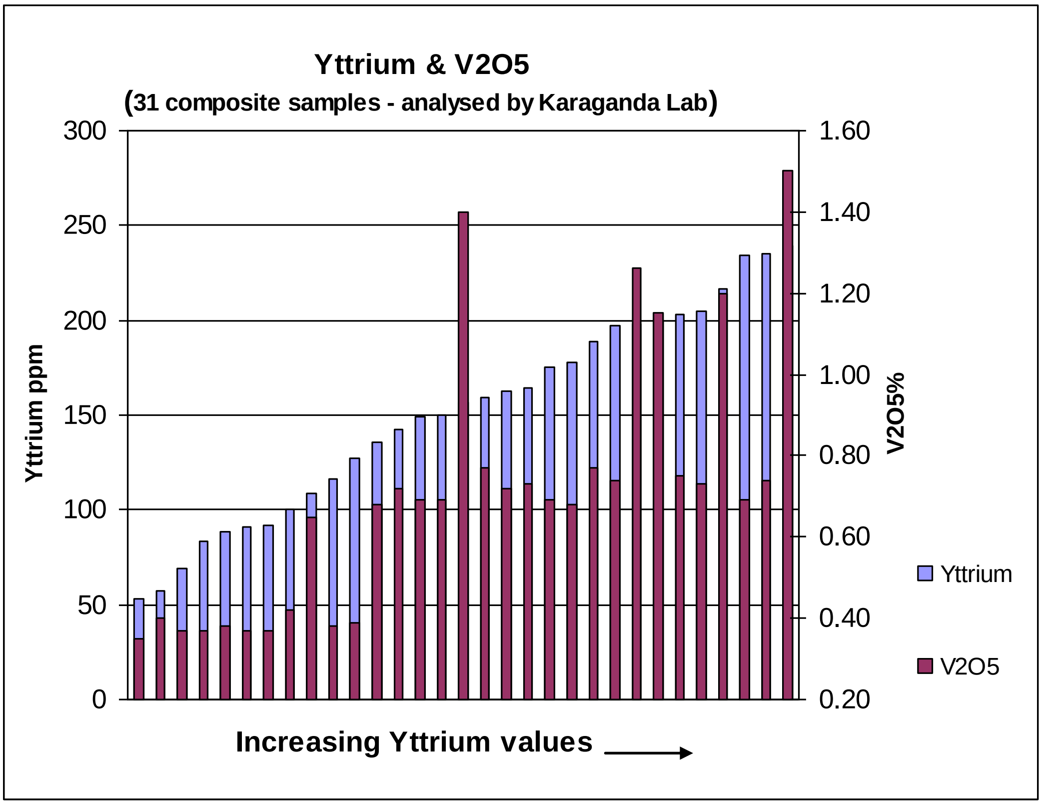


Figure 4-30: Yttrium and V_2O_5% levels

GMR recommend that a comprehensive study on the contained REE mineralogy is undertaken as soon as possible, in order to identify the specific REE containing minerals which are peculiar to Balausa, in order to improve the selection of the most appropriate elements for analysis and to optimise the economic contribution potential of the deposit's rare earths.

4.6.6.3 REE ANALYSES: TASHKENT RESEARCH INSTITUTE 1993

In the 1992 summary report, a table on REE analyses was submitted and these results are displayed in Table 4-34. GMR has no additional information, but the yttrium levels are broadly similar to those reported above.

Table 4-34: Rare Earth Element Grades 1993

REE	Grade in ore (g/t) ppm	Amount REE in (t)	Tonnes of Ore (millions)
Yttrium	142	10,366	73
Lanthanum	48	3,504	73
Cerium	60	4,380	73
Samarium	14	1,022	73
Praseodymium	30	2,190	73
Neodymium	30	2,190	73
Europium	4	292	73
Gadolinium	7	511	73
Terbium	3	219	73
Dysprosium	16	1,168	73
Holmium	7	511	73
Erbium	17	1,241	73
Thulium	3	219	73
Ytterbium	12	876	73
Lutetium	2	146	73
Total Lanthnium group	193	14,089	73
Total		**24,455**	

Note: this is the same ore tonnage as reported in the 1997 GKZ confirmation of off-balance reserves as shown in Table 4-14.

4.6.7 FAR'S BULK DENSITY DETERMINATIONS

4.6.7.1 INTRODUCTION

Historically, numerous rock density determinations have been reported, but most of this work was related to the oxidised or transition vanadium rocks at surface or at immediate sub-surface locations. For the primary vanadium rocks at depth, Komarnitski reported in 1973 that bulk density determinations of primary vanadium drillcore pieces averaged 2.42 and ranged from 2.24 to 2.62 for 48 samples – however, a density of 2.50 was used for official ore reserves.

For the reporting of historical ore reserves, the densities listed in Table 4-35 have been used.

Table 4-35: Historical Rock Bulk Densities Used for Ore Reserves

Year	Oxide	Primary
1947	1.60	2.25
1973		2.50
1992	1.73	2.00
2013	1.70	2.10

This variability in the historical bulk densities for the reserves necessitated a reappraisal of especially the primary vanadium rocks (estimated 95 % of total resource), and FAR undertook to measure the bulk density (mass per unit of volume) using drillcore from FAR's drilling exploration programme of OB1. An independent laboratory, TOO GeoAnalytica, based in Almaty performed the analyses and, for external QA/QC purposes, the Kazmekhanobr laboratory in Almaty re-analysed a random selection of these core samples.

Bulk density implies the density of extractable volumes of rock inclusive of voids, or "volume density". The in-situ density includes the void and grain boundary water present in the rock in its natural state. Whilst the latter is important for estimation of the tonnage of material to be moved during mining, for resource estimations a dry bulk density is required (Lipton, 2000 (13)) and this was used for these density determinations. Prior to this bulk density assessment, a similar number of true density (defined as density of particles that make up a solid) determinations had been undertaken by FAR, but although it can be a useful indicator, it is not reliable for resource tonnage estimations.

4.6.7.2 METHOD IN DETERMINING BULK DENSITY

From the OB1 drilling, 49 core samples were selected which provided a representative suite of samples that defined the distinct primary vanadium rock characteristics found at Balausa. It also included a few samples from the hangingwall and footwall margins of the vanadium layer. Note that most of the core samples had already been split and sampled for metal analyses, and so only half-core remained, plus some whole core samples taken from beyond the vanadium layer.

The following method was used to determine the bulk density at the GeoAnalytical laboratory in Almaty:

- The core pieces were first dried in an oven at a temperature of 105 degrees Celsius.
- Samples weighed in air.
- Each sample coated with a thin layer of wax and weighed again.
- Wax coated sample was immersed in deionised water and weighed.

Determination of the bulk density was as follows:

$$Bulk\ Density\ \frac{\mathrm{A} * p\ (\mathrm{liq})}{\mathrm{B} - \mathrm{C} - ([\mathrm{B} - \mathrm{A}]\ /\ \mathrm{p(wax)})}$$

Where:

A - weight of the sample in the air

B - weight of the coated sample in the air

C - weight of coated sample immersed in the water

p (wax) – specific gravity of the wax

p (liq) – specific gravity of the water

Note:

Specific gravity of the wax = *0.862 g/cm^3.*

Specific gravity of deionised *water = 1.0 g/cm^3.*

Table 4-36 provides details of the results of the bulk density determination.

Table 4-36: Bulk Density Results

Sample #	Depth down hole			weight in air. g	weight with wax in air. g	weight with wax in water. g	Bulk Density	Ore Samples
	Drillhole	From (m)	To (m)					
DSG-1	B213.5	60.6	60.7	336.57	343.69	193.26	2.37	2.37
DSG-2	B412	39.8	39.9	139.56	144.58	81.26	2.43	
DSG-3	B213.5	59.2	59.3	204.44	209.46	118.75	2.41	2.41
DSG-4	B313.5	106.5	106.6	288.97	299.83	160.87	2.29	2.29
DSG-5	B412	43.5	43.6	122.79	126.43	67.32	2.24	2.24
DSG-6	B412	125.3	125.4	154.64	161.29	91.14	2.48	2.48
DSG-7	B313.5	90.2	90.3	332.89	341.09	194.28	2.42	2.42
DSG-8	B313	50.3	50.4	190.68	198.43	101.48	2.17	2.17
DSG-9	B211	117.3	117.4	123.54	129	64.56	2.13	2.13
DSG-10	B412	121.1	121.2	459.7	469.52	284.11	2.64	
DSG-11	B313	63.5	63.6	103.58	110.66	60.77	2.49	
DSG-12	B313.5	104.6	104.7	156.66	160.69	96.72	2.64	2.64
DSG-13	B412	94.4	94.5	260.46	265.4	157.56	2.55	
DSG-14	B313	56.4	56.5	305.14	313.93	173.97	2.35	2.35
DSG-15	B113	44.7	44.8	348	357.27	205.17	2.46	2.46
DSG-16	B211	119.3	119.5	172.87	177.83	99.08	2.37	2.37

Sample #	Depth down hole		To (m)	weight in air. g	weight with wax in air. g	weight with wax in water. g	Bulk Density	Ore Samples
	Drillhole	From (m)						
DSG-17	B313.5	98.2	98.3	322.69	329.34	186.77	2.39	2.39
DSG-18	B313.5	85.4	85.5	265.75	276.83	147.6	2.28	2.28
DSG-19	B213.5	63.0	63.1	271.54	278.76	164.57	2.57	
DSG-20	B313.5	100.8	100.9	180.98	187.4	102.28	2.33	2.33
DSG-21	B213	117.1	117.2	363.56	372.42	220.79	2.57	2.57
DSG-22	B213	118.3	118.4	271.69	281.04	158.41	2.43	2.43
DSG-23	B213	122.5	122.6	187.6	192	107.53	2.36	2.36
DSG-24	B114	47.4	47.5	207.42	213.63	121.27	2.43	2.43
DSG-25	B114	46.8	46.9	158.16	162.51	91.15	2.38	2.38
DSG-26	B114	44.8	44.9	179.47	183.59	102.33	2.35	2.35
DSG-27	B112	77.2	77.3	161.1	164.9	102.49	2.78	2.78
DSG-28	B112	79.5	79.6	134.9	138.62	83.05	2.63	2.63
DSG-29	B112	82.3	82.4	136.66	141.95	79.94	2.44	2.44
DSG-30	B314	94.9	95.0	222.94	231.39	134.24	2.55	2.55
DSG-31	B314	92.8	92.9	161.74	166.1	98.04	2.57	
DSG-32	B314	95.5	95.6	120.91	124.47	71.36	2.47	2.47
DSG-33	B215	25.6	25.7	185.29	189.81	117.73	2.77	2.77
DSG-34	B215	26.9	27.0	138.74	142.97	81.3	2.44	2.44
DSG-35	B215	28.0	28.1	177.24	182.73	101.65	2.37	2.37
DSG-36	B115	23.2	23.3	109.42	115.06	52	1.94	1.94
DSG-37	B115-4	26.3	26.4	145.85	152.3	80.43	2.26	2.26
DSG-38	B115-5	22.3	22.4	164.78	169.75	99.08	2.54	2.54
DSG-39	B212.5	52.2	52.3	132.53	138.12	80.65	2.6	2.6
DSG-40	B215.5	52.9	53.0	257.16	264.78	143.62	2.29	2.29
DSG-41	B212.5	54.9	55.0	347.88	355.82	203.96	2.44	2.44
DSG-42	B212.5	57.5	57.6	252.16	259.83	151.06	2.52	2.52
DSG-43	B212.5	58.4	58.5	189.01	194.36	106.39	2.31	2.31
DSG-44	B312.5	22.5	22.6	231.99	237.58	125.67	2.2	2.2
DSG-45	B312.5	25.0	25.1	215.1	220.76	112.99	2.12	2.12
DSG-46	B312.5	26.7	26.8	187.16	191.93	98.93	2.14	2.14
DSG-47	B412.5	58.6	58.7	270	276.82	155.81	2.39	2.39
DSG-48	B412.5	61.4	61.5	313.51	320.16	185.77	2.47	2.47
DSG-49	B412.5	63.0	63.1	387.5	398.06	234.05	2.55	2.55
Average		**68 m**		**219 g**			**2.42**	**2.4**

Figure 4-31: Bulk density 2.13 (true density 2.25) V_2O_5% 0.5 and carbon 22.7 % DSG9

Figure 4-31 shows a core sample that has a relatively low density, as a result of having a high content of amorphous carbon at nearly 23 % of the total.

Figure 4-32: Bulk density 2.46, V_2O_5% 0.75 and carbon 23.0 % (no true density) DSG15

Figure 4-32 shows a core sample with a sharp contact of highly friable black carbonaceous rock with more normal siliceous vanadium ore.

Figure 4-33: Bulk density 2.57 (true density 2.57) V_2O_5% 0.2 and no carbon analysis (in FW waste zone) DSG19

The core sample in Figure 4-33 shows an equal result for both the bulk and true density. The rock is extremely solid with little porosity and this is a typical rock sample from the siliceous layer which lies stratigraphically below the vanadium stratum.

Scatterplots were prepared to see if statistically the carbon levels have an influence on rock density. With the bulk density (Figure 4-34) there is a slight propensity for higher carbon content to have a lower density, but with true density (Figure 4-35) there is a sharper correlation between high carbon and lower density, which suggests that the rock porosity has quite an important influence on the density rather than the carbon content. To put this into another context, V_2O_5 levels were compared with the carbon content, and here there is a distinct propensity for high grade V_2O_5 to be positively correlated with high carbon content (see Figure 4-36 and Figure 4-37).

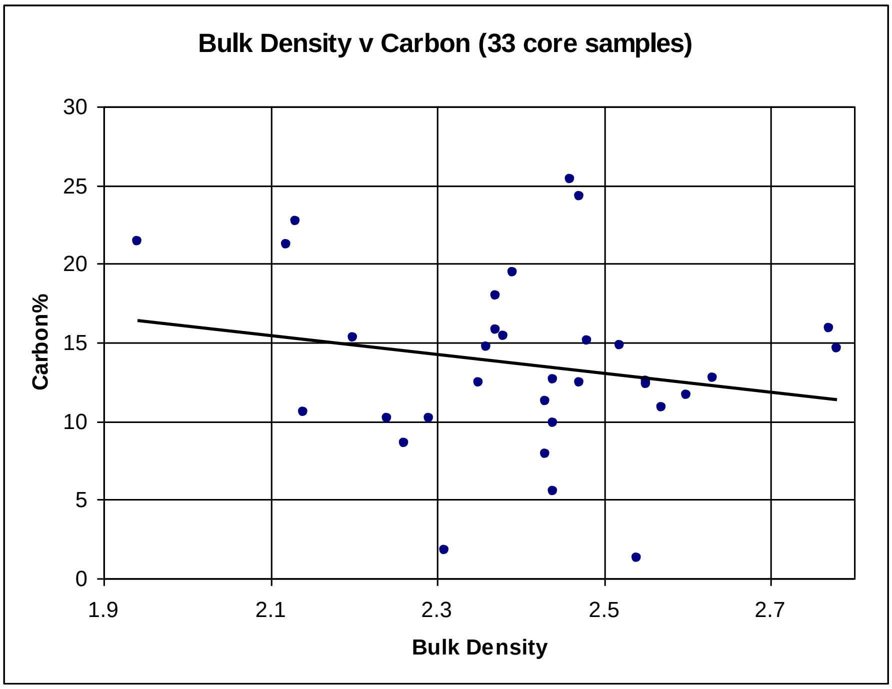


Figure 4-34: Bulk Density versus Carbon Content - Scatterplot

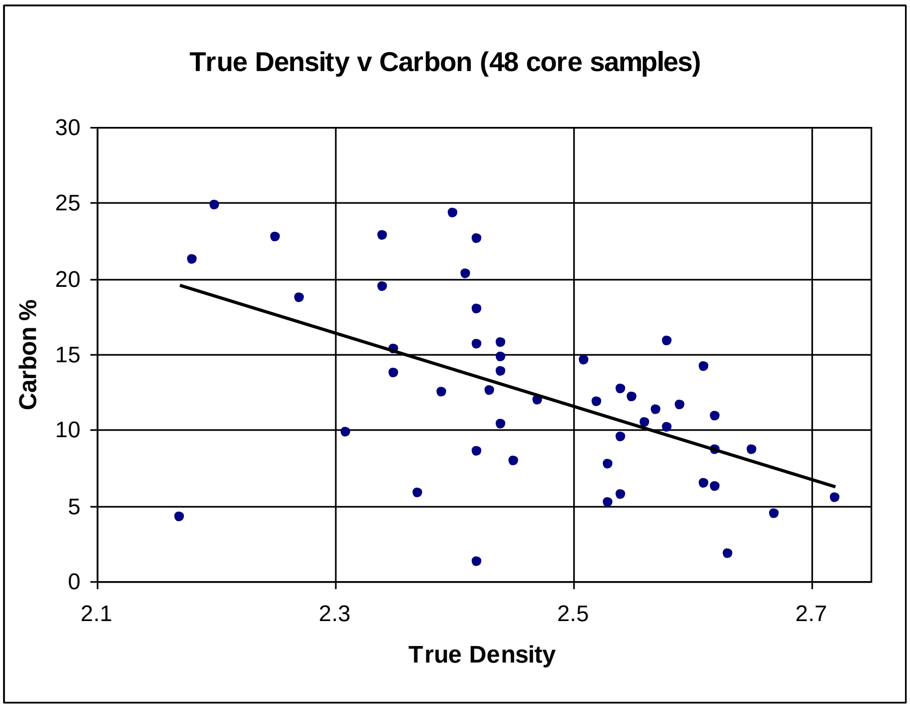


Figure 4-35: True Density Mean versus Carbon Content

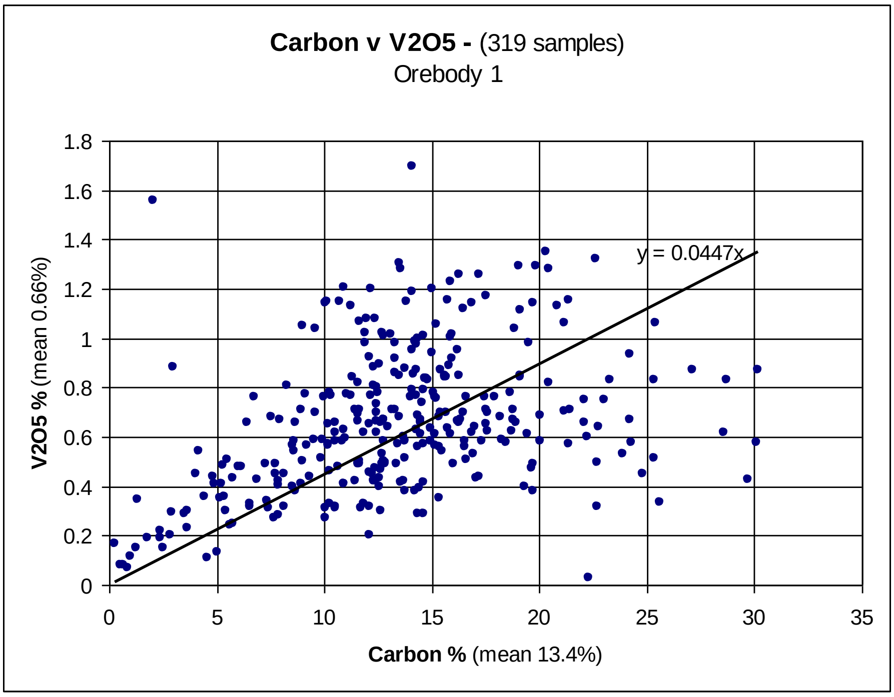


Figure 4-36: Carbon Content versus V_2O_5 – Correlation Coefficient 0.38

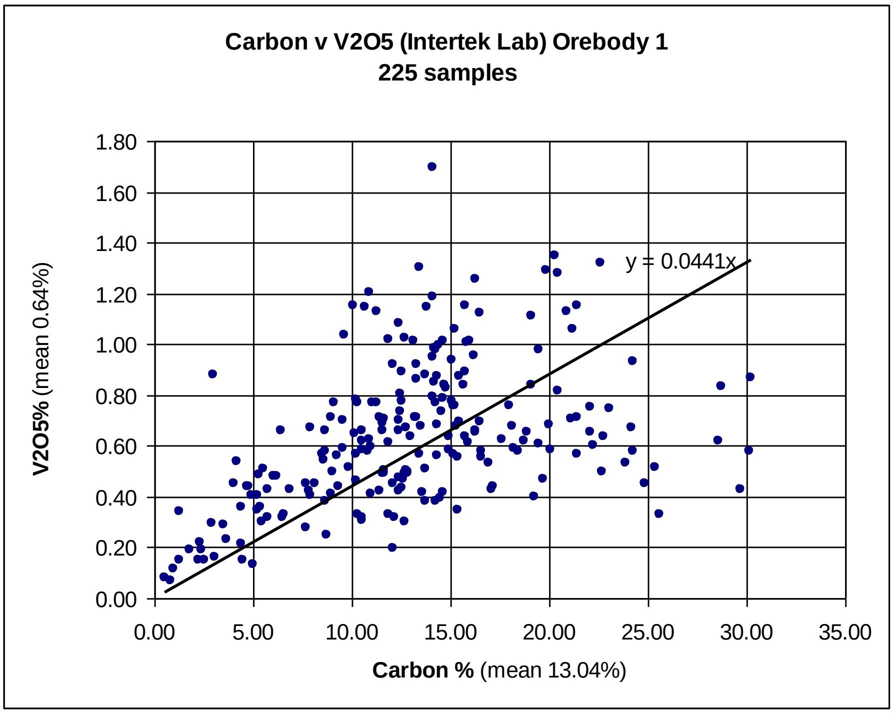


Figure 4-37: Carbon content versus V_2O_5 with only Intertek lab results

4.6.8 DRILLHOLE INCLINOMETRY

The drilling contractor for the 2010-2011 FAR exploration programme was responsible for surveying down-the-hole to determine the azimuth and angle from the vertical (zenith). A Russian built MIG-24 (gyro inclinometer) was used, but it is understood this measurement was undertaken as a separate exercise after the drillhole was completed. The result sheets only show the zenith measurements, at intervals of about 30 m. There appears to be very little deviation of the drillholes. The drillhole azimuth is only recorded on the geological drillhole logs, which is probably the azimuth direction from FAR's surveyor. During a site visit, the author was assured that these measurements were accurate and, from their experience, these drillholes will stay straight with the type of drilling equipment being used. Note: most of the deeper holes were drilled vertically, to target the bottom of the fold at depth, and their zenith angles were mostly zero.

4.6.9 DRILLHOLE GEOLOGICAL LOGS

The historical drillhole logs (passports) from the 1970s are extremely comprehensive, and include gamma log results and V_2O_5 sample grades through the orebody. GMR used these to confirm that the sample grades in the database are correct.

FAR's drillhole logs, include code abbreviations to record the geological features of the core, and also record each sample number with depth interval. A graphical log is also produced, showing rock-type symbols. These logs are stored in a spreadsheet format.

4.7 SAMPLE DATABASE

4.7.1 REVIEW OF HISTORICAL DATA

Two major episodes of mineral exploration, 1942-1947 (oxide surface layer only) and 1971-1972 (targeting primary mineralisation at depth), resulted in milestone reserve estimations and the data generated from these exploration periods was extracted from archived documentation with the objective of assessing the validity of the mineral resource. In support of this objective, exploratory data analysis processes on a range of data types, allow observations, with realistic conclusions about the historical results, with special emphasis on the use of statistical tools to evaluate the character of the vanadium grade distributions.

4.7.2 DATA PROVENANCE

FAR retrieved historical data records from archived sources in Kazakhstan on a range of information dating from the 1940s and 1970s. The documentation included copies of maps, exploration mine plans and various reports on sampling results, drillhole logs and feasibility studies with its associated graphical materials. According to priorities, documents were transcribed to computer storage and selected reports translated into English, especially the major feasibility reports for 1947 and 1973. This also included some digital scanning and the generation of CAD drawings of the associated graphical materials.

The final database comprised of a reasonably large percentage of the aggregated historical sample records, from surface trench channels, pits, underground exploration and diamond drilling, and these were imported into DM for evaluation. This review work included 3D terrain modelling from digital topographic point data, the 3D generation of ore body contact limits at the topographic surface from scanned historical geological maps, plus calculating the spatial coordinates of the individual samples from both the trench and drillhole records.

4.7.3 DATA AUDIT

Computerised records of various attributes from resource tables to drillhole geology logs and assays, were compared with a random selection of scans from the historical documents and the level of transcription errors were assessed to be low. Confidence in the correct replication of the data in the computerised format, used to evaluate the deposit, is therefore acceptable and, where anomalies were highlighted in the database records, as determined from statistical and interactive graphic interrogation, corrections were made accordingly. Apparent problems in some of the spatial positions of historical trenches and drillhole collars were resolved to a reasonable level of accuracy, but some unresolved anomalies remain with an estimated maximum of ± 25 m from the true location: these problems appear to be related to the identification of the correct historical drillhole collar locations during FAR's topographic survey of the property: see Section 4.6.1.

4.7.4 TRENCH SAMPLE DATA 1947

The trench sample summary in provided in Table 4-37 for the main orebodies OB1 to OB4, expresses an intensive exploration evaluation of the oxide mineralisation during the 1940s and the OB1 results are consistent with vanadium grades from FAR's pilot plant open pit mining operation. The U_3O_8 results show relatively higher correlation coefficient values than the vanadium and this erratic uranium distribution is a common feature at this deposit, and was also a feature within the primary layer. Note that during the former Soviet-era, only down-the-hole gamma logging was employed to measure the radiation levels within the primary ores.

Table 4-37: Summary Statistics for Surface Trench Data – 1947 (Mean Length 0.5 m)

Ore Body	Grade	Mean	Min	Max	Variance	SD	CV	# SAMPS
1	V_2O_5%	0.89	0	3.02	0.23	0.48	0.54	830
1	U_3O_8%	0.01	0	0.44	0.0005	0.022	1.69	830
2	V_2O_5%	1.1	0	4.4	0.32	0.57	0.52	1,088
2	U_3O_8%	0.02	0	0.38	0.0003	0.017	1.13	1,090
3	V_2O_5%	1.02	0	4.25	0.27	0.52	0.51	828
3	U_3O_8%	0.02	0	0.1	0.0002	0.013	0.81	829
4	V_2O_5%	1.07	0.07	3.16	0.29	0.54	0.51	349
4	U_3O_8%	0.01	0	0.39	0.0003	0.016	1.14	351
All	V_2O_5%	1.02	0	4.4	0.29	0.53	0.52	3,218
All	U_3O_8%	0.01	0	0.44	0.0003	0.018	1.29	3,223

4.7.5 DRILL SAMPLE DATA 1972-1973

In contrast to the trench channel samples in Section 4.7.4 above, the 1972-1973 drill core sample data are specifically related to intersections through the vanadium horizon within the primary mineralised zone, situated below the zone of oxidation. The top of the primary zone is not necessarily at a constant depth-level from the surface, but will reflect local geological characteristics of the formations. From the historical documentation, there appears to be no definitive consensus on the definition of oxide and primary ores, and no doubt there will be areas where there are well-developed intermediate transition zones between oxide and primary mineralisation. The core drilling was restricted to evaluating the vanadium and siliceous horizons in the synclinal limbs of OB1, OB2 and OB3, totalling seven, four and two drillholes respectively. The mean vanadium results for OB1 and

OB2 is 0.66 % V_2O_5 with OB3 slightly lower at 0.61 %, though this has been adjusted by GMR (see Table 4-44 and associated footnote comment). These primary grade results contrast with the higher trench grades found at surface. However, the core recovery is low with a weighted average recovery at only 52 %, so there could be doubt as to the quality of these samples to properly reflect the contained vanadium grades. This is therefore a critical problem in understanding the validity of the reported vanadium levels within the primary mineralised layer. Correlation coefficients for drillhole number and core recovery revealed a surprising but definite correlation trend, where there is a propensity for the higher drillhole numbers (i.e. later holes) to have higher core recoveries (see Figure 4-38). There is no such correlation between drillhole number and V_2O_5% grade or between V_2O_5% grade and core recovery. The validity of these relationships was also checked, using scatterplots, for distribution patterns that can give misleading coefficient results. It shows therefore, with better drilling techniques and experience, core recoveries gradually improved to 90 %+ for a number of later drillhole sample intersections during the 1971-72 drilling programme. Table 4-38 summaries the average weighted core recovery and V_2O_5 values in these drillholes.

According to the Komarnitski report in 1973, an in-depth study of the core loss was undertaken to understand its effect on the validity of the sample results. This included artificial core loss experiments from drilled core where the recovery was in excess of 90 %. The conclusion was that from laboratory tests, *the core recovery does not make any considerable difference to the measured contents of the principal components. However, further studies need to be continued.* The above statistical review of the data does indeed indicate that there is no obvious bias in the results, but the quality of the historical sampling, sample preparation and analyses needs to be supported by due diligence sample testing of the primary mineralisation. On a more qualitative note, the lowest mean core recovery occurs for Drillhole ID 7 at only 29 %, and yet the measured V_2O_5% grade is one of the highest at 0.71 %, and this high grade has been corroborated by a nearby FAR drillhole (profile 3), at a distance of only 25 m.

Table 4-38: Drillholes with % Core Recoveries and V_2O_5 Grades - 1973

Drillhole ID	Core recovery %	Mean V_2O_5%	OB #	Comment
1	42	0.65	1	
2	32	0.66	1	
3	43	0.61	1	
4	67	0.66	2	
5	47	0.61	1	
6	72	0.52	?	between 2 & 3
7	29	0.71	1	
8	57	0.51	2	
9	56	0.69	2	
11	55	0.74	1	

Drillhole ID	Core recovery %	Mean V_2O_5%	OB #	Comment
12	89	0.70	2	
13	68	0.66	3	
14	87	0.71	1	

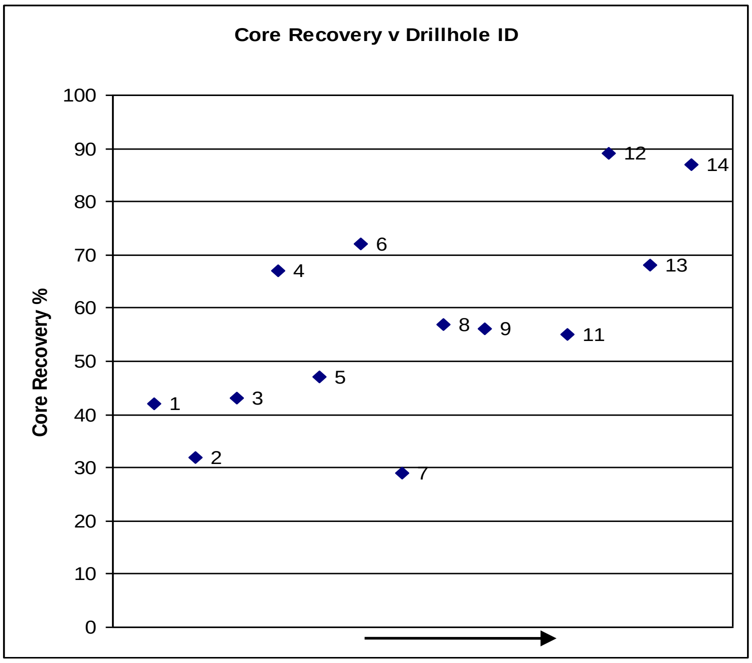


Figure 4-38: Scatterplot - core recovery trends with drillhole ID

The carbon statistics appear to be on total carbon which would include total organic carbon (TOC) including some inorganic graphite, and for all orebodies this is about 14 %, as based on the 1973 core data for the primary zone. In the oxide environment, TOC will tend to be depleted, but can be preserved locally where silicified. The concentration of TOC can therefore be an indicator on the degree of oxidation and reaches average maximum levels in the primary zone. In the 1947 Ankinovich report there was no indication of analysing for organic carbon but the reported total carbon values show an average grade of 2.69 % in the oxide and only 4.4 % in the primary ore. During the main leaching phase for the sample analytical process, high temperatures appeared to have caused combustion of the organic carbon, yet these values are not consistent with expected TOC levels for the oxide and primary zones. However, sometime after the 1947 reserves were published, Ankinovich re-investigated the TOC in old exploration samples and discovered that the TOC can reach levels of 25 %, and he confirmed that the early work was not correct.

Table 4-39 shows consistent V_2O_5 grade levels between the different ore bodies and supports the common acknowledgement from the various episodes of exploration that the grades are uniform within the vanadium layer and this is also supported as a common feature with other such vanadium deposits worldwide. In stark contrast with the trench sample results, the average organic carbon values are high at 14 %, vis-à-vis <1 % in the oxide zone, and this level of contained carbon is indicative of the primary mineralisation zone. The high coefficient of variation values of MGO and, sometimes for the P_2O_5, are caused by high erratic values and this has been reported as being due to the irregular distribution in the form of cluster concentrations within sub-layers.

4.7.6 SAMPLE DATABASE – FAR

A summary of the core drilling sample database is presented in Table 4-39. A number of different laboratories were used for analyses, and many individual samples, using pulp or coarse reject sample splits, were assayed by more than one laboratory, and this was extremely useful for QA/QC assessment. The different laboratories have been identified by a suffix in the in the database as follows:

- AU = Australia
- KA = Karaganda
- BAX = Balausa XRF
- BA = Balausa (this was the old FAR laboratory and now superseded by the Balausa XRF laboratory).
- KO = Kozlov

Note: as the BA and KO laboratory results were unacceptable, these were not included in the summary statistics, and for the OB1 JORC-based grade interpolations, only the analytical results from the AU, KA and BAX laboratories were utilized.

Table 4-39: FAR Drilling Sample Analysis Summary Stats – OB1 to OB3

OB	Grade% _ Laboratory	Mean	MIN	Max	Variance	SD	CV	# Samps
1	V_2O_5_AU	0.52	0.02	1.70	0.11	0.33	0.62	306
1	V_2O_5_KA	0.48	0.06	1.36	0.09	0.30	0.63	109
1	V_2O_5_BAX	0.66	0.05	2.86	0.18	0.42	0.64	97
1	C_AU	10.51	0.08	30.19	45.03	6.71	0.64	306
1	U_AU	0.0078	0.0008	0.5476	0.0012	0.0342	4.37	306
1	MO_AU	0.0165	0.0002	0.0747	0.0001	0.0104	0.63	306
1	Total Records 1,267							
2	V_2O_5_AU							0
2	V_2O_5_KA	0.45	0.06	1.04	0.09	0.30	0.68	21
2	V_2O_5_BAX	0.38	0.08	1.86	0.10	0.32	0.83	150
2	C_AU							0
2	U_AU							0
2	MO_AU							0
2	Total Records 476							
3	V_2O_5_AU	0.41	0.06	1.33	0.08	0.29	0.70	57
3	V_2O_5_KA	0.48	0.06	1.60	0.11	0.33	0.69	115
3	V_2O_5_BAX	0.12	0.12	0.12				1
3	C_AU	7.15	2.98	15.61	12.08	3.48	0.49	57
3	U_AU	0.0049	0.0011	0.0191	0.0000	0.0034	0.70	57
3	MO_AU	0.0101	0.0012	0.0370	0.0001	0.0075	0.75	57
3	Total Records 224							
All	V_2O_5_AU	0.51	0.02	1.70	0.10	0.32	0.64	363
All	V_2O_5_KA	0.47	0.06	1.60	0.10	0.31	0.66	245
All	V_2O_5_BAX	0.49	0.05	2.86	0.15	0.39	0.78	248
All	C_AU	10.01	0.08	30.19	41.56	6.45	0.64	363
All	U_AU	0.0074	0.0008	0.5476	0.0010	0.0316	4.27	363
All	MO_AU	0.0155	0.0002	0.0747	0.0001	0.0103	0.66	363
All	Total Records 1,967							

4.8 GEOLOGICAL INTERPRETATION AND MODELLING

4.8.1 OVERVIEW

The target mineralisation for modelling was characterised by the vanadium enriched layer, forming conformal contacts with both the hangingwall and footwall formations. Beyond these lithological contacts, the V_2O_5 levels dropped sharply to <0.2 % V_2O_5, and so it has been relatively straightforward to define the orebody contacts for the 3D modelling. All fundamental interpretations from these contacts were undertaken at pre-defined cross-strike profile sections (locations for the exploration drilling) and, on these profiles, contact strings were defined interactively. These profile strings were used as templates for interpreting the contacts at intermediate distances along strike and here the vanadium layer geometry is modified according to structural trends: see Figure 4-39. Both the exploration sections and the intermediate pseudo-sections, formed the basis for constructing a solid 3D vanadium model of the primary mineralisation.

4.8.2 DETAILS

Fundamentally, the exploration drillholes are located approximately at the Soviet-era defined exploration profiles: profiles 0 to 5, at a strike spacing of 670 m between 4 and 5 to 900 m between profiles 3 and 4, plus an additional two intermediate profiles, defined by FAR at profiles 2.5 and 3.5. Surface expression of the mineralisation was defined by trench exposures taken at intervals of 25 m to 200 m along both limbs of the syncline - in effect, the trenches proved the continuity of the mineralisation along the whole strike of the orebody.

The geological interpretation of the primary zone was based on drillcore sample analyses of the vanadium content and aided by lithological features; from both the historical (1973) and FAR’s drillhole exploration programme. Typically, the orebody’s hangingwall and footwall were seen to form conformable contacts with adjacent formational layers, and their precise intersection location is based on assay values from core samples plus support from apparent visible mineralogical changes along the core. See Figure 4-39 below.

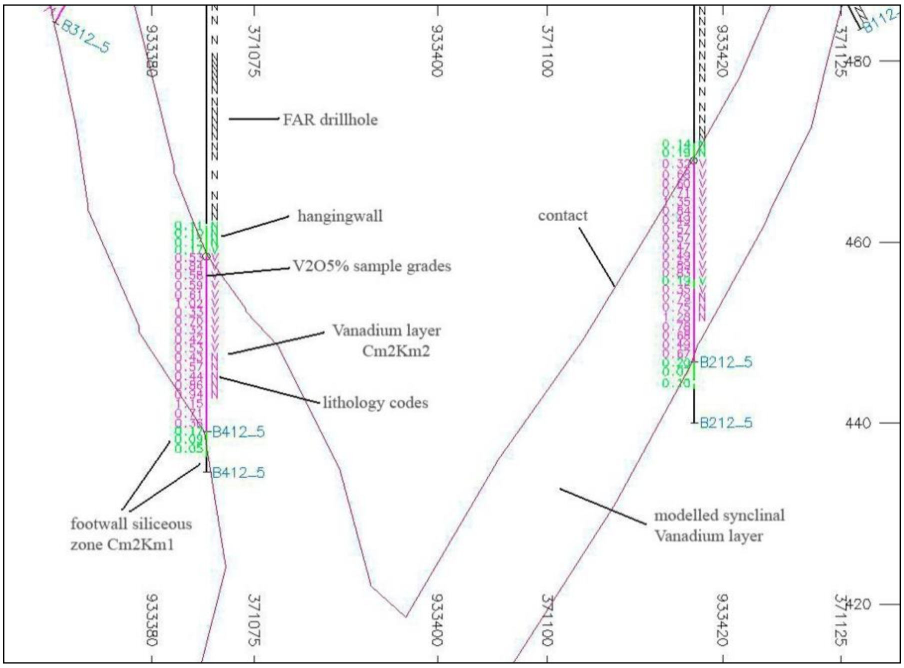


Figure 4-39: Profile 2.5 Showing Interpretation & Modelling Details of Orebody Syncline -OB1

The projection of the intersecting drillhole contacts, between adjacent drillholes on each of their respective exploration profiles, was aided by intersecting angles of the sedimentary micro-layering to the core axis, plus downward projections from surface trench orebody intersections. The interpretation of these contacts between adjacent drillholes on each profile was undertaken by 3D interactive digitising of the contacts, forming linking strings between drillholes.

Because of the relatively wide distances between exploration profiles, it was necessary to define orebody geometric trends along strike, through the construction of pseudo intermediate sections at 50 m intervals, to mid-way between adjacent exploration sections. This was achieved by first projecting the interpreted exploration section strings 50 m along strike, interactively modifying the geometry according to strike trend changes at that point, and then projecting this modified pseudo-section 50 m, editing these strings at this new pseudo-section and repeating the process accordingly. Note: in order to properly express geometric strike trend changes, such as fold amplitude and orebody thickness etc., cognisance was taken of topographic changes, surface expression of the orebody (defined by trenches) and longitudinal fold axial plunge angle trends between exploration profiles. These final OB1 section strings were tagged (red strings) to their respective adjacent profile strings for controlling wireframe modelling: wireframe model – see Figure 4-40.

The basic modelling approach was undertaken in DM, in which the geological contacts are interpreted interactively by digitising contact strings on cross-strike section profiles, at regular strike intervals. See Figure 4-40 and Figure 4-41. Interpretation control was based on actual spatial contacts, as defined by drillhole core contacts, mostly from FAR's exploration drilling and minor Soviet-era drilling and, orebody surface expression of the mineralisation, as defined from historical sampling of the mineralisation via trenches. Fundamentally, the drillholes are located at the Soviet-era defined exploration profiles: profiles 0 to 5, at a strike spacing of 670 m between 4 and 5 to 900 m between profiles 3 and 4, plus an additional two intermediate profiles, defined by FAR at profiles 2.5 and 3.5. Surface expression of the mineralisation were defined by trench exposures taken at intervals of 25 m to 200 m along both limbs of the syncline - in effect, the trenches proved the continuity of the mineralisation along the whole strike of the orebody.

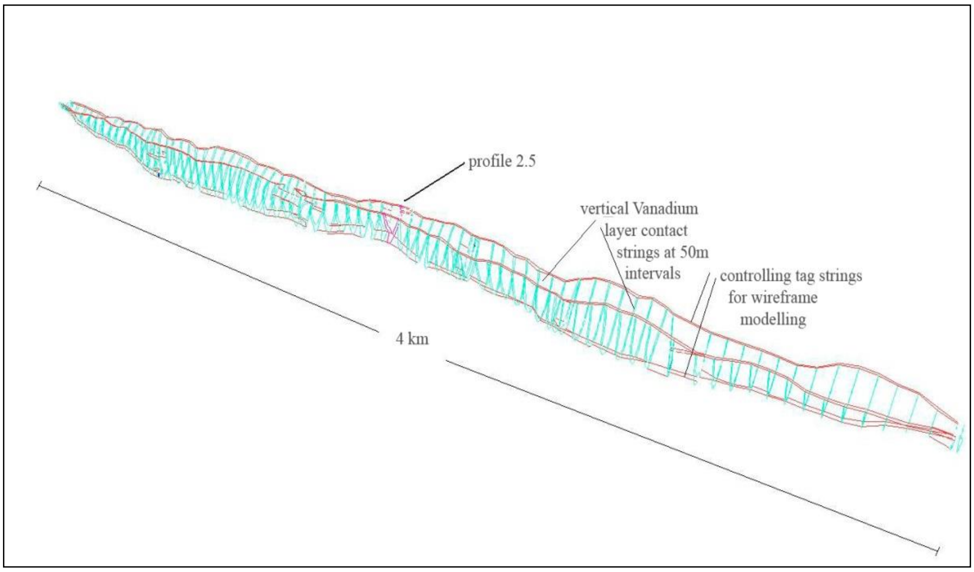


Figure 4-40: Perspective view looking north – showing string elements

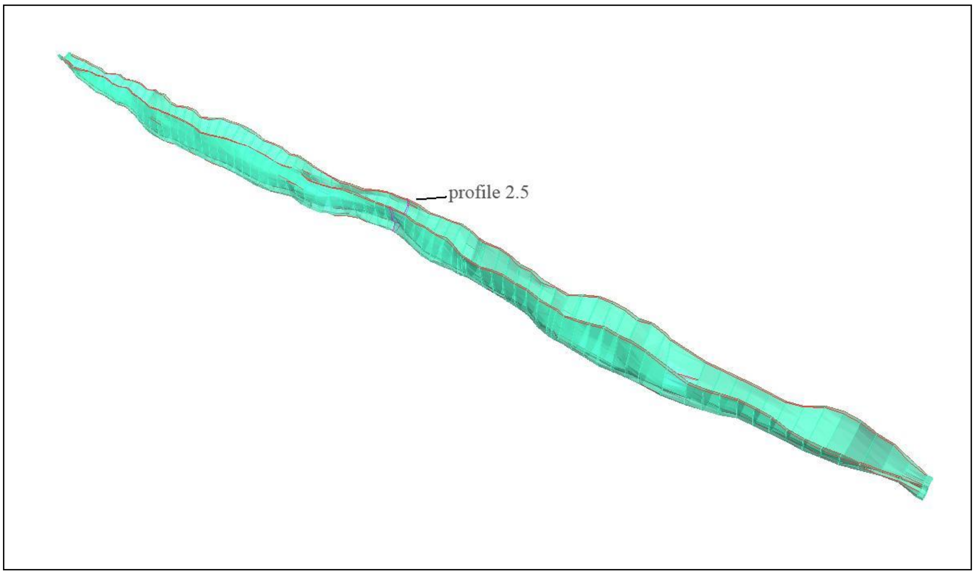


Figure 4-41: Perspective view showing completed solid wireframe model – OB1

Note: the intermediate profiles drilled by FAR, confirmed both the expected geometry of the folded vanadium layer and the level of V_2O_5 grades.

4.9 EXPLORATORY DATA ANALYSIS

In statistics, exploratory data analysis is a numerical and graphical examination of data sets to summarise their main characteristics. Here, the exploratory data analysis focuses on the sample data, with special emphasis on OB1 data used for grade interpolation.

Table 4-40: Summary Analysis Stats - Surface Trench Samples – 1947

Ore-Body	Grade	Mean	Min	Max	Variance	SD	CV	# SAMPS	# Trenches
1	V_2O_5%	0.89	0.00	3.02	0.23	0.48	0.54	830	39
	U_3O_8%	0.01	0.00	0.44	0.0005	0.022	1.69	830	
2	V_2O_5%	1.1	0.00	4.4	0.32	0.57	0.52	1088	79
	U_3O_8%	0.02	0.00	0.38	0.0003	0.017	1.13	1090	
3	V_2O_5%	1.02	0.00	4.25	0.27	0.52	0.51	828	47
	U_3O_8%	0.02	0.00	0.1	0.0002	0.013	0.81	829	
4	V_2O_5%	1.07	0.07	3.16	0.29	0.54	0.51	349	20
	U_3O_8%	0.01	0.00	0.39	0.0003	0.016	1.14	351	
5	V_2O_5%	1.03	0.00	2.86	0.29	0.54	0.52	123	9
	U_3O_8%	0.01	0.00	0.32	0.0003	0.017	1.31	123	
All	V_2O_5%	1.02	0.00	4.4	0.29	0.53	0.52	3218	194
	U_3O_8%	0.01	0.00	0.44	0.0003	0.018	1.29	3223	

Average sample length 0.5 m

Summary statistics for the 1940s data were restricted to the oxide surface trench samples with only individual grade results for vanadium and uranium available: see Table 4-40 above. Uranium grades (U_3O_8%) are very low and therefore not economic to mine, but would be produced as a by-product from the vanadium processing. A useful statistic is the coefficient of variation and for U_3O_8 this can reach values greater than one which indicates that erratic highs are present in the sample database. These high values could have a significant impact on the estimation of the resource and it would be necessary to curtail the influence of these values. In contrast, the vanadium (V_2O_5) coefficient of variation does not indicate problems with erratic high values and all ore bodies show a consistent statistic. V_2O_5 mean values for OB2, OB3 and OB4 are about 1 %, with OB1 samples showing a 10 % lower grade at 0.89 %.

4.9.1 PRIMARY ORE DATA 1973 AND CORE LOSS

In contrast to the trench channel samples above, the 1972-1973 sample data are derived from drill core samples which intersected the vanadium layer within the primary mineralised zone, situated below the zone of oxidation: OB1, OB2 and OB3 only. The top of the primary zone is not necessarily at a constant depth-horizon from the surface, but will reflect local geological characteristics of the formations. From the historical documentation, there appears to be no definitive consensus on the definition of oxide and primary ores, and no doubt there will be areas where there are well-developed intermediate transition zones between oxide and primary mineralisation. The core drilling was restricted to evaluating the vanadium and siliceous layers in the synclinal limbs of OB1, OB2 and OB3, totalling seven, four and two drillholes respectively.

The mean vanadium results for OB1 and OB2 is 0.66 % V_2O_5 with OB3 slightly lower at 0.61 % and these results contrast with the higher trench grades at surface. However, the core recovery is low with a weighted average recovery at only 52 %, so there could be doubt as to the quality of these samples to properly reflect the contained vanadium grades. This is therefore a critical problem in understanding the validity of the reported vanadium levels within the primary mineralised layer during the 1970s.

Correlation coefficients for drillhole number and core recovery revealed a surprising but definite correlation trend, where there is a propensity for the higher drillhole numbers (i.e. later holes) to have higher core recoveries. There is no such correlation between drillhole number and V_2O_5% grade or between V_2O_5% grade and core recovery. The validity of these relationships was also checked, using scatterplots, for distribution patterns that can give misleading coefficient results. It shows therefore, with better drilling techniques and experience, core recoveries gradually improved to 90 %+ for a number of later drillhole sample intersections during the 1971-72 drilling programme. Table 4-41 summarises the average weighted core recovery and V_2O_5 values in these drillholes.

According to the Komarnitski report in 1973, an in-depth study of the core loss was undertaken to understand its effect on the validity of the sample results. This included artificial core loss experiments from drilled core where the recovery was in excess of 90 %. The conclusion was that from laboratory tests, *the core recovery does not make any considerable difference to the measured contents of the principal components. However, further studies need to be continued.* The above statistical review of the data does indeed indicate that there is no obvious bias in the results, but the quality of the historical sampling, sample preparation and analyses needs to be supported by due diligence sample testing of the primary mineralisation. On a more qualitative note, the lowest mean core recovery occurs for Drillhole ID 7 at only 29 %, and yet the measured V_2O_5% grade is one of the highest at 0.71 %.

Table 4-41, Table 4-42, Table 4-43, Table 4-44 and Table 4-45 summarise the 1973 drill sample results.

Table 4-41: Drillholes 1973 V_2O_5 grades and OBs

Drillhole ID	Core Recovery %	Mean V_2O_5%	Orebody #
1	42	0.65	1
2	32	0.66	1
3	43	0.61	1
5	47	0.61	1
7	29	0.71	1
11	55	0.74	1
14	87	0.71	1
4	67	0.66	2
8	57	0.51	2
9	56	0.69	2
10	?	0.31	2
12	89	0.70	2
6	72	0.52	3
13	68	0.66	3

Table 4-42: Summary statistics 1973 drillholes OB1 - 106 samples

	Mean	Min	Max	Variance	SD	CV
V_2O_5%	0.66	0.03	1.56	0.08	0.29	0.44
C%	13.88	0.28	27.15	25.59	5.06	0.36
P_2O_5%	0.54	0.11	4.50	0.20	0.45	0.83
TIO_2%	0.18	0.04	0.53	0.01	0.09	0.50
MGO%	1.16	0.16	13.23	5.89	2.43	2.09
FE_2O_3%	4.09	1.25	11.56	3.92	1.98	0.48
SIO_2%	63.41	22.48	89.25	157.70	12.56	0.20
AL_2O_3%	4.39	0.90	11.63	3.64	1.91	0.44

Note: average length of core sample is 1.0 m

Table 4-43: Summary statistics 1973 drillholes OB2 - 83 samples

	Mean	Min	Max	Variance	SD	CV
V_2O_5%	0.66	0.09	1.96	0.12	0.34	0.52
C%	14.02	0.36	31.95	42.46	6.52	0.47
P_2O_5%	0.57	0.09	4.10	0.31	0.55	0.96
TIO_2%	0.17	0.05	0.54	0.01	0.08	0.48
MGO%	1.61	0.17	12.97	6.36	2.52	1.57
FE_2O_3%	3.46	0.24	11.68	4.68	2.16	0.62
SIO_2%	64.24	14.90	91.00	186.60	13.66	0.21
AL_2O_3%	3.80	0.65	14.35	3.61	1.90	0.50

Note: average length of core sample is 1.0 m

Table 4-44: Summary statistics 1973 drillholes OB3 - 52 samples

	Mean	Min	Max	Variance	SD	CV
V_2O_5%	*0.61	0.05	1.46	0.11	0.36	0.59
C%	13.42	0.42	26.05	41.19	6.42	0.48
P_2O_5%	0.66	0.14	6.87	0.61	0.78	1.18
TIO_2%	0.22	0.05	1.00	0.03	0.19	0.86
MGO%	0.20	0.00	0.67	0.04	0.20	1.00
FE_2O_3%	3.73	1.16	8.45	1.99	1.41	0.38
SIO_2%	67.42	30.80	90.13	116.60	10.80	0.16
AL_2O_3%	4.87	1.10	10.52	4.31	2.08	0.43

Note: average length of core sample is 1.0 m

*Note: GMR considers that drillhole #6 does not belong to OB3, as it appears to have intersected a thrusted ore slice, which lies between OB2 and OB3 – so this depressed the *V_2O_5*% grade. Taking the other drillhole #13, as a truer representation for OB3, then 0.66 % should be the mean value and this also supports the grade indications from FAR's drilling results.*

Table 4-45: Summary statistics all 1973 drillholes - 241 samples

	Mean	Min	Max	Variance	SD	CV
V_2O_5%	0.65	0.03	1.96	0.10	0.31	0.48
C%	13.82	0.28	31.95	34.03	5.83	0.42
P_2O_5%	0.57	0.09	6.87	0.32	0.57	1.00
TIO_2%	0.19	0.04	1.00	0.01	0.12	0.63
MGO%	1.04	0.00	13.23	4.62	2.15	2.07
FE_2O_3%	3.82	0.24	11.68	3.80	1.95	0.51
SIO_2%	64.53	14.90	91.00	159.80	12.64	0.20
AL_2O_3%	4.32	0.65	14.35	3.92	1.98	0.46

Note: average length of core sample is 1.0 m

4.9.2 FORMER SOVIET ERA (1947 AND 1973)

The frequency histograms for V_2O_5 (see Figure 4-42) for both oxide and primary mineralisation show typical log normal distributions with a positive skew, and the oxide grades have a greater range spread than the primary V_2O_5. The cumulative log probability plots show similar gradient curves and exhibit bimodality with the curve inflexion at about 0.5 % to 0.6 % V_2O_5, which marks the overlapping point of the two sub-populations. These sub-populations may represent synsedimentary or post-depositional diagenetic changes within the vanadium stratum, and especially related to organic and inorganic forms of the vanadium in the anoxic marine environment. Oxides show obvious enrichment relative to the primary ore, because of “acid leaching” and re-deposition of vanadium to more stable mineral forms.

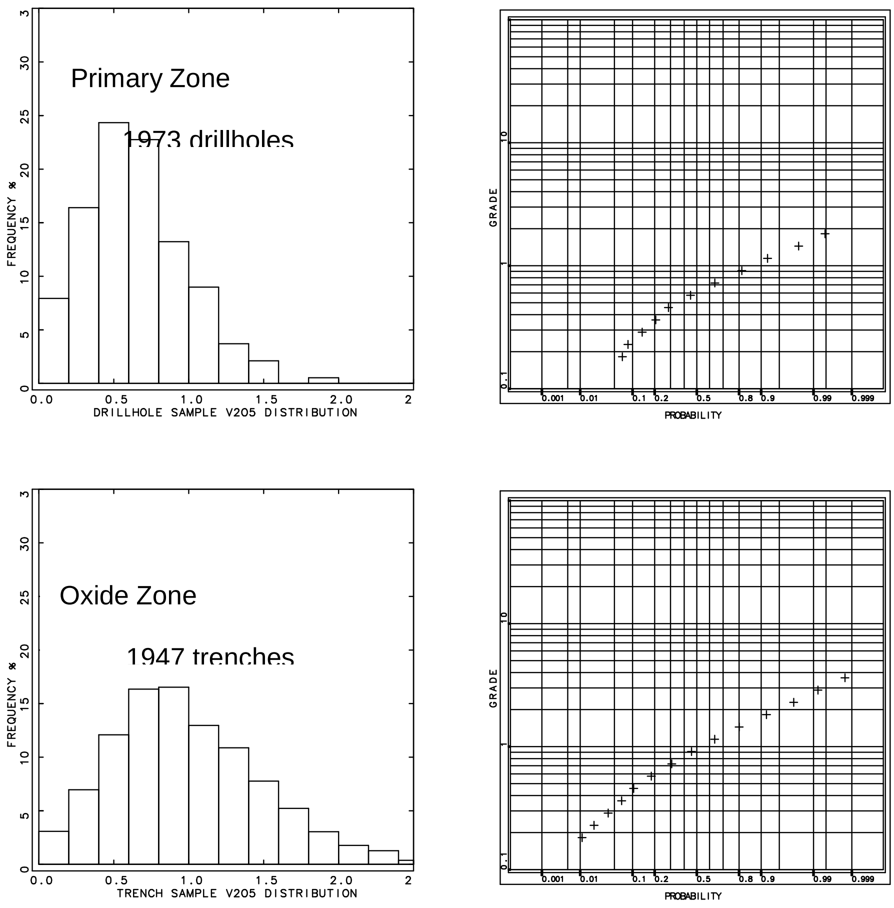


Figure 4-42: V_2O_5 Trench & Drillhole Histograms and Log Probability Plots Soviet-Era

Oxidation due to weathering removed TOC as a rock component, with residual enrichment of V and Mo. The majority of the V (~60 %) and Mo (~85 %) contents in fresh rock are remobilised during oxidation/weathering and redeposited in oxide/oxyhydroxide phases.

A combined cumulative probability plot, Figure 4-43 below, shows that both the oxide and primary V_2O_5 values have almost exactly the same paralleling curve form, with the trench data having consistently higher grades throughout the curve. This curve is therefore a type of distinct fingerprint for the deposit and shows that the vanadium distributions still retain the same grade relationship within both the oxide and primary zone.

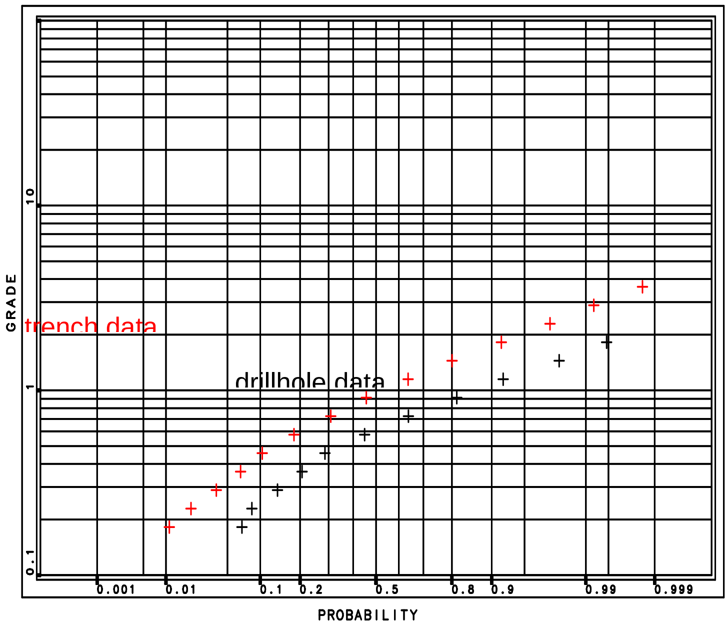


Figure 4-43: Combined probability plot – trench & drillhole V_2O_5% data

To see how the V_2O_5 mineralisation may be variable within the defined 14 sub-units of the vanadium layer, the trench oxide grade logs also included geology and, this was coded, and where possible mean grade statistics were derived for the sub-layers. Table 4-46 lists the sub-units in stratigraphic order from lowest to highest. Although there is variability of V_2O_5 within the sub-layers, this is within a fairly narrow grade band ranging from 0.77 % in the dolomite to 1.25 % within the argillaceous ore. It was not possible to determine the statistics for a number of the sub-units, as the geology description was sometimes ambiguous or sometimes the grade was related to a combination of more than one layer, resulting in an insignificant number of samples for statistical evaluation.

Table 4-46: 1947 V_2O_5 Oxide Trench Results Split on Sub-Units within Vanadium Layer

Interlayer Number	Name	Mean V_2O_5%	Number of Samples
1	Lower Ore	0.81	154
2	Shaly Ore	0.94	193
3	Under Phosphoritic Ore	1.10	240
4	Phosphoritic Shale	Unclear from log description	
5	Phosphoritic Ore		
6	Lower Agillaceous Interlayer		
7	Under Dolomite Ore	1.17	258
8	Dolomite	0.77	135
9	Above Dolomite Shale	1.06	210
10	Above Dolomite Ore	0.85	673
11	Middle Agillaceous Interlayer	Unclear from log description	
12	Argillaceous Ore	1.25	378
13	Upper Agillaceous Interlayer	Unclear from log description	
14	Shaly Ore Member	0.98	673

Quantile analysis is a useful tool to see if too small a number of high grade samples contain a disproportionately large percentage of the contained metals within the sample database. Comparing the decile intervals between the 1973 and 1947 sample data, the percentage V_2O_5 metal levels are very similar within each decile, though the mean weighted V_2O_5 grades are consistently higher within the oxide samples (see Table 4-47 and Table 4-48). The results show that the metal content for each decile and, the top decile percentile splits, are at acceptable levels and there is no need to top-cut the higher grade samples for resource grade estimations. The percentage metal variations also give support to the consistent nature of the grade distributions, within both the oxide and primary zones.

Table 4-47: Quantile Table of 1973 Drillhole Data from the Primary Vanadium Layer OB1

Q%_from	Q%_to	Nsamples	Mean	Min	Max	Metal	Metal%
0	10	14	0.16	0.03	0.28	1.82	2.29
10	20	13	0.34	0.29	0.41	4.38	5.52
20	30	10	0.48	0.42	0.51	5.34	6.73
30	40	8	0.57	0.53	0.60	5.67	7.13
40	50	6	0.61	0.61	0.63	8.91	11.22
50	60	9	0.66	0.64	0.67	8.42	10.59
60	70	11	0.73	0.67	0.76	7.92	9.97
70	80	13	0.82	0.77	0.85	10.99	13.84
80	90	8	0.94	0.87	1.05	10.65	13.40

Q%_from	Q%_to	Nsamples	Mean	Min	Max	Metal	Metal%
90	100	14	1.19	1.06	1.56	15.35	19.32
91	92	2	1.06	1.06	1.07	2.50	3.15
92	93	1	1.08	1.08	1.08	1.19	1.50
93	94	2	1.14	1.14	1.14	1.48	1.87
94	95	1	1.14	1.14	1.14	0.97	1.22
95	96	1	1.17	1.17	1.17	1.05	1.33
96	97	1	1.20	1.20	1.20	1.74	2.19
97	98	2	1.21	1.20	1.23	2.54	3.19
98	99	2	1.27	1.26	1.28	1.65	2.08
99	100	2	1.44	1.29	1.56	2.23	2.81
0	100	106	0.66	0.03	1.56	79.46	100.00

Table 4-48: Quantile Table of 1947 Surface Trench Data from the Oxide Vanadium Layer

Q%_from	Q%_to	Nsamples	Mean	Min	Max	Metal	Metal%
0	10	342	0.26	0.00	0.43	43.21	2.54
10	20	324	0.52	0.44	0.60	86.77	5.10
20	30	298	0.65	0.60	0.70	108.26	6.36
30	40	309	0.76	0.70	0.82	127.80	7.50
40	50	311	0.88	0.82	0.94	147.29	8.65
50	60	302	1.00	0.94	1.06	166.94	9.80
60	70	319	1.14	1.06	1.22	190.60	11.19
70	80	331	1.32	1.22	1.40	219.57	12.89
80	90	331	1.54	1.40	1.68	258.44	15.18
90	100	351	2.12	1.68	4.40	354.02	20.79
90	91	35	1.71	1.68	1.74	27.60	1.62
91	92	34	1.77	1.74	1.80	30.01	1.76
92	93	37	1.81	1.80	1.85	30.41	1.79
93	94	33	1.88	1.86	1.92	30.49	1.79
94	95	35	1.94	1.92	1.96	33.26	1.95
95	96	32	2.02	1.96	2.08	34.15	2.01
96	97	36	2.14	2.08	2.20	35.91	2.11
97	98	37	2.27	2.20	2.36	38.16	2.24
98	99	36	2.51	2.36	2.74	41.62	2.44
99	100	36	3.11	2.77	4.40	52.40	3.08
0	100	3218	1.02	0.00	4.40	1702.90	100.00

4.9.3 DATA USED FOR GRADE ESTIMATION – OB1

Table 4-49 and Table 4-50 detail the data used for grade estimation in OB1.

Table 4-49: OB1 Composite Data Statistics Split on Laboratory Prior to Compositing

Laboratory	Field %	Number of Samples	Mean	From	To	Variance	SD	CV	% of Samples
AU	V_2O_5	217	0.66	0.07	1.70	0.08	0.28	0.43	52.80
KA	V_2O_5	15	0.71	0.40	1.07	0.04	0.20	0.28	3.65
BAX	V_2O_5	77	0.78	0.08	2.86	0.16	0.39	0.51	18.73
SO	V_2O_5	102	0.67	0.03	1.56	0.08	0.28	0.41	24.82
ALL	V_2O_5	411	0.68	0.03	2.86	0.09	0.30	0.44	100.00

Update 17 December 2013

Table 4-50: FAR Composite Data Stats Split on Laboratory – 2m Lengths

Laboratory	Field %	Number of Samples	Mean	From	To	Variance	SD	CV	% of Samples
AU	V_2O_5	118	0.66	0.08	1.29	0.05	0.22	0.34	53.15
KA	V_2O_5	7	0.71	0.60	1.03	0.02	0.14	0.20	3.15
BAX	V_2O_5	39	0.78	0.28	1.36	0.07	0.27	0.35	17.57
SO	V_2O_5	58	0.67	0.23	1.03	0.03	0.17	0.25	26.13
ALL	V_2O_5	222	0.68	0.08	1.36	0.05	0.22	0.32	100.00

NOTE: there are 16 samples with both KA and BAX analyses, the KA mean is 0.68 % and the BAX average is 0.73 %, which is higher by 7.4 %, which is considered to be within acceptable limits, though it does exemplify the higher grade tendency for the BAX results. For the other BAX results, there is only the unreliable KO or BA analytical results for comparison.

4.9.4 ADDITION OF BY-PRODUCTS OB1

By-products C, MoO_3 and U_3O_8 and REE are important revenue contributors to the project. The ore also contains 335 ppm of REE and although no value is currently being ascribed, FAR expects to produce this by-product in future. For REE, there is a need to first determine which REE minerals are in the vanadium layer and then make comprehensive analyses of these REEs accordingly, before global estimates can be produced. Table 4-51 shows summary data for V_2O_5, C, MoO_3 and U_3O_8.

Table 4-51: Summary Data

Field %	Number of Samples	Mean	From	To	Variance	SD	CV
V_2O_5	411	0.68	0.03	2.86	0.09	0.30	0.44
MoO_3*	217	0.028	0.000	0.062	0.000	0.013	0.46
U_3O_8*	217	0.012	0.001	0.646	0.002	0.047	4.12
C	319	13.49	0.28	30.19	31.09	5.58	0.41

**Assays from Australia only*

Table 4-52: Summary Statistics after Top-Cutting U_3O_8 (Prior to Compositing)

Field %	Number of Samples	Mean	From	To	Variance	SD	CV
V_2O_5	411	0.68	0.03	2.86	0.091884	0.303124	0.44
MoO_3*	217	0.028	0.0003	0.062	0.000	0.013	0.46
U_3O_8*	217	0.008	0.001	0.1	0.000	0.009	1.13
C	319	13.49	0.28	30.19	31.08909	5.575759	0.41

**Assays from Australia only (Update 17 December 2013) Note: U_3O_8 top-cut to 0.1*

Quantile analysis was used to assess the U_3O_8, as the coefficient of variation was very high and this statistic indicated that there were some anomalously high grade samples which caused this high ratio. Results of the quantile analysis suggested that top-cutting to 0.1 % was appropriate and Table 4-52 shows an acceptable coefficient of variation ratio. It also resulted in a change in the correlation coefficient from random to a weak correlation trend with V_2O_5.

For geostatistical applications, samples are composited to equal lengths and the summary statistics of these composited samples are shown in Table 4-53. Note for details on compositing the samples refer to Section 4.9.5.

Table 4-53: Summary data after compositing

Field %	Number of Samples	Mean	From	To	Variance	SD	CV
V_2O_5	222	0.68	0.08	1.36	0.05	0.22	0.32
MoO_3*	118	0.028	0.000	0.049	0.000	0.010	0.37
U_3O_8*	118	0.008	0.002	0.052	0.000	0.006	0.79
C	176	13.48	0.81	26.80	20.36	4.51	0.33

**Assays from Australia only (Update 17 December 2013)*

4.9.5 COMPOSITING

Within the vanadium layer, the target core size was NQ (43 mm) and for geostatistical interpolation, equal support is required, and this is normally achieved by compositing the samples into equal lengths. The mean sample length for all input samples is 1.09 m with a wide range from 0.25 m to 5.3 m in length (see Table 4-54) and for compositing a target length of 2 m was considered to be suitable.

Table 4-54: Length Statistics for Uncomposited and Composited Samples

Orebody	N Samples	Mean	From	To	Range
1 (uncomps)	434	1.09	0.25	5.3	5.05m
1 (comps)	222	2.01	1.15	2.9	1.15m

Statistically, the composited samples at ±10 % of the 2.01 m mean, range from 1.81 to 2.21 m and this comprises >99 % of the total samples and therefore is acceptable for grade interpolation. Note: for the uncomposited samples, only 56 % of the samples are within ±10 % of the 1.09 m mean, and would not have been acceptable for geostatistical estimations. See Figure 4-44 and Figure 4-45 for an illustration of the sample length distribution uncomposited and composited samples respectively. Figure 4-46 and Figure 4-47 show histograms and log probability plots for the uncomposited and composited samples respectively.

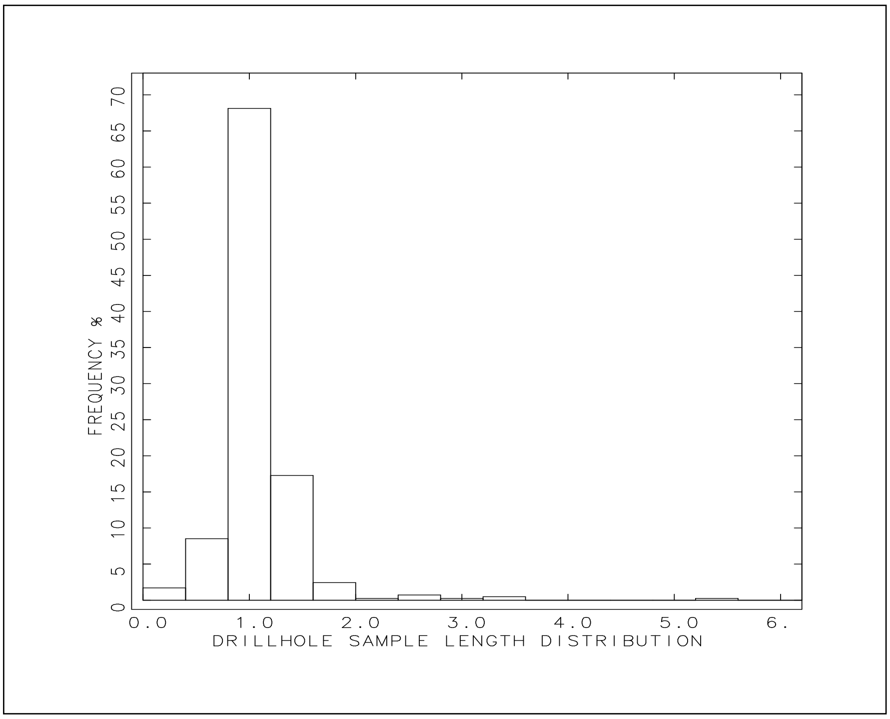


Figure 4-44: OB1 Uncomposited Sample Lengths

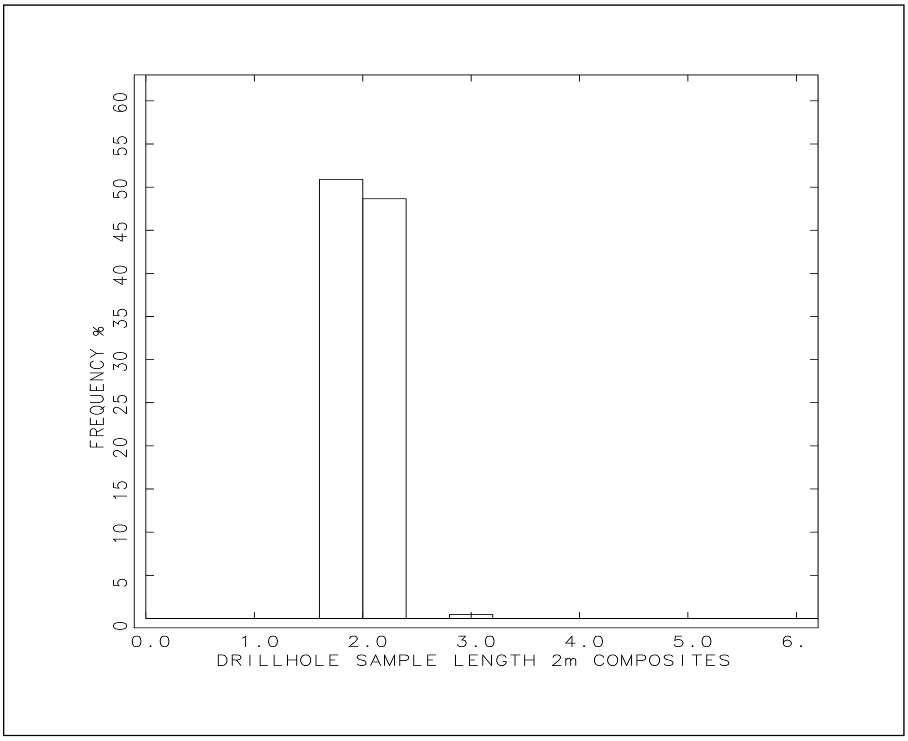


Figure 4-45: OB1 Composited Sample Lengths

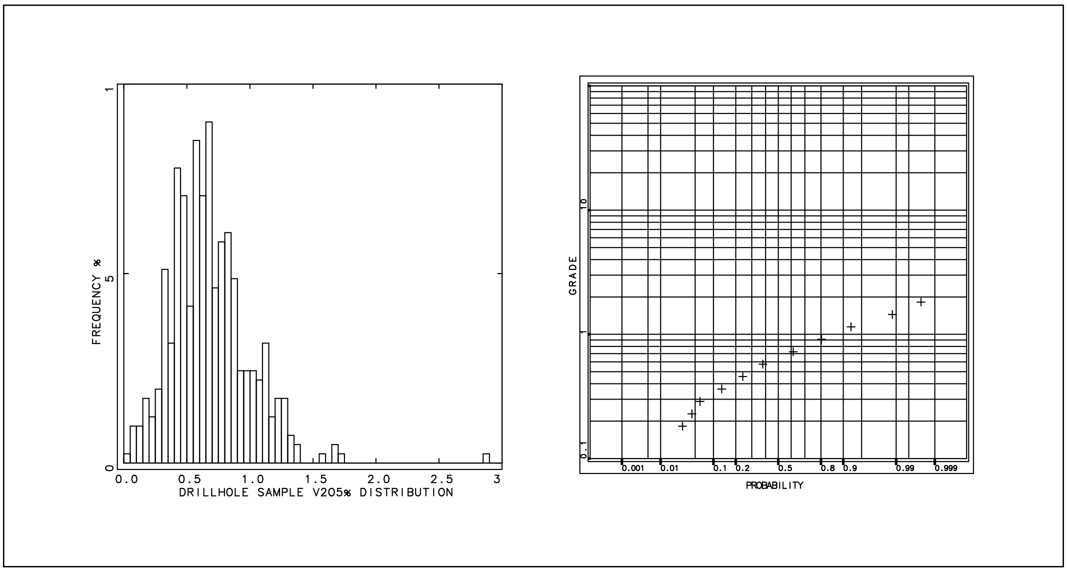


Figure 4-46: OB1 V_2O_5 Histogram & Log Probability Plots Uncomposited Samples

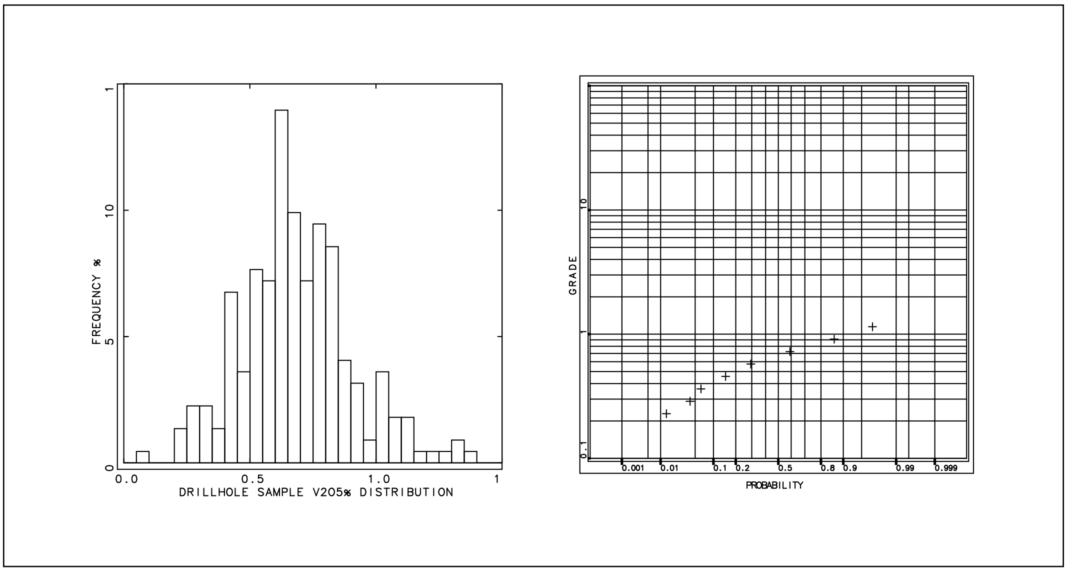


Figure 4-47: OB1 V_2O_5 Histogram & Log Probability Plots 2m Composites

4.9.6 BY-PRODUCTS (UNCOMPOSITED)

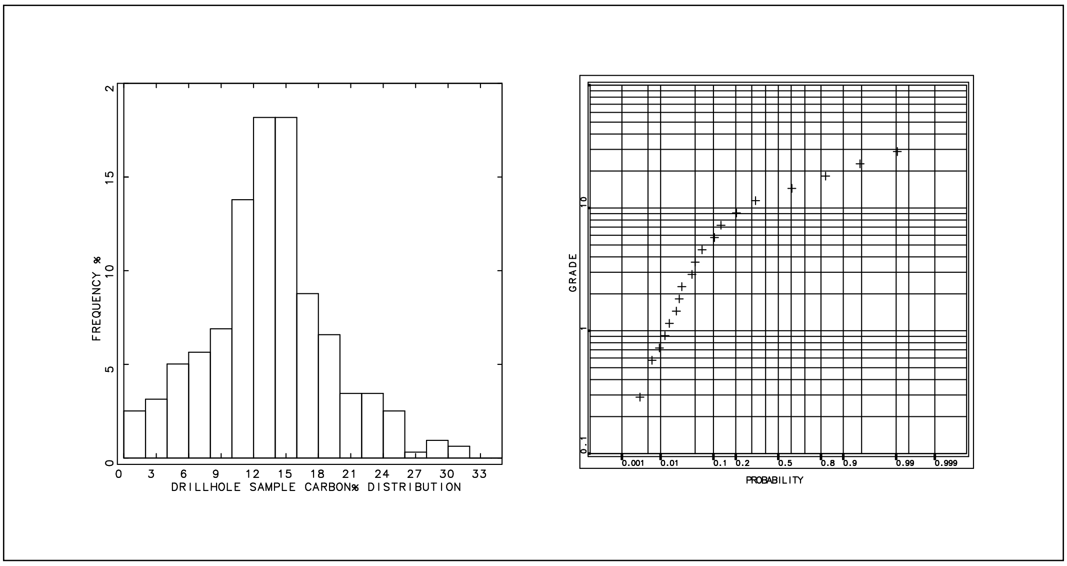


Figure 4-48: Carbon Histogram and Log Probability Plots OB1

Figure 4-48 contains a carbon probability plot showing a distinct bimodal distribution. The lower population (about 10 % of samples) probably represents migration and loss of carbon during diagenesis and low grade metamorphism. Figure 4-49 shows MoO_3 and Figure 4-50 shows U_3O_8 probability plot data.

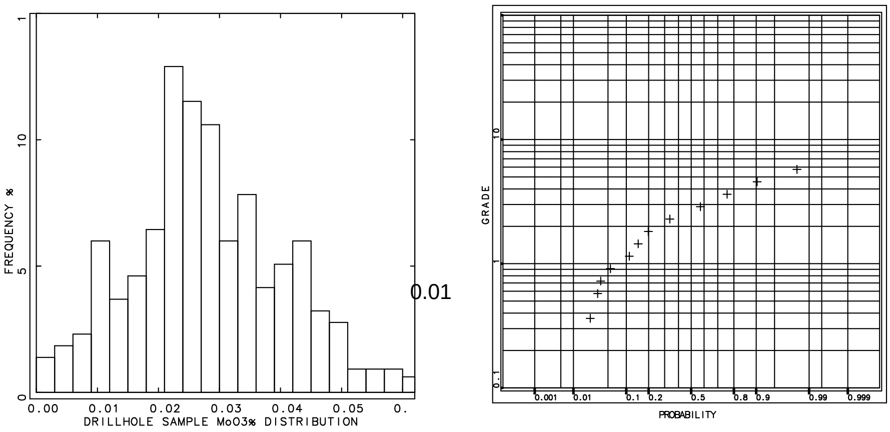


Figure 4-49: MoO_3 Histogram & Log Probability Plots OB1

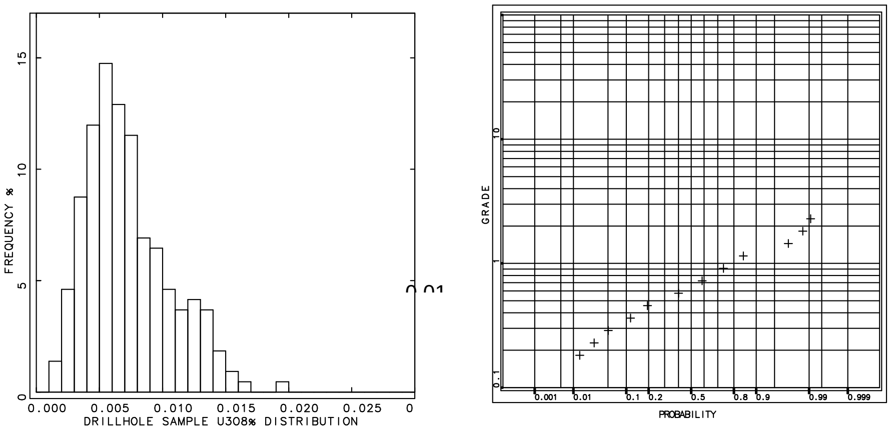
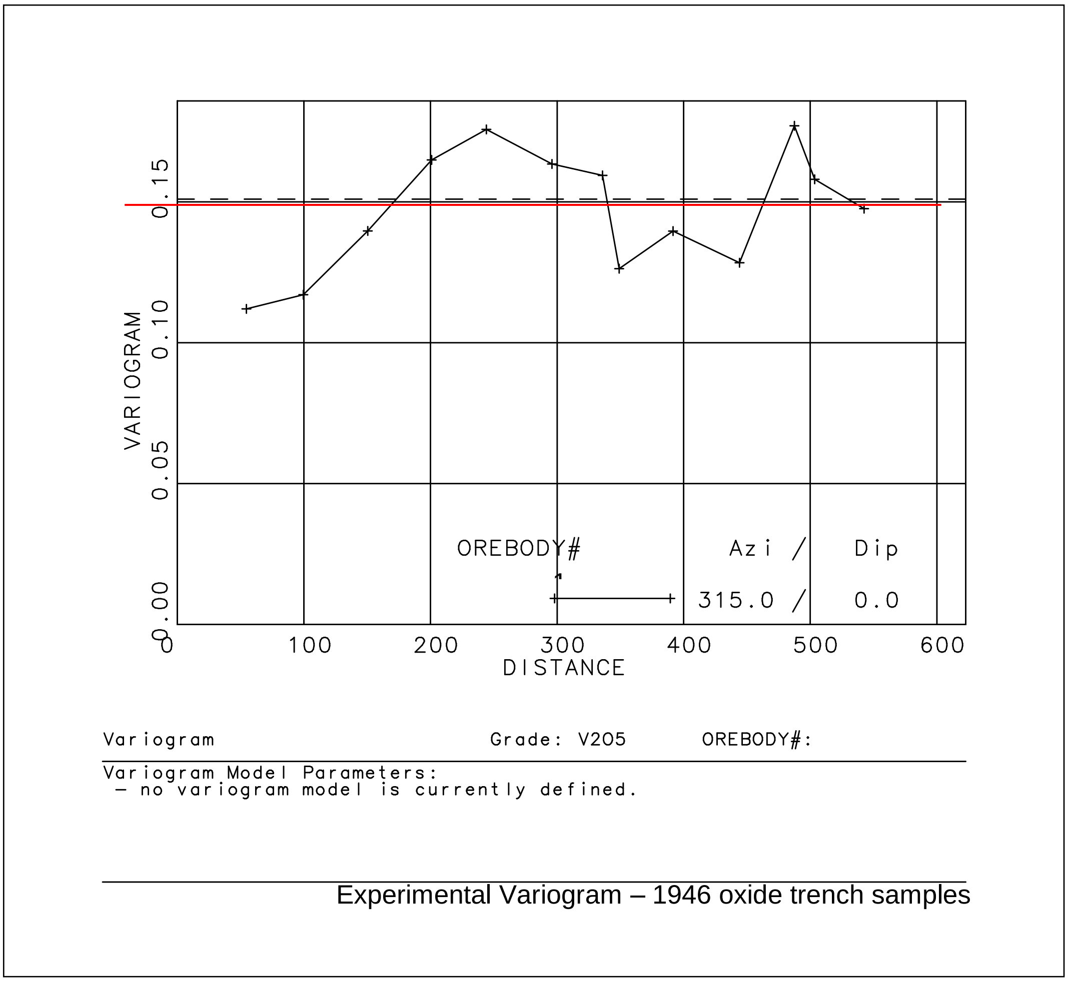


Figure 4-50: U_3O_8 Histogram & Log Probability Plots (top-cut <0.1) n=217

4.10 VARIOGRAPHY

4.10.1 VARIOGRAPHY OXIDE

In understanding the grade variability along strike, experimental variograms were generated from the oxide sample trench-grade results, as an aid to help plan the drillhole spacing for the exploration programme. The 1946 oxide surface trench samples averaged 0.5 m in length and to ensure uniform support (length) the samples were composited to 1 m lengths. It was found from the limited data available, that the OB1 trench samples provided sufficient density coverage to generate acceptable experimental variograms along strike.



Variogram Model Parameters:
– no variogram model is currently defined.

Experimental Variogram – 1946 oxide trench samples

Figure 4-51: Experimental variogram for oxide trench data – OB1

Figure 4-51 shows the experimental variogram for the oxide zone, using 1 m composites (average input sample length 0.5 m), totalling 419 samples with a mean of 0.88 % V_2O_5. It was constructed from a basic lag of 60 m and the resulting range of influence is about 200 m along the strike at 315°.

The nugget effect appears quite high relative to the sill, and may account for up to half of the variability range, but this would need to be confirmed from variogram modelling. It does however indicate that the historical 50 m trench spacing along strike was probably overzealous.

4.10.2 VARIOGRAPHY FOR OB1 RESOURCE ESTIMATION

Because OB1 is a tight synclinal fold structure, it was necessary to split the drillhole sample data into their respective fold limbs, designated as the SW and NE limbs, prior to generating experimental variograms. Numerous 3D experimental variograms, along variable strike directions and dip, showed that a very strong anisotropy was apparent, with the strike range direction, parallel with the fold axis, being several times greater than the downdip range direction. It was discovered that by removing the drillhole sample grades at the noses of the syncline, a clearer downdip range structure was apparent, as the localised “nose” mineralisation could not appropriately represent the general grade distribution down dip.

Although along the strike of the fold, there are numerous localised complexities in the fold geometry, affecting both strike and dip directions, only typified variogram structures were modelled and these formed the basis for V_2O_5 grade estimations for each limb structure. Additionally, it was also possible to model a separate variogram for the carbon grades within the NE limb, and the range structure closely mirrored the vanadium model. However, for the SW limb there were insufficient carbon grade analyses from the drillcore samples to generate useable experimental variograms. This also applies to both U_3O_8 and MoO_3, where unfortunately it has not been possible to produce acceptable variograms for either the SW or NE limb.

The results of the variogram modelling (see Table 4-55, Table 4-56, Table 4-57, Figure 4-52, Figure 4-53, and Figure 4-54) reflect the geological environment of deposition, within a relatively shallow marine basin (graben), in which strong mineralogical distribution trends follow the “shoreline” strike and where the graben slope directions showing much more rapid mineralogical changes. It is notable that the SW limb demonstrates longer variogram range structures, especially downdip, and it is suggested that this is an artefact which has been induced by the Karatau thrust fault (40 Ma), so that the relatively nearer NE limb shows greater and abrupt geometric complexity. Both limbs however do exhibit variable changes in strike and dip and the final variogram models presented here were used as templates for generating local variogram models during grade estimation procedures.

Table 4-55: V_2O_5 NE limb variogram parameters

Limb	Single Anisotropic Spherical Structure	Nugget Variance C_0	Spatial Variance C_1	Range SW-NE Dip 300 Azimuth 41^0 (Z)	Down-dip Azimuth 221^0 (X)		Strike direction Dip = 0^0 (Y)	
					Dip	Range	Range	Azimuth
NE	1	0.0055	0.0491	15 m	600	75 m	550 m	3110

The nugget effect is typically only 10 % of the variogram sill, which reflects the high level of consistency of the samples.

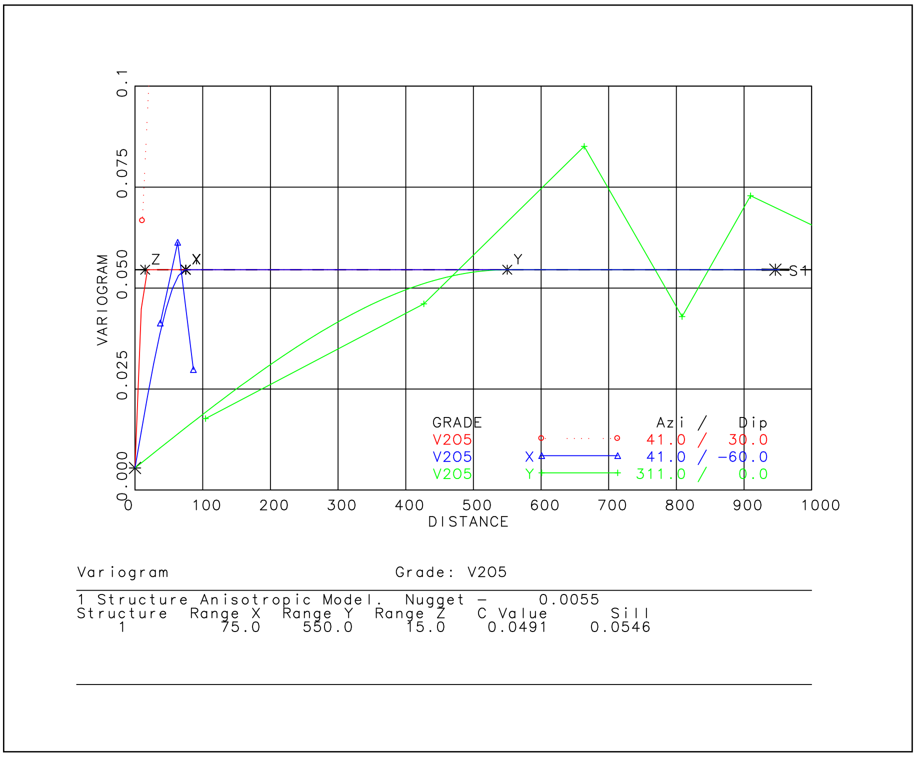
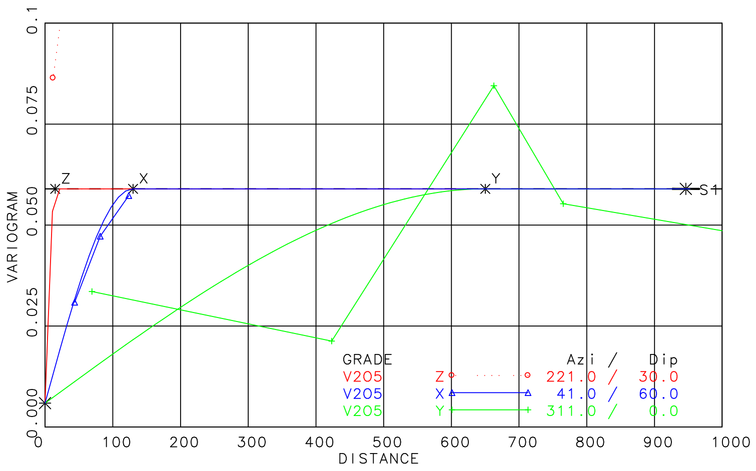


Figure 4-52: NE limb variograms – basic models

Table 4-56: V_2O_5 SW limb variogram parameters

Limb	Single Anisotropic Spherical Structure	Nugget Variance C_0	Spatial Variance C_1	Range SW-NE Dip 300 Azimuth 41^0 (Z)	Down-dip Azimuth 221^0 (X)		Strike direction Dip = 0^0 (Y)	
					Dip	Range	Range	Azimuth
SW	1	0.0059	0.0531	15 m	700	130 m	650 m	3110



```
Variogram                          Grade: V205
1 Structure Anisotropic Model.  Nugget -     0.0059
Structure  Range X  Range Y  Range Z   C Value      Sill
    1        130.0    650.0     15.0    0.0531    0.0590
```

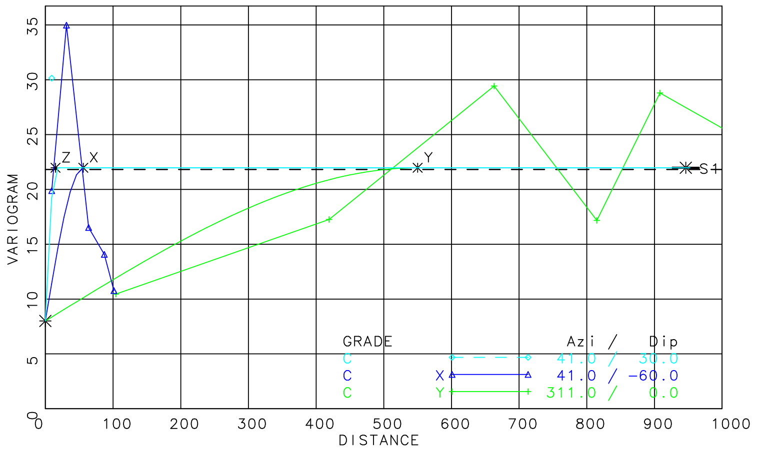
Figure 4-53: SW limb variograms for V_2O_5

Note: this basic model's (Figure 4-53) ranges were used as the search ellipsoid for IPD^2 carbon interpolations (SW limb only), and also for MoO_3 and U_3O_8 grade interpolations.

Table 4-57: Carbon NE limb variogram parameters

Limb	Single Anisotropic Spherical Structure	Nugget Variance C_0	Spatial Variance C_1	Range SW-NE Dip 300 Azimuth 41^0 (Z)	Down-dip Azimuth 221^0 (X)		Strike direction Dip = 0^0 (Y)	
					Dip	Range	Range	Azimuth
NE	1	8.0	14.0	15 m	600	56 m	550 m	3110

Carbon nugget is 36 % of the sill, indicating a much more erratic distribution than V_2O_5 grades, even though there is a notable correlation between the two elements.



Variogram Grade: C

1 Structure Anisotropic Model. Nugget - 8.0

Structure	Range X	Range Y	Range Z	C Value	Sill
1	56.0	550.0	15.0	14.0	22.0

Figure 4-54: Carbon variograms – NE limb only

Note: this basic model's ranges (Figure 4-54) were used as the search ellipsoid for MoO_3 and U_3O_8 grade interpolations using IPD^2`.

4.11 GEOLOGICAL BLOCK MODELLING

Testing of the most appropriate block size for allowing sufficient sensitivity, yet keeping the number of blocks to a low optimal level resulted in a fundamental block size of 40 m x 20 m x 20 m (X, Y & Z directions) and these parent blocks were split at the contacts of the orezone wireframe model, according to splitting criteria, resulting in a block dimension of 5 m along the X (easting) direction, 2.5 m along Y (northing) direction and along the Z (elevation) direction the blocks were split exactly at the wireframe contact, resulting in a minimal block dimension of 0.1 m to a maximum dimension of 20 m (mean length of 10.33 m). No subzone identifiers were required for this block model, though later surface oxide blocks would be removed to form a final primary orebody model. Due to the somewhat uniform grade distributions of the commercial products, these variable block sizes are not expected to induce grade biases. Details of the OB1 block model are provided in Table 4-58, Table 4-59 and Table 4-60.

Table 4-58: Spatial definition of OB1 block model (m) (oxide and primary)

Axes	X	Y	Z
Model Origin	369450	931960	290
Max. No. of Basic Blocks	77	139	18
Basic Block Size	40	20	20
Dimension	3080	2780	360
Max. Co-ordinates	372530	934740	650

Table 4-59: Actual dimensions of OB1 block model (m) – oxide

Direction	Coordinate		Range [m]
	Min	Max	
X (east)	369502.5	372467.5	2965
Y (north)	932003.75	934688.75	2685
Z (elevation)	449.4	598.35	148.95
Total # blocks = 6578			

NE limb split into 50,383 blocks and SW limb 32,079

Table 4-60: Actual dimensions of OB1 block model (m) – primary

Direction	Coordinate		Range [m]
	Min	Max	
X (east)	369497.50	372472.50	2975
Y (north)	932006.25	934686.25	2680
Z (elevation)	328.20	586.55	258.35
Total # blocks = 71598			

NE limb split into 50,383 blocks and SW limb 32,079 blocks

4.11.1 SPLITTING OB1 BLOCKS INTO SEPARATE SW AND NW LIMB MODELS

To better control the estimation of grades into the block model, it was necessary to split the fundamental block model into its SW and NE limb components and, to control this splitting, a subvertical wireframe surface was generated and used to extract the blocks to their respective structural limb components (see Figure 4-55). After grade interpolations these blocks were recombined into the basic model again.

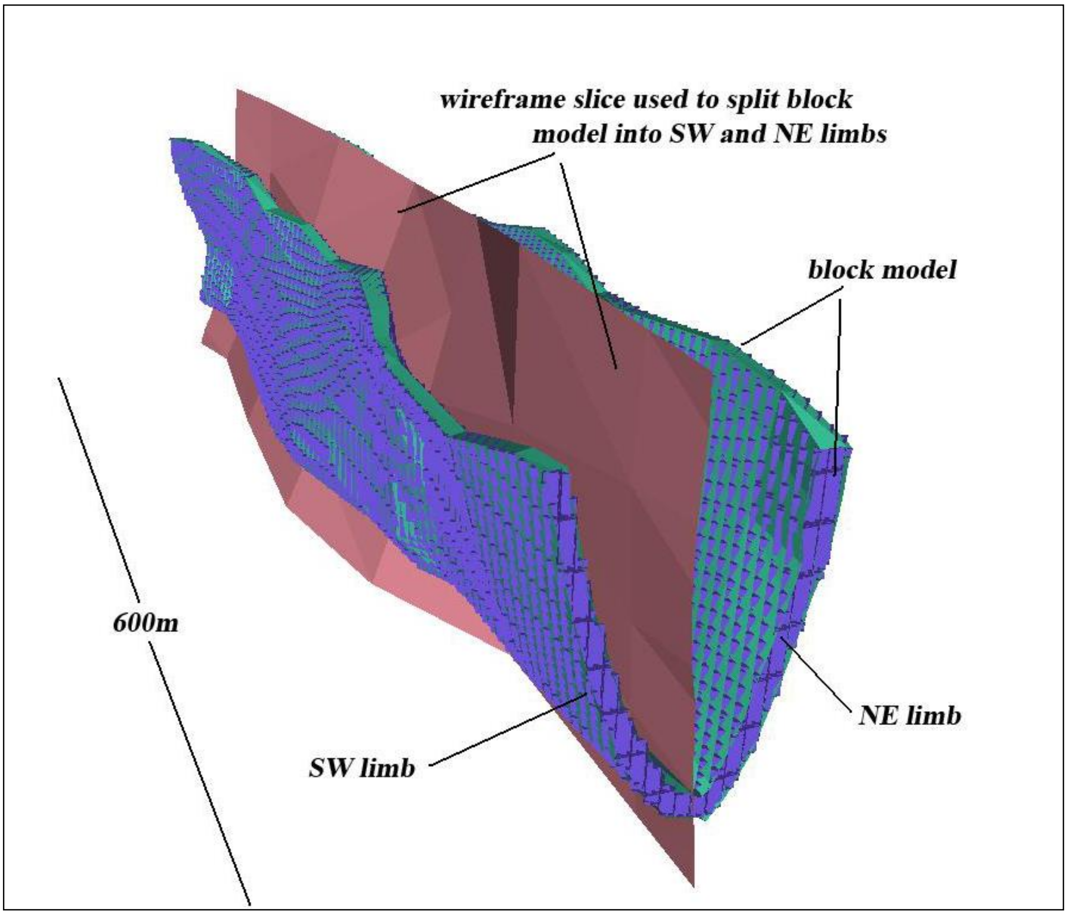


Figure 4-55: Perspective view along part of OB1 showing wireframe slice used for splitting block model into separate limbs for grade estimation

4.11.2 EXTRACTING THE PRIMARY AND OXIDE ORE BLOCKS

As the fundamental basic block model also includes the surface oxide layers, these oxide blocks need to be removed from the model in generating the primary blocks. The depth of oxidation from surface is about 10 m, though locally this can be erratic, and this notional 10 m layer is removed from the block model as follows:

The topographic wireframe surface is dropped 10 m vertically and this new oxide/primary interface surface is used to control the extraction of primary mineralised blocks below this interface and the extraction of oxide blocks are controlled by the topographic surface and the oxide/primary interface, whereby the intervening blocks are removed to form the oxide sub-model.

4.12 GRADE ESTIMATION PRIMARY ZONE OB1

Ordinary kriging (OK) was selected as the most appropriate geostatistical approach for estimating the grades into the block model (split into the SW and NE limb submodels) using the modelled variogram parameters as presented in Section 4.10 (see Table 4-55 and Table 4-56). Both V_2O_5 and C were estimated using OK, but for MoO_3 and U_3O_8 only IPD grade interpolations were realistic and IPD for C was required for the SW limb submodel because valid experimental variograms could not be generated due to limited sample assays.

4.12.1 GEOMETRIC COMPLEXITY AND ESTIMATING GRADES

Although basic variogram models were generated, because of the folding complexity due to variable strike and dip directions, it is difficult to represent the continuity of mineralisation using specific 3D search ellipsoids, according to length and orientation of the three axes. Therefore, when estimating the grade values for each cell within the block model, it is very important to get the orientation of the search volume and estimation parameters correctly aligned. To overcome this problem, DM Studio 3 has developed an application called Geodynamic Anisotropy and this was tested by GMR, as it appeared ideal for this folded tabular orebody. The application allows the anisotropy rotation angles, for defining the search volume and variogram models, to be defined for each cell within the model.

Dip and dip directions were generated from the structural wireframe model using the Datamine ANISOANG process, which generates a point file containing the dips and dip directions within the model: dips and dip directions are developed from direction and orientation of the wireframe triangles. The point file is used to define the true dip and dip direction within the block model of the vanadium layer, using IPD with a circular search distance of 25 metres where a maximum of 10 points are used for dip and direction interpolations in each model cell: IMETHOD=8.

The most appropriate representative 3D variogram models were selected for OK and, controlled by local changes in orientation of the model, according to the defined dip and direction of each block, the V_2O_5 grades were interpolated using the dynamic anisotropy process. Figure 4-56 illustrates the orientation of the geological modelling and geostatistical nomenclature.

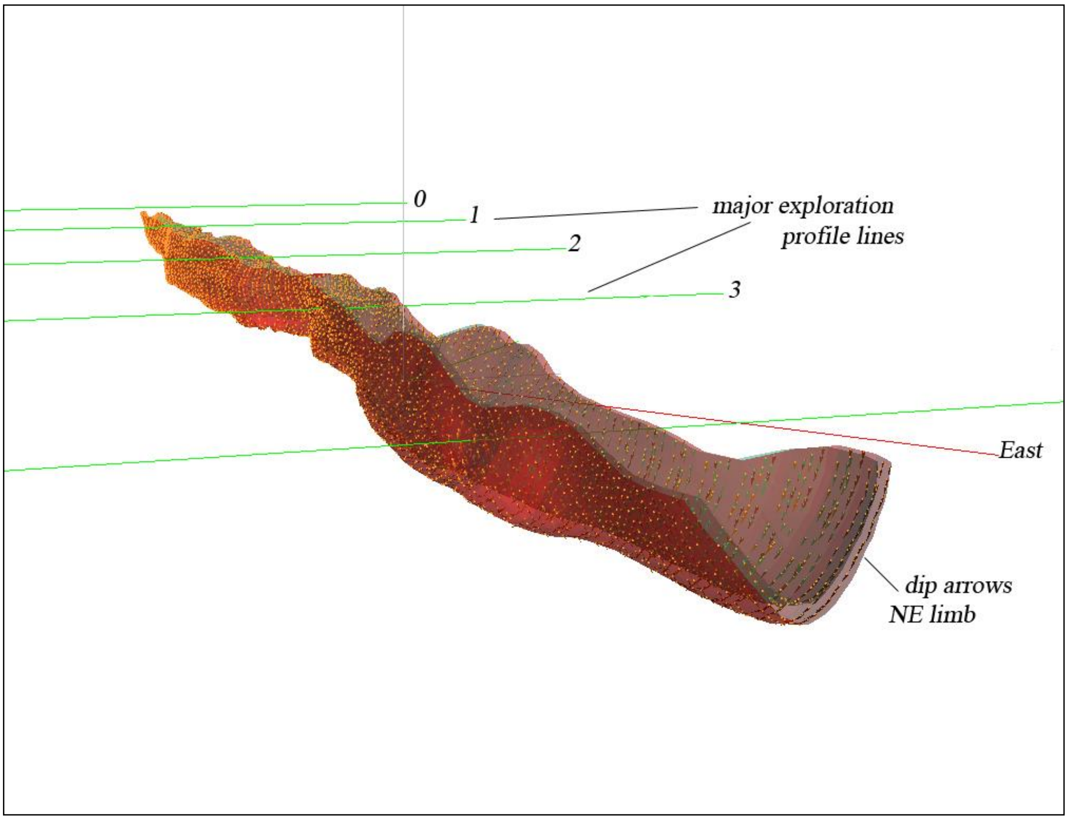


Figure 4-56: 3D perspective looking northerly and showing OB1 model with true dip directions

However, because of the extreme local geometric complexity, it appeared impractical to generate non-conflicting trends into the model, for interpreting the true local changes in the ellipsoid. Therefore, GMR took an alternative approach to the Geodynamic Anisotropy application, and this involved generating geometric sub-domains, according to local changes in dip and strike, along the strike length for each limb. Specifically, limbs SW and NE were treated differently; based on variable dip/strike directions and based on a consistent dip but variable strike, respectively.

4.12.2 SW LIMB ESTIMATION

The SW limb appears more complex than the NE limb, due to variability of both dip and strike changes. This resulted in defining seven sub-zone geometric domains according to their distinct dip and strike directions; dip directions range from of 75° westerly to 80° easterly, and strike range directions from 307° to 320°. A fundamental 3D variogram model was generated and used to represent the search ellipsoid for selecting samples, but corrected for sub-domain geometric changes in dip and strike. Sub-model blocks were extracted accordingly and sub-region search ellipsoids defined in accordance to their respective sub-domains. Note that for both limbs the experimental and modelled variograms are comparatively similar, and have a strong consistency within the defined dip and strike ranges.

4.12.3 NE LIMB ESTIMATION

The grade estimation approach was slightly different for the NE limb. The dip was more or less constant at 60° west but strike changes were quite variable and abrupt along the length of the syncline and it was not realistic to identify distinct sub-domains. However, it was possible overall, to represent the variable strike change directions, according to three direction angles: 311°, 313° and 315° – to form three strike models. Sample selection search ellipsoids were based on the 3D variogram models, and blocks that were estimated within the variogram ranges (JORC (2012) "indicated") were given precedence over block grade estimations generated from ellipsoid ranges beyond the variogram ranges, when combining the three strike grade models together. This successfully allowed the correct selection of model blocks that were estimated according to the "indicated" category for the NE limb.

4.12.4 GRADE BLOCK MODEL

There were a number of drillholes which intersected the axial zone of the fold and their samples were common to both the SW and NE limb sample sets for the grade interpolation. A rectangular anisotropic search ellipsoid, for the selection of the 2 m composite samples, was based on the variogram models. A general rule for the selection of the 2 m composite samples, within the basic ellipsoid ranges require at least three samples to a maximum of ten samples to be selected before a block grade can be estimated (classed at a JORC "indicated" level). For the second search distance, the basic variogram or ellipsoid ranges are expanded by 1.5x and a minimum of two to a maximum ten composite samples are required for estimation (classed at a JORC "inferred" level). For the very minor number of blocks still unestimated, a factor of 3x the ellipsoid, with a minimum of one to a maximum of ten samples selected and also classed at the lowest JORC level of "inferred".

The final two grade limb models were added together to form the completed grade block model for the primary zone. The output model included the following attributes:

- OK estimates
- Kriging variance
- IPD estimates
- NN (nearest neighbour) estimates
- Mean of sample values involved for each estimate
- Number of samples involved for each estimate
- JORC classification code 1= "indicated", 2= "inferred"

4.13 GRADE ESTIMATION FOR THE OXIDE ZONE

Detailed historical grade information and information from FAR's pilot plant open pit operations, demonstrate that the oxide zone's V_2O_5 grade is consistently higher than the primary zone at depth. Additionally, the oxide layer on average can be defined to a depth of 10 m from surface, but vertical thickness locally can be quite variable. However, the oxide represents only about 5 % of the total resource for OB1.

To define the model oxide blocks, the wireframe digital terrain model was dropped 10 m and this oxide contact surface was used to flag the oxide blocks in the model. No grade interpolation of the oxide blocks was undertaken, because overall the historical (1947) trench sample data were too sparse to properly represent this surface layer: 31 trenches intersected the NE limb and only nine trenches for the SW limb. However, the average weighted trench sample grades were used to assign a global grade for these oxide blocks, and GMR considered that an “inferred” JORC classification was acceptable for this zone at 0.89 % V_2O_5. Note: the overall sparse density of the trench data, precluded a grade interpolation for the orebody oxide model blocks, and this was confirmed by indications from experimental variogram ranges. Additionally, appropriate due diligence sampling would be necessary to confirm the historical trench results before upgrading this JORC resource, in addition for the need to increase the number of samples along the strike length of the oxide cap.

4.14 VALIDATION OF GRADE ESTIMATES

The results of the block-grade estimations in the resource model need to be endorsed, which includes extensive visual checking of the blocks in relation to actual drillhole sample assays and comparing the results of different grade interpolations with a de facto geostatistical approach. For the block-grade interpolation comparison, nearest neighbour (NN – value of the nearest sample to the estimated block) and inverse power distance squared (IPD^2), were used to compare with the ordinary kriging (OK) geostatistical results. This interpolation comparison is based on a statistical summary of “swaths” or regular slices, along three orthogonal axial directions of X, Y and Z through the grade model. These results are graphically represented by swath plots and for both the X (rows) and Y (columns) directions the width of the swath was set at 200 m and for the Z (level) direction a width of 20 m was selected. Figure 4-57 illustrates the rows, columns and levels used in the swath plots.

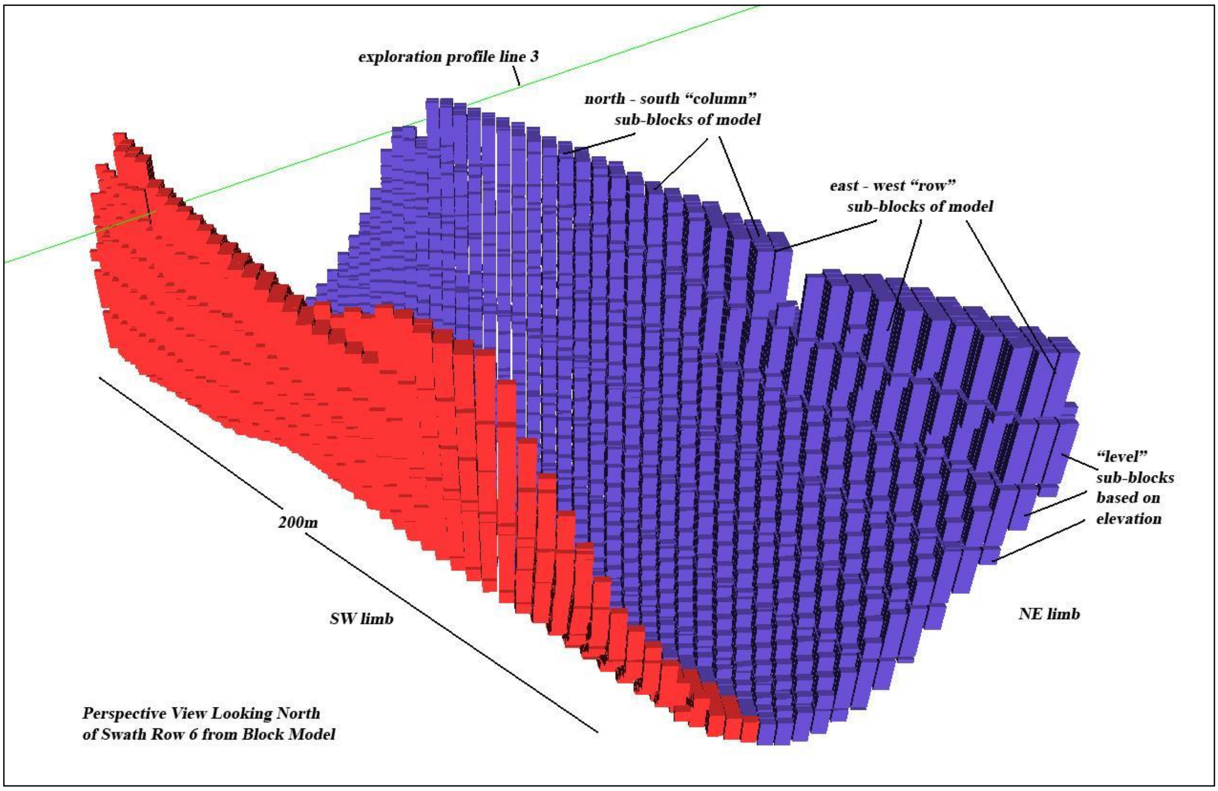


Figure 4-57: Illustration of "column", "row" and "level" used for the swath graphs – 3D block model

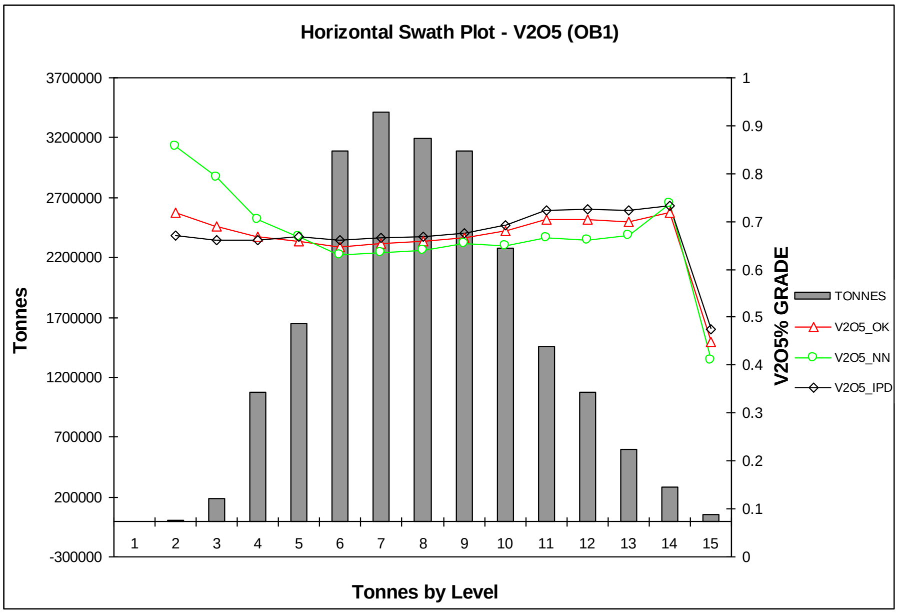


Figure 4-58: Horizontal swath plot - 310mRL to 590mRL in 20m increments (x14 levels 2 to 15)

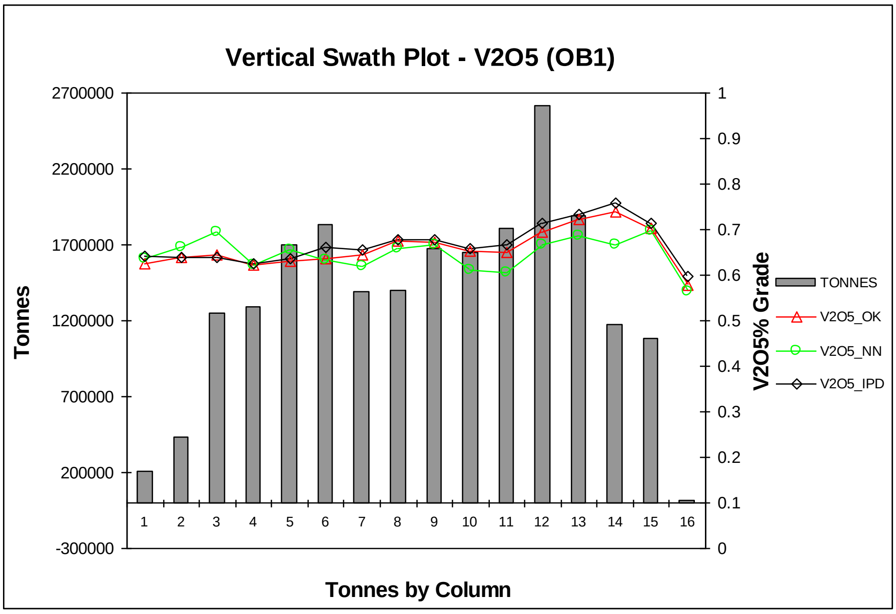


Figure 4-59: Vertical column swath plot 9450E to 372650E in 200m increments (x16)

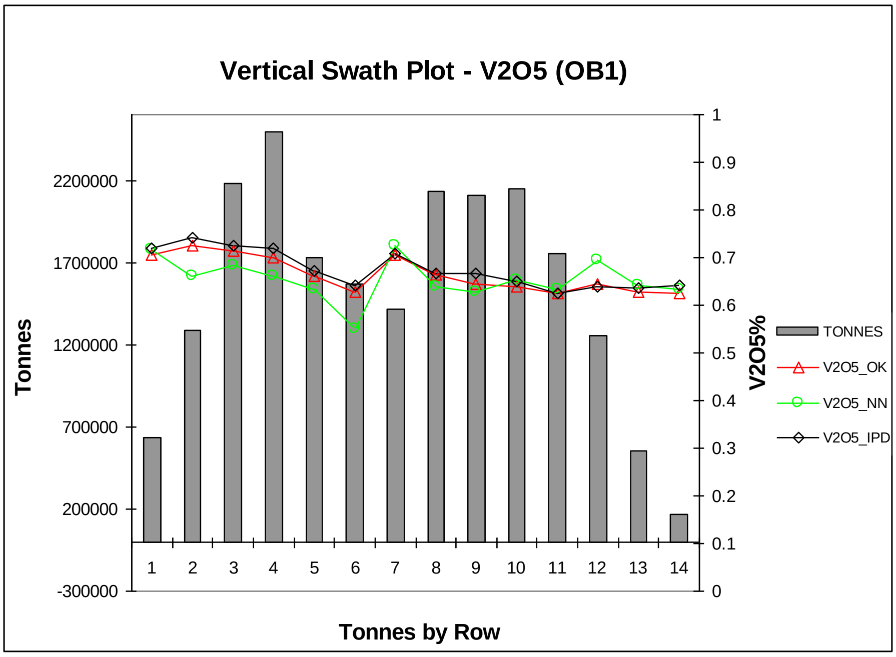


Figure 4-60: Vertical row swath plot (931960N to 934760N) in 200m increments x14

The V_2O_5 swath graphs presented in Figure 4-58, Figure 4-59, and Figure 4-60 show reasonably good correlations between the various estimation methods. OK and IPD exhibit very similar grades, though the IPD does show a slightly higher trend, which is more pronounced at higher grade levels, and this is a typical grade interpolation observation between these two popular methods of estimation. For the NN results, a more random-type of observation might be expected, in relation to the other two methods, but there is an unexpected observable lower grade trend bias for each model swath direction (ignoring low tonnage areas). A careful interrogation of this anomaly showed that this bias is due to the generally low angle of drillhole intersection through the steep tabular synclinal limbs and coupled with a tendency of lower V_2O_5 grades at the footwall and hangingwall margins. This will result in blocks which are physically higher than the hangingwall contacts or lower than the footwall contacts to have a greater spatial affinity with the lower assay grades at these contact zones. This does not affect the validity of the estimations and this nearest neighbour bias trend would not be seen if the same drillholes had intersected more orthogonally through the ore layer. The overall global averages presented in Table 4-61, show very good correlations between the various methods used to estimate the block model vanadium grades.

Table 4-61: Showing overall mean grades for OB1 JORC (2012) "indicated"

Model Estimates	OK %	NN %	IPD %	Samples* %	Tonnes (millions)
V_2O_5	0.67	0.65	0.68	0.68	21.4
Carbon	14.08	13.87	13.97	13.93	10.7

** Overall arithmetic mean grade of the samples used for each block estimation*

Carbon, although a by-product, is a major revenue generator, and the validation results for the estimation of the block grades are presented in swath plots in Figure 4-61, Figure 4-62 and Figure 4-63.

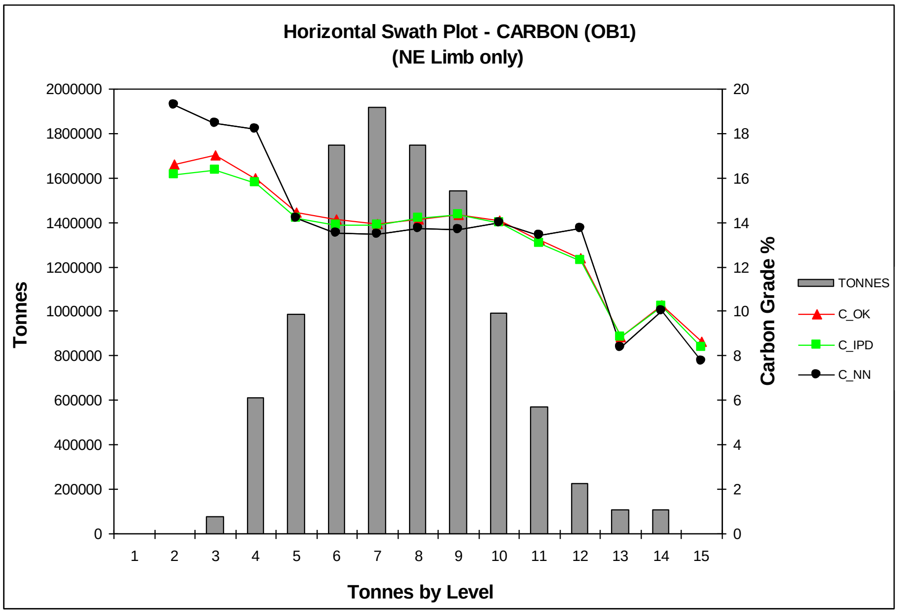


Figure 4-61: Horizontal swath plot Carbon 330mRL to 570mRL in 20m increments (x12 levels - 3 to 14)

Figure 4-61, Figure 4-62, Figure 4-63 and Figure 4-64 illustrate a trend of higher grades at the hinge of the syncline, where the carbonaceous sediments are more silicified in areas where there are intense compressional forces, as seen in the open pit where complex buckle folding occurs at the nose of the syncline.

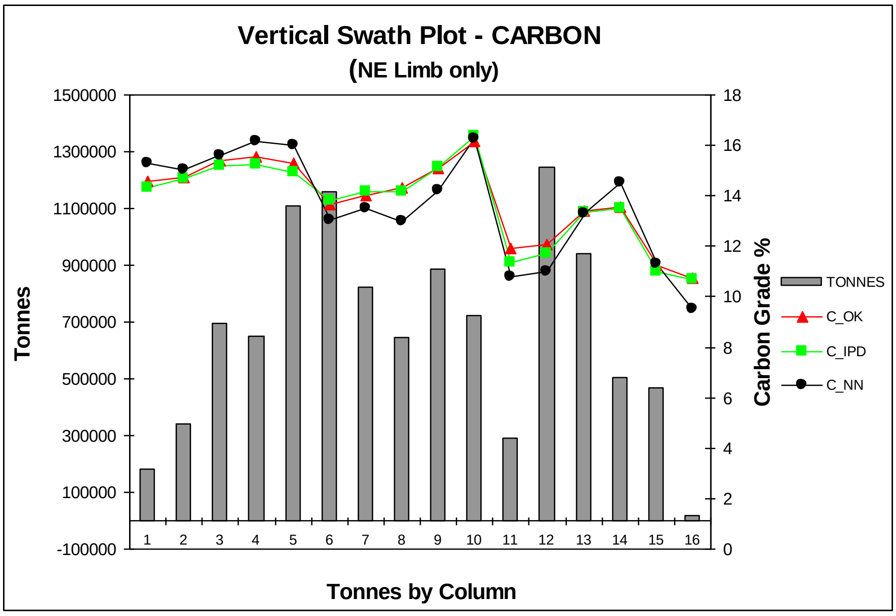


Figure 4-62: Vertical column swath plot Carbon (369450E to 372650E in 200m increments) (x16)

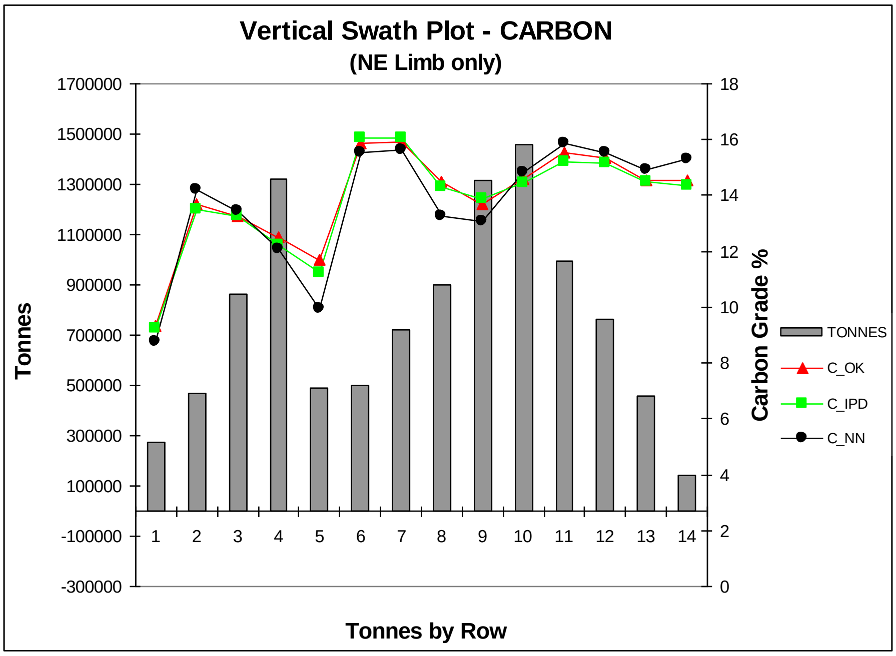


Figure 4-63: Vertical row swath plot Carbon (931960N to 934760N in 200m increments x14)

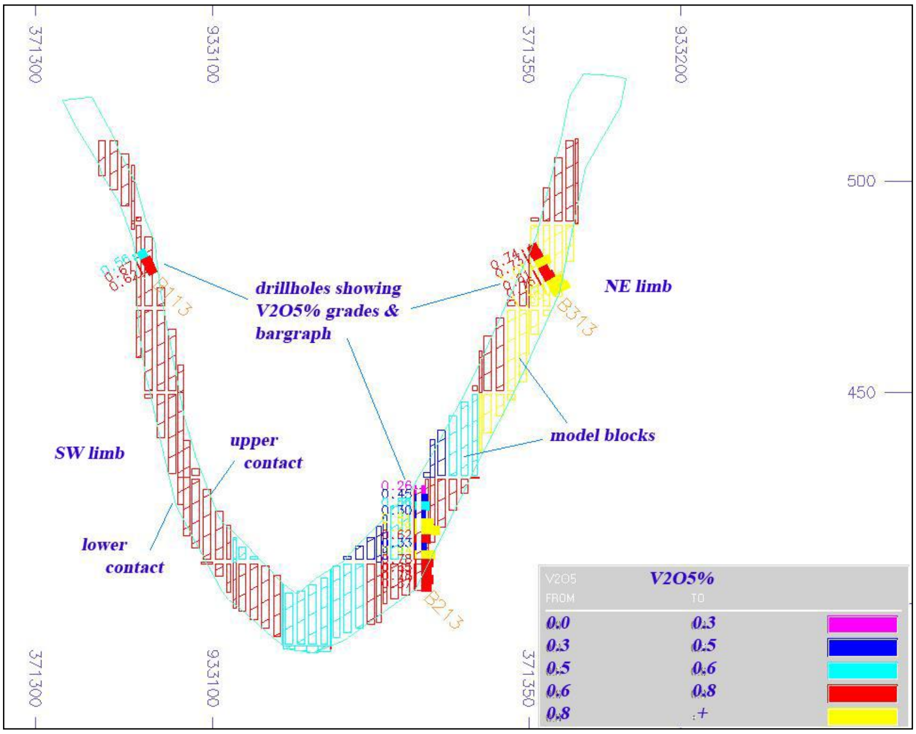


Figure 4-64: SW-NE section of model (Profile 3) showing drillhole grades and estimated block grades

4.15 RESOURCE CLASSIFICATION

The author has used the guidelines endorsed by the established Australasian Code for the Reporting of Mineral Resources and Ore Reserves (JORC, 2012). The system is split into Mineral Resources and Ore Reserves. A Mineral Resource is an in situ concentration of material of intrinsic economic interest in or on the Earth’s crust in such form and quantity that there are reasonable prospects for eventual economic extraction. Depending on levels of confidence, the Mineral Resource is split into Measured, Indicated, and Inferred. JORC (2012) states that an Ore Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource, allowing for dilution and losses that may occur during mine extraction.

The three main categories of Measured, Indicated and Inferred Resources have been defined as follows:

- Measured Mineral Resource is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. It is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are spaced closely enough to confirm geological and/or grade continuity.

Guidelines in the code show that for a Resource to be classified as Measured, the confidence in the estimate is such that additional technical information would not significantly affect technical or economic decisions made on the basis of the estimate.

- Indicated Mineral Resource is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a reasonable level of confidence. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed.

  The Indicated Resource has a lower level of confidence than the Measured Resource, but a higher confidence level than the Inferred Resource category. Confidence in the estimate would be such as to allow the application of technical and financial parameters and to enable an evaluation of economic viability. As an example, if infill drilling could significantly affect the shape and/or distribution of the mineralised zones but not substantially affect the tonnage/grade estimate, then an Indicated classification could be justified.
- Inferred Mineral Resource is that that part of a Mineral Resource for which tonnage, grade and mineral content can be estimated with a low level of confidence. It is inferred from geological evidence and assumed but not verified geological and/or grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that may be limited or of uncertain quality and reliability. Confidence in this estimate would be too low to allow the appropriate application of technical and economic parameters for Ore Reserve assessment.

4.16 RESOURCE STATEMENT

4.16.1 SCHEDULE OF MINERAL RESOURCES

Table 4-62 summarises the schedule of JORC (2012) mineral resources for both the primary resource and the oxide resource, which are discussed in the following sections. The JORC (2012) mineral resources are broadly analogous to the state GKZ mineral resources using similar geological systems to define the resource category and should be read as complimentary.

Table 4-62: Schedule of JORC (2012) Mineral Resources

JORC Vanadium Resource OB1				By-Products OB1 (primary ore only)						
				JORC indicated		JORC inferred		Total C% Mean	JORC inferred MoO_3 % mean	JORC inferred U_3O_8 % mean
JORC Class	V_2O_5 % Cut-off	V_2O_5 % Mean	Tonnes [m]	C% Mean	Tonnes [m]	C% Mean	Tonnes [m]			
Indicated	0.0	0.67	21.43	14.08	10.68	13.09	10.75	13.59	0.0300	0.0090
Inferred	0.0	0.67	1.56			13.43	1.56	13.43	0.0297	0.0085
Combined	0.0	0.67	22.99					13.58	0.0300	0.0090
Oxide cap inferred	0.0	0.89	1.33							
Total	**0.0**	**0.68**	**24.32**							

4.16.2 PRIMARY RESOURCE

The primary mineralisation is the major contributor of the resource at about 95 % of the total and the remainder of the vanadium resource is confined to the oxide cap. Table 4-63 provides details of the OB1 primary mineralisation resource, while Figure 4-65, Figure 4-66 and Figure 4-67 illustrate grade tonnage curves for JORC (2012) Indicated, Inferred and combined resources, respectively, for the primary mineralisation.

Table 4-63: OB1 Resource (Primary Mineralisation)

				By-Products						
JORC Class	V_2O_5% Cut-Off	V_2O_5% Mean	Tonnes Millions	JORC Indicated		JORC Inferred		Total C% Mean	JORC Inferred MoO_3% Mean	JORC Inferred U_3O_8% Mean
				C% Mean	Tonnes Millions	C% Mean	Tonnes Millions			
Indicated	0.0	0.67	21.43	14.08	10.68	13.09	10.75	13.59	0.0300	0.0090
	0.1	0.67	21.43	14.08	10.68	13.09	10.75	13.59	0.0300	0.0090
	0.2	0.67	21.43	14.08	10.68	13.09	10.75	13.59	0.0300	0.0090
	0.3	0.67	21.34	14.13	10.62	13.11	10.72	13.62	0.0301	0.0091
	0.4	0.67	21.25	14.16	10.55	13.11	10.70	13.63	0.0301	0.0091
	0.5	0.67	21.06	14.16	10.43	13.10	10.63	13.62	0.0302	0.0091
	0.6	0.69	18.43	14.13	8.87	13.08	9.56	13.58	0.0302	0.0093
	0.7	0.77	5.67	13.66	3.10	13.34	2.57	13.51	0.0303	0.0109
	0.8	0.87	1.38	14.75	0.63	13.59	0.74	14.12	0.0309	0.0098
	0.9	0.98	0.35	16.47	0.10	13.60	0.25	14.43	0.0290	0.0086
	1.0	1.04	0.11	17.57	0.02	13.77	0.09	14.56	0.0302	0.0090
Inferred	0.0	0.67	1.56			13.43	1.56	13.43	0.0297	0.0085
	0.1	0.67	1.56			13.43	1.56	13.43	0.0297	0.0085
	0.2	0.67	1.56			13.43	1.56	13.43	0.0297	0.0085
	0.3	0.67	1.56			13.43	1.56	13.43	0.0297	0.0085
	0.4	0.67	1.56			13.43	1.56	13.43	0.0297	0.0085
	0.5	0.67	1.56			13.43	1.56	13.43	0.0297	0.0085
	0.6	0.67	1.49			13.41	1.49	13.41	0.0298	0.0085
	0.7	0.72	0.52			13.23	0.52	13.23	0.0297	0.0094
	0.8	0.82	0.03			14.00	0.03	14.00	0.0286	0.0112
Combined	0.0	0.67	22.99					13.58	0.0300	0.0090
	0.1	0.67	22.99					13.58	0.0300	0.0090
	0.2	0.67	22.99					13.58	0.0300	0.0090
	0.3	0.67	22.90					13.61	0.0301	0.0090
	0.4	0.67	22.81					13.62	0.0301	0.0090
	0.5	0.67	22.62					13.61	0.0301	0.0090
	0.6	0.69	19.92					13.57	0.0302	0.0092
	0.7	0.77	6.19					13.49	0.0302	0.0108
	0.8	0.87	1.41					14.12	0.0309	0.0098
	0.9	0.98	0.35					14.43	0.0290	0.0086
	1.0	1.04	0.11					14.56	0.0302	0.0090

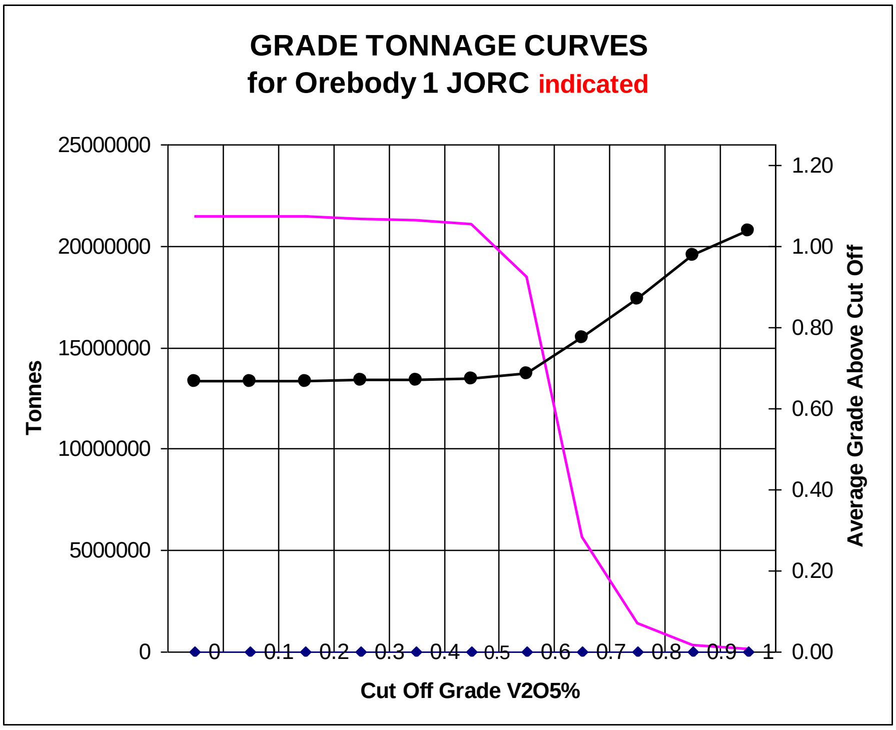


Figure 4-65: Grade tonnage JORC Indicated

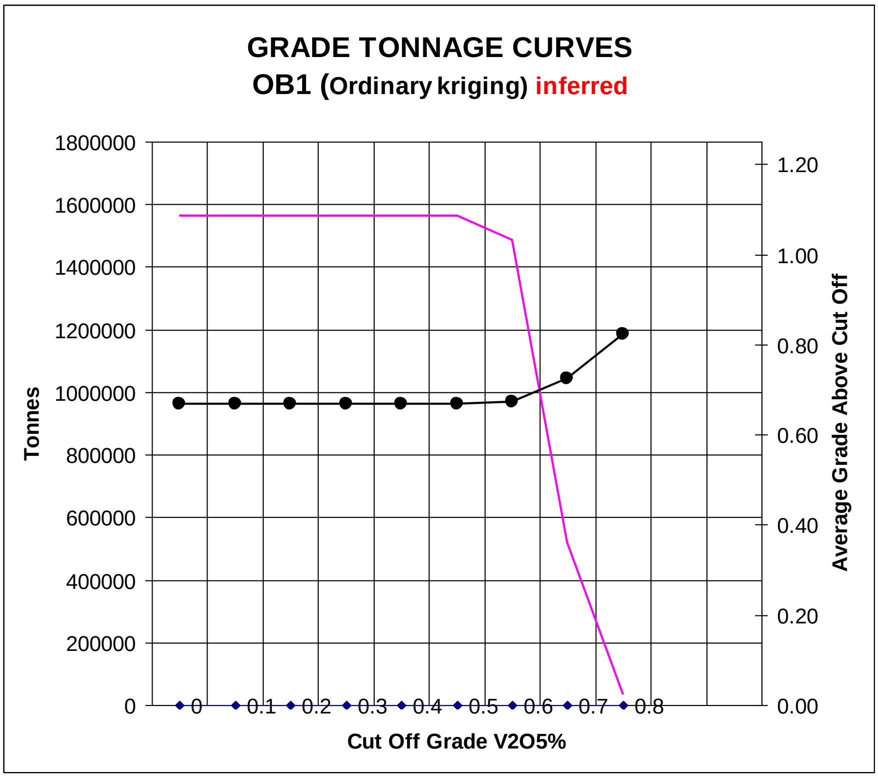


Figure 4-66: Grade tonnage JORC Inferred

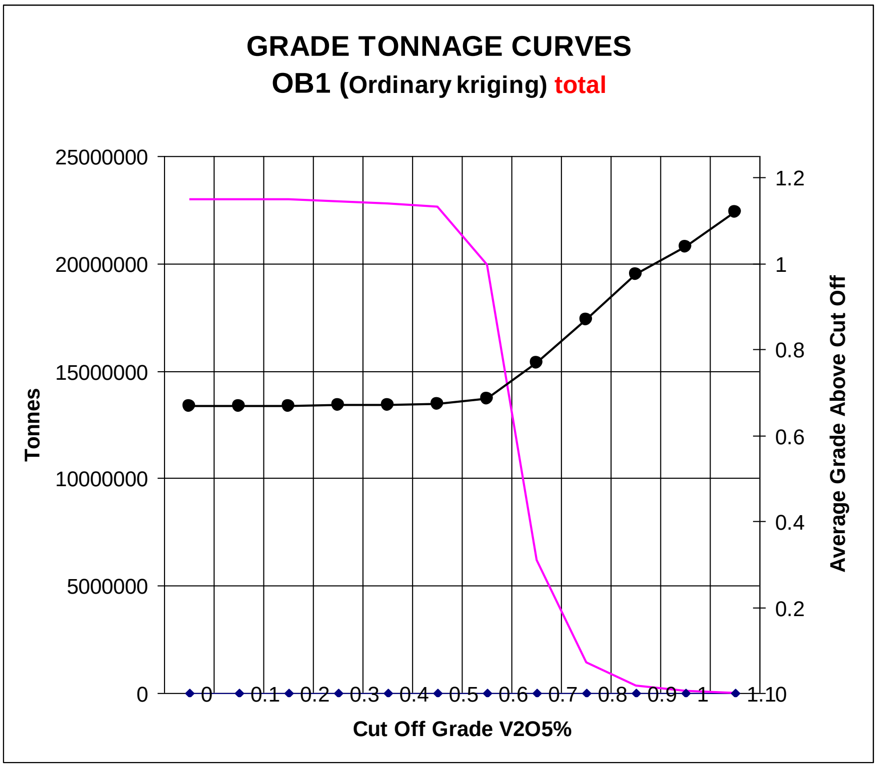


Figure 4-67: Grade tonnage JORC Combined

4.16.3 OXIDE RESOURCE

Although an oxide block model has been generated, it has only been possible to estimate a global vanadium grade from the historical surface trench data. Table 4-64 summarises the JORC (2012) Inferred Mineral Resource for the oxide mineralisation.

Table 4-64: OB1 Resource (Oxide Mineralisation)

JORC (2012) Vanadium Resource OB1 – Oxide Cap			
JORC Class	V_2O_5% Cut-off	V_2O_5% mean	Tonnes millions
Inferred	0.0	0.89	1.33

4.17 JORC EXPLORATION TARGETS (OB2, OB3, OB4 AND OB5)

JORC (2012) permits the reporting of exploration target size and type with strict conditions, whereby grades and tonnes are expressed in ranges and a detailed explanation for the basis for such a statement is submitted.

Historically, the surface vanadium oxide zone has been well documented from trench sampling plus some subsurface exploration drives and shows distinct continuity, but at depth where the primary ore constitutes about 95 % of the resource, only a limited amount of drilling has been completed for OB2 and OB3. However, based on the distinct surface expression of these orebodies (as illustrated in Figure 4-68) and, by analogy with the OB1 resource, it has been possible to derive approximate ranges of expected tonnages for OB2 to OB5. The exploration targets for the primary resource are summarised in Table 4-65 and Table 4-66, while the exploration targets for the oxide resource are summarised in Table 4-67.

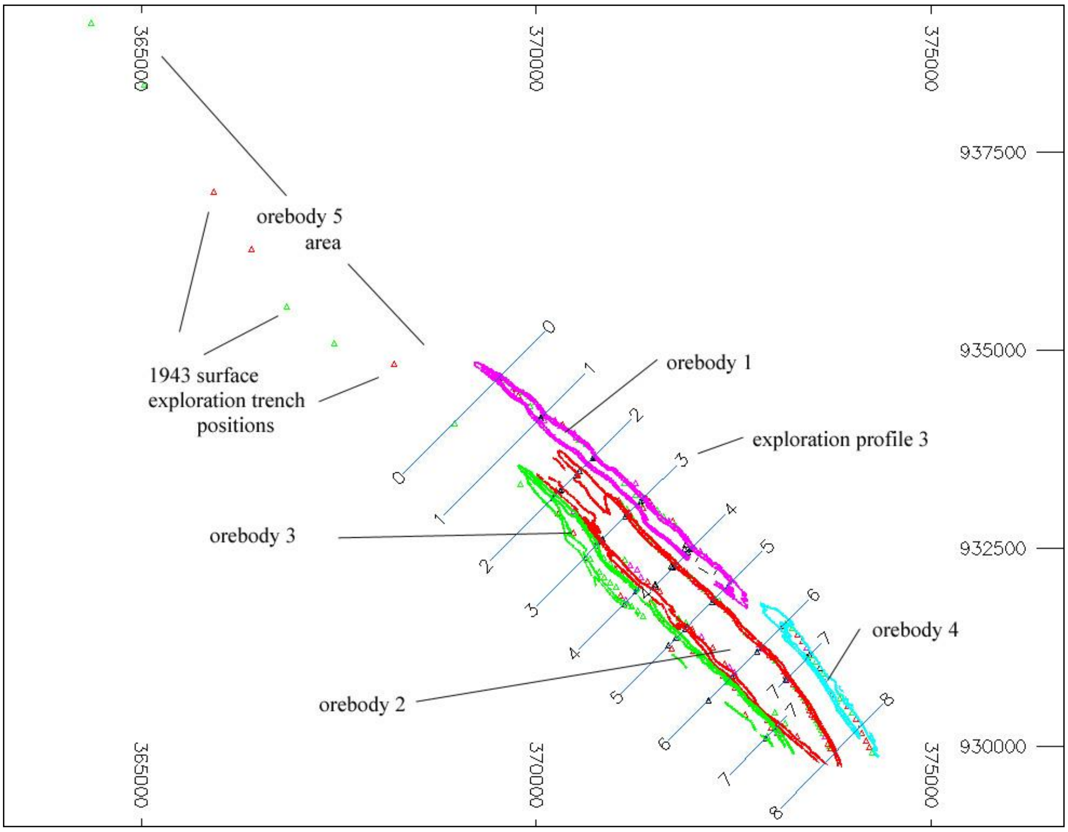


Figure 4-68: Historical surface outline of orebodies – based on 1947 map

4.17.1 PRIMARY EXPLORATION RESOURCE

4.17.1.1 DETAILED EXPLANATION

The following parameters were used to define the tonnages and V_2O_5 grades for assessing JORC exploration targets in the primary mineralisation, refer to Table 4-66:

1. Tonnages based on the OB1 JORC resource, where there are 23 million tonnes of resource with a strike length of 4.5 km and this equates to 5.11 million tonnes per km. The factor 5.11 is multiplied to the strike length for each orebody to obtain the expected median tonnage and the tonnage range has been calculated as ±15 % to the median tonnage. For OB5, which only has a single limb structure, the tonnage has been halved.
   Total tonnes = strike length in km*5.11 million.
   - OB2 global tonnage = 28.1 million (strike length =5.5 km)
   - OB3 global tonnage = 24.5 million (strike length 4.8 km)
   - OB4 global tonnage = 13.3 million (strike length 2.6 km)
   - OB5 global tonnage = 40.9/2 = 20.4 million (strike length 8.0 km)
2. V_2O_5 Grade: The expected global mean grade for each orebody was derived from the relatively scant historical and/or FAR drilling mean grade results and the grade range applied at ±5 % of the mean grade.
   - OB2 mean grade = 0.66 % - based on former Soviet-era drilling
   - OB3 mean grade = 0.69 % - based on Soviet-era drilling and FAR drilling
   - OB4 mean grade = 0.67 % - considered as a strike extension of OB1 and likely to contain a similar global grade.
   - OB5 mean grade = 0.69 % - it appears to be a strike extension to OB3 and so the same global grade has been applied.

Table 4-65: JORC-based Exploration Target (JORC 2012 guidelines)

Orebody	Strike Length (km)	TONNES (million)*		V_2O_5% GRADE RANGE	
		From	To	From	To
2	5.5	24	32	0.63	0.69
3**	4.8	21	28	0.66	0.73
4	2.6	11	15	0.64	0.71
5**	8.0	17	23	0.66	0.73
Total	20.9	73	98	0.65	0.71

** based on bulk density of 2.4 ** OB5 is a strike continuation of OB3, as a single limb layer only*

Table 4-66: JORC-based Exploration Target (JORC 2012 guidelines) - By-products applied to all Ore Bodies 2 to 5

Target	Global Grades based on OB1	GRADE RANGE ± 5%	
		From	To
Carbon	13.58%	12.9	14.26
MoO_3	0.030%	0.029	0.032
U_3O_8	0.009%	0.009	0.009
REM	335 ppm	318	352
Total Tonnes (millions)		73	98

4.18 OXIDE EXPLORATION RESOURCE

4.18.1 DETAILED EXPLANATION

The following parameters were used to define the tonnages and V_2O_5 grades for assessing JORC exploration targets in the oxide mineralisation:

1. Tonnages based on the OB1 JORC resource, where there are 1.33 million tonnes of resource with a strike length of 4.5 km and this equates to 0.296 million tonnes per km. The factor 0.296 is multiplied to the strike length for each orebody to obtain the expected median tonnage and the tonnage range has been calculated as ±15 % to the median tonnage. For OB5, which only has a single limb structure, the tonnage has been halved.
   Total tonnes = strike length in km*0.296 million.
   - OB2 global tonnage = 1.63 million (strike length 5.5 km)
   - OB3 global tonnage = 1.42 million (strike length 4.8 km)
   - OB4 global tonnage = 0.77 million (strike length 2.6 km)
   - OB5 global tonnage = 2.37/2 = 1.18 million (strike length 8.0 km)
2. V_2O_5 Grade: Only the expected global mean grade of 0.89 % for the oxide cap of OB1 has been applied to these orebodies and the grade range applied at ±5 % of this global grade.

Table 4-67: JORC-based Exploration Target (JORC 2012 guidelines) Oxide Cap

Orebody	Strike Length (km)	TONNES (million)*		V_2O_5% GRADE RANGE	
		From	To	From	To
2	5.5	1.39	1.87	0.85	0.98
3**	4.8	1.21	1.63	0.85	0.98
4	2.6	0.65	0.89	0.85	0.98
5**	8.0	1.00	1.36	0.85	0.98
Total	20.9	4.25	5.75	0.85	0.98

** based on bulk density of 1.7 ** OB5 is a strike continuation of OB3, as a single limb layer only*

4.19 RESERVES

Detailed mine planning for OB1 has been undertaken using open pit optimisation software (“Datamine”). The open pit was designed to take into account the Indicated resource which also includes the Inferred resource representing the oxide cap together with inferred by-product values. Since only the Indicated part of the resource can be used to form a Reserve under the JORC (2012) system of classification the inferred resources were taken as waste for Reserve reporting purposes in the following table:

JORC Based Mineral Reserves (JORC 2012 Guidelines) - Ore Body 1 only

Category	Reserve Tonnes (000)	Mean grade V_20_5 [%]
Probable	22,938	0.59

GKZ RESERVES

A mineral resource in Kazakhstan is registered with the state committee GKZ. This is a legal requirement and the mine will operate under the GKZ jurisdiction and in accordance with the state rules for subsoil use. The JORC resource estimate is a separate exercise and is used to provide an additional and complimentary resource estimate for the reader who may not be familiar with the GKZ resource classification system. In May 2014 the GKZ reserve detailed in Table 4-68 was confirmed and placed onto the national register. This GKZ reserve totals 71 M tonnes and is the basis for regulatory control.

Table 4-68: 2014 GKZ Reserve Summary

Category	Reserve [1000 t]	Mean grade V_20_5 [%]
B	832	1.00
C1	15,649	0.75
C2	54,366	0.74
B+C1+C2	70,847	

4.20 CONCLUSIONS AND RECOMMENDATIONS

Potentially, the primary resource is huge, as expressed by the surface continuity of the vanadium mineralisation along strike. The reflection at depth of such observable surface mineralisation has been confirmed by FAR’s drilling of OB1 and also confirmed from the more limited drilling of OB2 and OB3. Currently, based on the OB1 JORC resource, plus JORC-based Exploration Targets for OB2 to OB5, a total vanadium GKZ or JORC resource of over 100 million tonnes is considered to be a rational prediction. Note: additional anomalous surface vanadium mineralisation has also been defined from historical exploration and which has not been included in this estimate –see Figure 4-9 geological map.

GMR has advised that for an optimum assessment of the rare earths potential, there is a need to identify the specific REM associated minerals within the primary ore, to allow a better understanding as to which rare earth elements should be analysed. It is understood that FAR has made arrangements for this work, but results have not been seen.

SECTION 5 MINING

5.1 INTRODUCTION

The ore bodies of the Balasausqandiq deposit are outcropping. The commercial vanadium mineralisation geometry forms elongated synclinal structures. The shape of the mineralisation lends itself to a conventional open pit mine design.

The dipping of the ore body at the flanks of the synclinal structures generally ranges from 65 to 90°. The surface expression of the vanadium-bearing horizon typically shows a thickness range from 5.0 to 19 m.

Within the limits of the outcropping ore bodies, the siliceous intercalations form clear-cut, saddleback hills and hilltop surfaces with gaps in between them; thus the terrain of the deposit area is inverse, i.e. the trough cores of the synclinal structures repeat the positive shape of the surface terrain. Outcrops of the ore bodies are associated with the upper part of the slopes. Striking of the ridges matches the striking of the structures. The top parts of the ridges are quite flat.

Mining excavation commenced at Balausa in 1971 during the Soviet era. The open pit that was in operation is now called the Old Pit which is located in the middle of OB1. Whilst in production a total of 12 kt of ore was extracted for pilot plant mineral processing analysis.

In 2009, FAR developed another open pit called New Pit which is located at the north western end of OB1. The current status of New Pit is shown in Figure 5-1.

5.2 MINING PLANNING

It is intended to mine the New Pit at a rate of 500ktpa with a rapid ramp up to 1Mtpa as commissioning proceeds. Long term mining design, planning and operations are conventional as the open pit layout is orthodox and uncomplicated. The mining operation extraction program will follow the orebodies along strike and to an economically defined depth which, for OB1, is the full depth of the synclinal structure.

Figure 5-1: Current Status of New Pit

5.2.1 MINING CUT-OFF GRADE

According to normal mining convention, cut-off grade is defined as the level of mineralisation that yields a nil operating revenue, with operating revenue being the net mine site realisation from sales, less all mine site direct operating costs for mining, milling, and general and administration.

This process was used to create mining excavation boundaries, defining what is waste and thus is not economically viable to process and sell.

The cut-off grade used to outline the mine excavation plan was determined using the following formula:

*Cut-off Grade = Operating Costs / Metal price * Metallurgical Recovery*

The following values were used:

Operating Cost	35 USD/t of ore (excluding transport and smelter charges)
Metallurgical Recovery	90 %
Revenue Price	135 USD/t
Cut-off Grade	0.1 % Vanadium (V_2O_5)

A cut-off grade of 0.1 % V_2O_5 was therefore used to determine the mine plan and ore volumes. In practice, there is a sharp boundary between ore and the waste, which contains little vanadium, so the cut-off can be taken as the limit of the vanadium-containing strata.

5.2.2 OPEN PIT DESIGN

The mine design has been undertaken using industry standard computerised design software. Figure 5-2 shows the current pit design and Figure 5-3 shows the cross-sectional view through the middle of the pit, looking towards the northwest. The mine design is based on 15 m bench height, 5 m safety berms, with an overall slope angle of the open pit expected to be approximately 75 °.

5.3 GEOTECHNICAL PARAMETERS

Historic geotechnical investigations were completed based on the geological and geo-mechanical programme carried out in the Soviet era. Subsequently a mine design geotechnical report was written in 2001 by CJSC “KazGIIZ”. This report was part of investigations to ensure that the area designated for the mineral process pilot plant was suitable, did not sterilise mineral resource, allowed the development of the open pit and also determined the location of the final operational mineral process plant.

It is noted that the rock quality designation is good and stable open pit mine walls are envisaged. It is worth noting that since the original excavations in 1943, no caving or sloughing was noticed in the trench walls or in the roof of the underground adits.

5.4 HYDROLOGICAL PARAMETERS

The ground water of the region that hosts the mineral resource is confined to the outcropping zone of middle Cambrian age; the lithological nature comprises carbonaceous-clayey, siliceous, and sericite-chloritic schists. The rocks are highly schistose and broken by tectonic fissures into separate blocks. The depth or thickness of this rock type is approximately 100 m in average. The depth of ground water in the area of the open pits varies from surface to 14.5 m below surface.

The permeability or filtration properties of the rocks are low. The filtration ratio varies from 0.047 to 0.262 m/day. Test well flow rates vary from 0.5 to 4.2 l/s, specific flow rates vary from 0.03 to 0.196 l/s. The water is fresh, dissolved salt mineralization is 0.8-1.0 g/l and the chemical composition is mixed, hydrocarbonate-sulfate, calcium-sodium. Underground water mainly migrates to the south-west with surface expression in the topographic relief depressions in the form of springs. Water inflow to the open pit mining area will be minor, of nuisance value, from the underground water body, which is classified as a low producing water body. There will be some ingress of water due to rain and melting snow but the majority of this precipitation will be directed away from the open pit by the establishment of a surface draining system designed as part of the surface road network.

Any water ingress will be diverted from the open pit by judicious mine design; the north western area of the open pit intersects low lying topography and it will be possible to excavate a drain allowing the water from the open pit to drain naturally by a graded pit floor. This allows water to drain naturally from the south east to the north west and out of the pit.

5.4.1 SURFACE WATER FROM OPERATIONS

The mine water emanating from the open pit will be directed via surface drains to a series of dams, which will allow water to be collected for road dust suppression and general operational use. Excess ground water will be discharged according to the discharge permits as set out in the OVOS.

5.5 MINING METHOD

The mining operation is based on conventional open pit mining using conventional drilling, blasting, excavation and haulage of waste and mineralised rock by truck to the respective material location. The equipment used will be modern industry standard, and sized according to the excavation volumes.

The mine design studies have been based on an optimisation process which uses standard industry calculations to maximise the value of the waste to ore extraction ratio. The current design shows a 4.2:1 stripping ratio of waste and ore. As mining costs are a fraction of total costs and the forecast profit margins are high, a very much higher stripping ratio could be contemplated but this mine design already caters for 100 % extraction of OB1. The general inter bench slope angle is 75°.

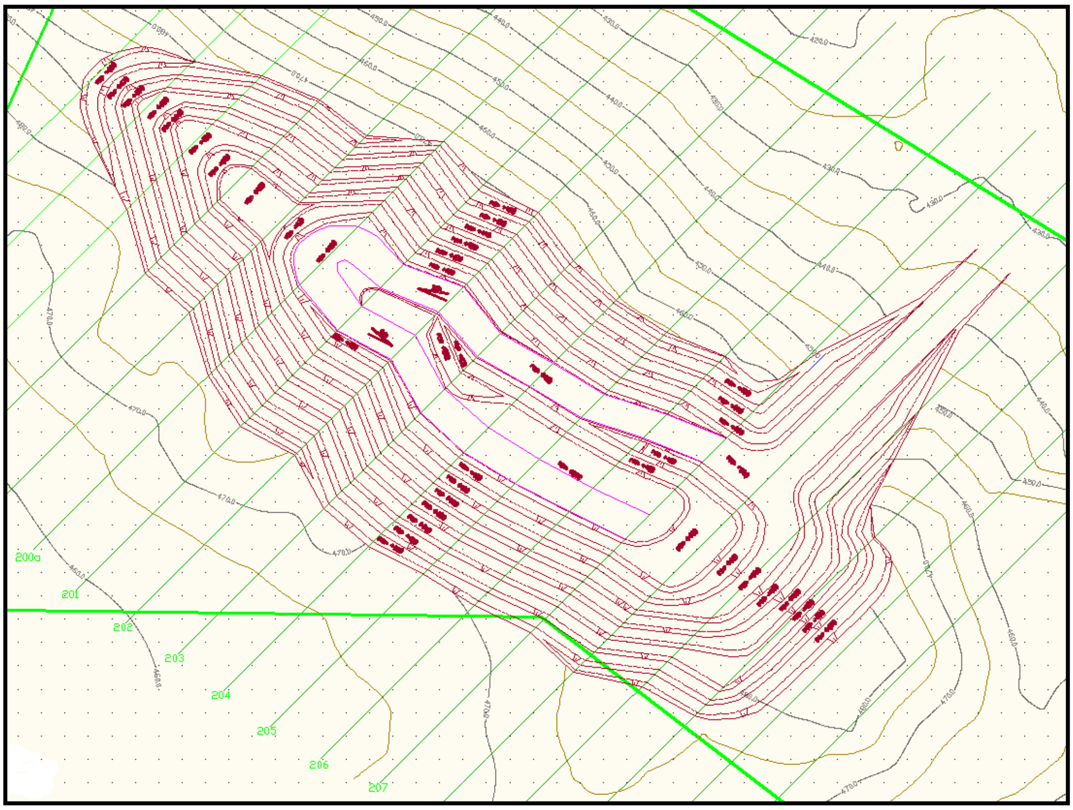


Figure 5-2: Current Design for New Pit Ore Body 1

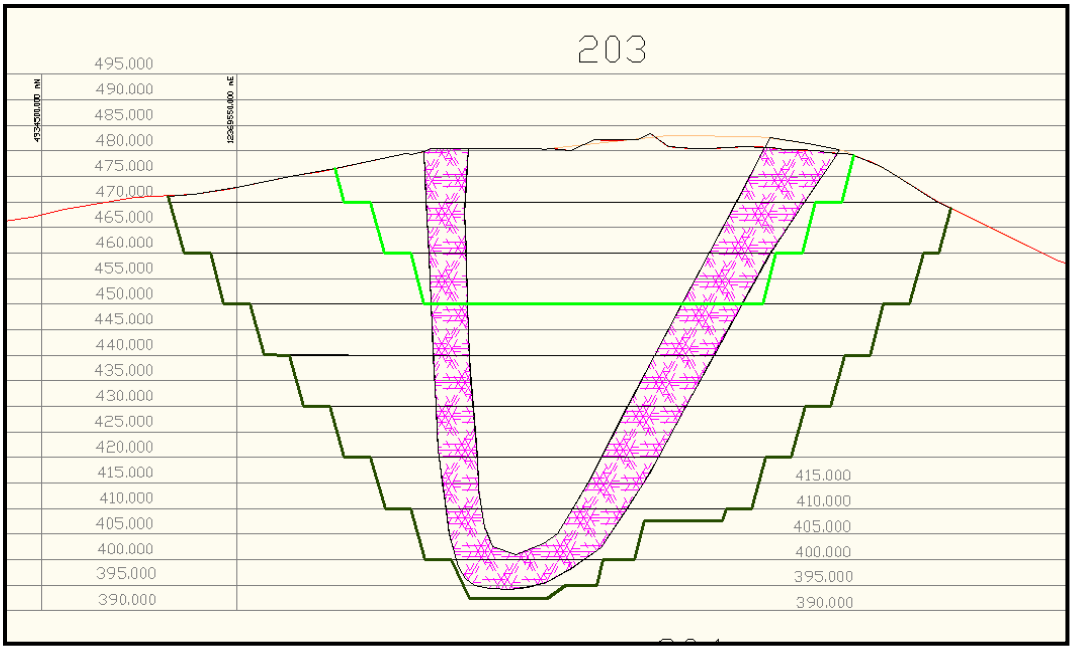


Figure 5-3: Cross-section 203 of New Pit Design Ore Body 1

5.6 DRILLING AND BLASTING

The pilot operation drilling and blasting tasks to date have been performed by a contractor. FAR intends initially to continue using a contractor for drilling and blasting, with a commercial structure based on the larger volume of rock. It will remain an option to bring the operation in-house if suitable contractual terms cannot be negotiated.

The use of a contractor minimises the need for explosives storage, handling facilities and specialist skills.

5.7 OPERATIONAL ORE GRADE CONTROL / WASTE DETERMINATION

The rocks-strata classified as ore and those classified as waste are clearly visible at footwall contact within primary ore. To ensure the quality of the mined ore is consistent and constant, a program of grade control sampling will be undertaken. This will involve the sampling of drill cuttings from the blast hole drilling process, tested in the onsite laboratory and excavation plans used daily to ensure that that the ore extracted will be at the required grade.

5.8 OVERBURDEN / WASTE REMOVAL

A waste emplacement location has been developed to the northern side of the New Pit. This material will be stored until mine closure and used for restoration purposes. During the life of the mine, the waste rock will be used for site roads, water drainage bunds and safety barriers around the site. There is also a market for aggregates for construction and local road building. Since 2010 a large quantity of rock has been sold for regional road construction.

5.9 MINING FLEET

Current and future mining fleet Table 5-1 sets out the fleet of equipment proposed for Balausa.

Table 5-1: Proposed New Capital Plant and Equipment for Mining

Name of Equipment	Number	Cost 000 USD
Main mining equipment		
Excavator Hitachi ZX850 – 5G (4.3m)	2	1,474
Loader ZL50	2	100
Bulldozer Shantui SD2	3	616
BelAZ – 7547 45 tonne	12	2,900
Grader DZ 98	2	144
Vibrating roller Dynapac CA 3500	1	136
Excavator with hydraulic hammer Hitachi ZX280-5G/Furukawa F35XP	1	245
Blasting rig Furukawa HCR 1500 – EDII	1	398
Compressor KB-12/12P CHKZ	2	73
Sprinkler truck Ural 420	2	140
Charging machine BB-MZU-16 KAMAZ-6520	1	85
Mobile workshop AROK with IM-50	1	60
Fuel Truck GAZ 36135-11	1	35
Mobile lighting system	4	57
Total		**6,464**

5.9.1 MINING OPERATIONAL INFRASTRUCTURE

The mine site already has a suitable machinery refuelling depot and diesel generator for the workshops and administration facilities. A coal-fired boiler will provide the heating requirements, later supplemented by waste heat from the sulphur burner.

5.10 LABOUR REQUIREMENTS

It is anticipated that approximately 271 personnel will be required for the mining department for Phase 1 which will consist of 158 operators and 73 management and support staff. The intended work roster from March till November will be based around two x 12 hour shifts (day and night), 10 hours work time with two one hour breaks. During the short harsh winter period a single 12 hour shift operation will take place. Workers will be on site for 15 days, sleeping in on-site accommodation, followed by 15 days off, throughout the year.

5.11 MINING SERVICES

5.11.1 MINE SERVICES, COMMUNICATION

Communications around the mine are via hand held two-way radios and mobile phones.

A system consisting of a series of base station(s), located in the control office, workshop and supervisory vehicles will be established to communicate with each piece of mobile plant.

Multiple channel systems are available which can be used to provide separate communications for different departments (e.g. mine, maintenance and surface staff), yet still enable all personnel to communicate when necessary.

5.11.2 MINE SERVICES, MAINTENANCE FACILITIES

Given the proposed size of the Balausa mine a site workshop will be constructed. This will include crane and vehicle pit facilities, storeroom, welding bays, tyre changing facilities, engine and transmission removal and other conventional tasks associated with machinery maintenance.

This will enable all vehicles to be repaired and serviced. The fuelling and lubrication facility will comprise a secure, bunded fuel tank with an electrically operated pump; commonly used greases and lubricants with either electric or pneumatic pumps; environmental and spill protection equipment; a pressure washer; and a range of tools for carrying out basic repairs and servicing.

5.11.3 MINE OFFICE AND CHANGE ROOM

The mine office and changing room facilities are established in the site building and will be suitable for the expanded operations. The office facility has sufficient room for technical and supervisory staff as well as contractors. This includes an open plan area for survey plans and geological drawings to be laid out for planning meetings. The laboratory facilities located nearby are for grade control works.

Changing facilities are equipped with a clean and dirty section where personnel can shower and change into or out of their work clothes. In addition, a heated area is provided so that industrial work clothing (i.e. overalls) can be dried ready for the next shift. A system of lockers and employee space is established to ensure security and a high regard to personal welfare.

5.11.4 HEALTH, SAFETY AND WELFARE

The company will continue to develop a culture in which each individual will be trained in a safe system of working, and thereby adhere to the health, safety and welfare management plans and system that enable the company to aim for zero harm to employees; the motto, ‘everyone-home, every shift’ exemplifies this culture.

The company’s activities will focus on risk assessment, hazard identification and appropriate and reasonable control measures. Management will ensure a programme of continual monitoring and review of health, safety and welfare to provide a work environment that is safe from harm. By nature humans err, therefore a comprehensive program of safe systems of working will be implemented to define safe work procedures for higher risk tasks, this will be linked to the personal risk management ethos adopted and will be used as an educational tool to increase health, safety and welfare awareness.

Effective annual health screening will be undertaken to test cardiovascular, physiological, lung function, hearing, sight and fitness to ensure that all employees are capable of undertaking the tasks prescribed to the job functions they undertake. This will be reinforced with training in personal health and hygiene. There will be specific training in the control of chemicals hazardous to health.

The company will abide by and exceed the requirements of the Health and Safety Authority of Kazakhstan and where appropriate will utilise the suite of guidance and approved codes of practise from the international mining industry.

SECTION 6 METALLURGICAL TESTWORK

6.1 INTRODUCTION

The optimised processing regime has been derived from experience of operating the existing pilot plant which was itself designed after testwork carried out from 2008 onwards. In October 2008, a series of testwork programmes were commissioned to investigate alternative methods of leaching the ore using atmospheric and autoclave leach technologies.

Between November 2008 and January 2010 the National Centre for Complex Processing of Raw Materials, Republic of Kazakhstan, undertook a programme of testwork with the following objectives:

- Undertaking a mineralogical examination of the ore,
- Establishing crushability and grindability of the ore,
- Identifying a selective oxidiser suitable for the recovery of vanadium,
- Minimising acid consumption during autoclave leaching,
- Establishing a flowsheet for the selective adsorption of uranium and molybdenum, vanadium and rare earths,
- Establishing a flowsheet for the recovery of “carbon black” using flotation technology, and
- Reducing levels of aluminium inside barren solution by production of potassium alum.

Both atmospheric and autoclave leaching were investigated as part of the study. A summary of the testwork undertaken is now provided in the following sections.

6.1.1 ATMOSPHERIC LEACHING

High activation energy is required in order to achieve sufficient dissolution of vanadium from patronite and phengite using sulphuric acid. The leach process was therefore carried out at elevated temperatures. The following conditions were used:

- Material ground to -0.1 mm,
- Temperature of sulphidisation of 350 °C,
- Temperature of leach 80 °C,
- Solid-liquid ratio of 1:2, and
- Leach durations of three and six hours.

The sample tested had a high content of acid consuming minerals such as carbonates and chlorites which led to high acid consumption rates. It was noted that during the decomposition of chlorite minerals, silicon hydroxide was formed which caused a sharp increase in the viscosity of the pulp which in turn negatively affected the dissolution of vanadium.

The testwork established that with the low temperature atmospheric leach process it was possible to obtain relatively high vanadium extraction rates. Recoveries in excess of 95 % were achieved with sulphuric acid consumptions of up to 500 kg/t of ore.

However, due to the high consumption of sulphuric acid it was concluded that autoclave leaching would be superior.

6.1.1.1 AUTOCLAVE LEACHING

The decomposition of patronite and phengite were found to be more effective with the application of pressure leach technology. This resulted in improved vanadium dissolution rates in comparison to atmospheric leaching.

The results of the pressure leach tests also demonstrated that sulphuric acid consumption could be reduced by three to four times compared with atmospheric leaching. Nitric acid is added as a catalyst and its consumption depends on the quantity of sulphides within the ore.

Another advantage of the process was that the dissolution of chlorites was significantly reduced. At an operating temperature of 150 °C, with between 15 g/l and 20 g/l of sulphuric acid, it was found that only 5-10 % of the chlorite minerals dissolved. A further benefit of operating the leach at elevated pressure is that aluminium and iron are precipitated as jarosite and alunite resulting in the recovery of some of the acid.

The tests were undertaken on a two-tonne composite sample consisting of a blend of primary and oxide ore types. The blend consisted of 95 % primary ore, 3 % oxide ore and 2 % transitional ore.

The semi-pilot plant tests were undertaken using a 1 m^3 autoclave unit. Tests were undertaken using material ground to 100 µm, with the autoclave operating at a temperature and pressure of 150 °C and 13 atm respectively. A nitric acid strength of 1 % was used and leaching was undertaken for five hours. Using these conditions, the following variables were investigated:

- Sulphuric acid (concentration 10 % to 16 %),
- Temperature (100 °C to 170 °C),
- Leach time (2 to 7 hours),
- Nitric acid (concentration 0 to 2 %), and
- Pulp density (solids-liquid ratios of between 1:0.5 to 1:3).

The optimum conditions found were as follows:

- Grind size of -0.1 mm,
- Solids/liquid ratio of 1:1.7,
- Pressure of 13 atm,
- Temperature of 150 °C,
- Leach time of two hours,

- Sulphuric acid concentration of 12 %, and
- Nitric acid concentration of 1 %.

Using these conditions, a vanadium leach extraction of 95 % was predicted. The filtration of the leach pulp was considered to be favourable due to the coarse grind size of 100µm.

6.1.1.2 SORPTION AND DESORPTION TESTWORK

After the autoclave process, the filtered solutions contain uranium and molybdenum, vanadium and rare-earth metals which can be selectively recovered by sorption technology and subsequently processed to give separate saleable concentrates. A summary of the testwork undertaken on the various metals is now provided.

Solution generated from the autoclave testwork was used for adsorption testwork. The solution assayed 3 g/l V, 0.11 g/l U, 0.17 g/l Mo and 0.1 g/l RE. The solution had a pH of 1.7 and a redox potential of 450 mV. The solution contained 2 g/l NO_3.

Testwork showed that the anion exchange resin Ambersep IRA 920 (supplied by Rohm and Haas) was suitable for the sorption of uranium, molybdenum and vanadium. The maximum capacity of the uranium was found to be 106 kg/t after 48 hours. At pH 2.5 the maximum resin capacity for vanadium was found to be 450 kg/t after two hours.

Prior to adsorption of uranium and molybdenum the solution Eh (redox potential) was modified to 350 mV. Following adsorption of the uranium and molybdenum, the vanadium was then oxidised using H_2O_2 at a redox potential of 1000 mV. Further tests were undertaken using a weakly basic resin (Purolite A-100), but this was found not to be as effective as the Ambersep resin. The Ambersep resin has been commercially used in the uranium industry for a number of years in Kazakhstan.

The desorption of uranium was achieved with the use of ammonia nitrate (100 g/l) and sulphuric acid (10 g/l) while the desorption of vanadium was effective with ammonia nitrate (150 g/l) and ammonia hydroxide (10 g/l).

It was recommended that the uranium be precipitated as polyuranate of ammonia (yellow cake).

6.1.1.3 RECOVERY OF POTASSIUM ALUM

Alum was produced as part of the operation of the pilot plant from the potash and aluminium sulphate contained within the ore. The alum was precipitated in evaporation ponds in order to remove the pulp from the solutions before sorption. This product was analysed and reportedly conformed to GOCT 4329-77.

In the proposed autoclave leach flowsheet, alum will be produced in stoichiometric proportion to potash leached into solution from the ore, with the other products also in stoichiometric proportion. Although any remaining aluminium oxide is sedimented upon neutralisation, the recovery into alum can be increased by the introduction of additional potassium, either recycled from production or purchased. Testwork was undertaken on barren solution containing 9g/l $Al_2(SO_4)_3$ at pH 2.5 and 60 °C and the resulting alum conformed to the GOST standard for standard potassium alum as sold for use in the purification of water and other industries.

Alum is commonly used in China and acceptable terms have been offered by Chinese intermediaries. However, FAR considers that in the long term, the use of alum for water purification is likely to be phased out as easier-to-handle liquids are introduced. FAR therefore plan to refine the potassium alum into alumina and ammonium and potassium sulphates which are commonly sold as fertilizers. The alumina will be sold to Kazakhstan smelters and the fertilizers to local and regional users.

6.1.1.4 PHYSICAL TESTING

A bond ball mill work index test indicates that the ore is moderately hard at 14.8 kwh/t.

6.1.1.5 OPTIMUM CONDITIONS – AUTOCLAVE ROUTE

The results of the tests undertaken at the Kazakhstan National Centre for the Integrated Processing of Raw Materials are summarised as follows:

- Milling size: 0.1 mm (100 %),
- Liquids-to-solids ratio: 1:1.7,
- Pressure: 13 atm,
- Temperature: 150 °C,
- Leaching time: 2 hours,
- H_2SO_4 consumption: 12 %, and
- Nitric acid consumption: 0.5 %.

The dissolution rates of the metals and key operating parameters were found to be:

- Vanadium dissolution: 91 %,
- Uranium and molybdenum dissolution: 100 % and 80 % respectively,
- Rare earth elements dissolution: 75 %,
- H_2SO_4 consumption: 100 kg/t,
- Loss of weight in the slurry: 27 %,
- In the sorption process the efficiencies were 99 % V, 99 % U-Mo and 95 % rare earths,
- In the desorption process the efficiencies were 98 % V, 97 % U-Mo and 94 % rare earths, and
- In the alum crystallisation and precipitation: 60 %.

Based on this data the total predicted recoveries were 89.4 % V, 98.3 % U, 78.6 % Mo, 72.2 % rare earths and 64.7 % alum.

6.2 PRE 2008 METALLURGICAL TESTWORK

The vanadium within the Balausa mineralisation is chemically bound to the host mineralogy and is not liberated by grinding to a fine particle size. The separation of the vanadium requires pressure oxidation.

Due to the fact that the economic ore mineralisation is locked in a sulphide lattice and associated with deleterious minerals such as arsenopyrite and carbon (commonly referred to as double refractory), the first and foremost task was to determine an effective treatment scheme for these ores and thus being able to register the deposit's reserves on the state balance of the Republic of Kazakhstan which requires the definition of optimally effective processing conditions.

During 1998 to 2006, a series of engineering works and heavy industrial testing was undertaken to determine the optimal processing technology. FAR also conducted research and development to determine optimal industrial chemistry and mineral processing characteristics associated with the pilot plant. Lists of the testwork undertaken are contained in APPENDIX B.

6.2.1.1 LIST OF METALLURGICAL PATENTS

The FAR research and development provided much intellectual property. Table 6-1 lists details of the metallurgical patents that FAR has obtained.

Table 6-1: Metallurgical Patents

#	Name	Number
1	Patent RK 19294 Method of sorption treatment of vanadium ore	2006/0725.1 27.06.2006
2	Патент RF №2374344 Method of sorption treatment of vanadium ore	2007119178 24.05.2007
3	Patent RK 19176 Method of sorption treatment of vanadium ore	2006/0726.1 27.06.2006
4	Patent RF №2374345 Method of sorption treatment of vanadium ore	2007119179 24.05.2007
5	Method of treatment of black shales	№2011/0890.1 12.08.2011
6	Method of treatment of black shales	2011147849 24.11.2011
7	Method of autoclave treatment of black shales	№2011/0891.1 от 12.08.2011
8	Method of autoclave treatment of black shales	2011147850 от 24.11.2011
9	Method of treatment of black shales	№2011/0892.1 от 12.08.2011
10	Method of treatment of black shales	2011147852 от 24.11.2011
11	Method of treatment of dominic formations	№2012/0301.1 от 12.03.2012
12	Method to produce alum	№2012/0302.1 от 12.03.2012
13	Ore preparation for ore treatment	№2012/0303.1 от 12.03.2012

Two additional patents relating to the briquetting and use of the carbon-silica tailings for making ferro-silicon are being applied for.

SECTION 7 METALLURGY AND MINERAL PROCESSING

7.1 INTRODUCTION

Much of the world's vanadium production is sourced from vanadium-bearing magnetite found in ultramafic gabbro geological bodies, either as primary production, where the main product is vanadium, or as co-production with steel. Vanadium is mined mostly in South Africa, north-western China, Russia and Brazil. In 2016 these four countries mined more than 90 % of the world's production. Balasausqandiq is a different type of deposit, comprising shale and containing little iron.

Vanadium's current primary use is as a steel hardening agent. When used in the steelmaking process vanadium increases the strength and corrosion resistance of steel. The demand for vanadium, therefore, is strongly dependent on steel production rates. Historically the price of vanadium has shown periodic spikes, followed by longer periods of lower prices.

Vanadium producers can be categorised into primary, co-product, and secondary.

The dominant source of vanadium is co-product production. Vanadium is recovered as a co-product from steelmaking during the 'two-step' steel-making process where the feed is vanadium-bearing magnetite. In 2016, vanadium from co-product sources accounts for about 71% of global production.

Primary sources of vanadium are operations that mine rock specifically for the purpose of extracting vanadium. Primary sources accounted in 2016 for 17 % of the global production. Secondary, recycled sources of vanadium account for 12 % of world vanadium production.

7.2 MINERALOGY

The Balasausqandiq deposit is confined to a metamorphosed schisty stratum layer ranging from a primary black carbonaceous siliceous rock to oxidized quartzite-roscoelite schists.

Two main mineralised metal bearing rock types are present:

- Oxidised rock (nominally called 'oxide ore') consisting of quartz and roscoelite. This is found in the weathered surface layer of the deposit to a depth of between 25 m and 30 m; and
- Primary mineralised metal bearing rock (nominally called 'primary ore') consisting of un-weathered black quartz carbonaceous schists found directly below the oxidised layer.

Vanadium is found mainly in the minerals of patronite (V_2S_4) and phengite (BaV_2O_4) which account for 55 % and 35 % of the total vanadium in the ore respectively. A smaller quantity (10 %) is associated with micas and uranium-bearing oxide minerals.

The ore also contains organic substances such as anthraxolite which has high carbon content (96-98 %). Anthraxolite is the name given to anthracite-like materials that occur in veins and as disseminated masses in sedimentary rocks in which commonly the primary vanadium mineral montrozeite may be preserved, as it is within the oxide zone at Balausa.

The ore body also contains aluminium, uranium and molybdenum, rare earth elements and potassium which are all considered to be of economic value.

The chemical composition of the primary vanadium ore type is shown in Table 7-1.

Table 7-1: Chemical Composition of Primary Ore

Compound	Composition %
SiO_2	72.0
C	14.0
Fe_2O_3	5.5
Al_2O_3	4.7
CaO	3.5
V_2O_5	0.67
BaO	0.9
P_2O_5	0.7
MgO	0.6
MnO	0.3
Others	0.7

As detailed in Section 4 Mineral Resource estimations indicate that the deposit contains approximately 5 % of oxide ore and 95 % of primary ore. Both types of ore are amenable to the same processing method.

7.3 MINERAL PROCESS PLANT

In 2012, a flow sheet for the processing by pressure oxidation of vanadium ore from Balausa was completed by the Kazakh National Center for Complex Processing of Mineral Raw Materials, adapted from work by the research institute VNIIKHT (a subsidiary of Rosatom, of the Russian Federation). From this work, design data for the operation of an experimental pilot plant with a processing capacity of 15,000tpa was developed. Detail engineering of the plant and autoclave was carried out by LLC Interfos (Moscow).

FAR completed the construction of the pilot plant and in 2013 began operations to prove the technology and optimize the design for the planned 1Mtpa expansion and, in particular, to:

- Improve the machinery and equipment layout to reduce the capital costs of the proposed 1Mtpa plant
- Achieve operational stability to provide steady state operations of the future enterprise

- Optimize the technological parameters (pressure, temperature, acid consumption, etc.) of the autoclave processing of vanadium ore to reduce operating costs

After a programme of testing and optimisation, and the implementation of several amendments to simplify and improve the operation, the final plant configuration, operating parameters and performance were finalised as further described in this section. In section 7.4 the adaptations made to the pilot plant to treat purchased concentrates will be described.

The pilot plant was designed with a capacity of 15,000 tonnes of ore per annum which was not sufficient scale at the time and the prevailing prices to be considered a commercial operation. FAR therefore adapted the plant to treat higher grade purchased concentrates. As proof of feasibility, the plant was adapted with a minimum of changes so that it can now produce around 144 tonnes of vanadium pentoxide equivalent per year. FAR now proposes to increase output tenfold to around 1,500 tonnes per annum. In parallel, FAR proposes to build a separate new plant to treat 1Mtpa of mined ore using the processing scheme optimised in the pilot plant.

The current processing plant produces only vanadium in the form of ammonium metavanadate, with no by-products. Once this is expanded, it will be capable of converting the production to vanadium pentoxide or to electrolyte if there is demand for vanadium flow batteries. Future operations treating mined ore will produce vanadium in its various forms as well as a uranium-molybdenum bulk concentrate, a rare earth concentrate, potassium alum and a carbon-rich silica product for use as a smelting flux or a “carbon black” concentrate which can be sold to rubber manufacturers. The Phase 1 operation will additionally have the capability to convert vanadium pentoxide into ferro-vanadium which is the form that the steel industry generally requires. Equipment for the conversion of AMV into vanadium pentoxide (by heating and capture of off-gas) and ferro-vanadium (by *aluminothermy*) has been procured and was tested on FAR produced AMV at the manufacturer’s site.

By early June 2014 the process testing at the pilot plant had been completed, the process parameters optimised and required modifications had been completed. A post-implementation inspection and report was organised by the developers of the process VNIIKHT who reported on the success of the plant and of the significant improvements that had been made by FAR to the initial technology. Minor changes planned by FAR that might be made in the scale-up to 1Mtpa to improve efficiency were confirmed.

The process is covered by 13 patents but also involves considerable know-how and for the purposes of confidentiality, the following process description is abbreviated.

The following process description refers to the pilot plant, which is practically identical to the planned full-scale process plant. Some mechanical engineering improvements will be included at full-scale and in particular, it was discovered that the highly sophisticated test autoclave is not necessary and a much simpler vertical autoclave without mixing or oxygen will suffice. Some additional testwork of such an autoclave will be carried out as part of the final design process. Other changes will be to the filtration process and dry milling will be considered instead of wet milling to reduce water demand.

7.3.1 PROCESS DESCRIPTION

The optimised final process plant flowsheet for the pilot plant is set out in Figure 7-1.

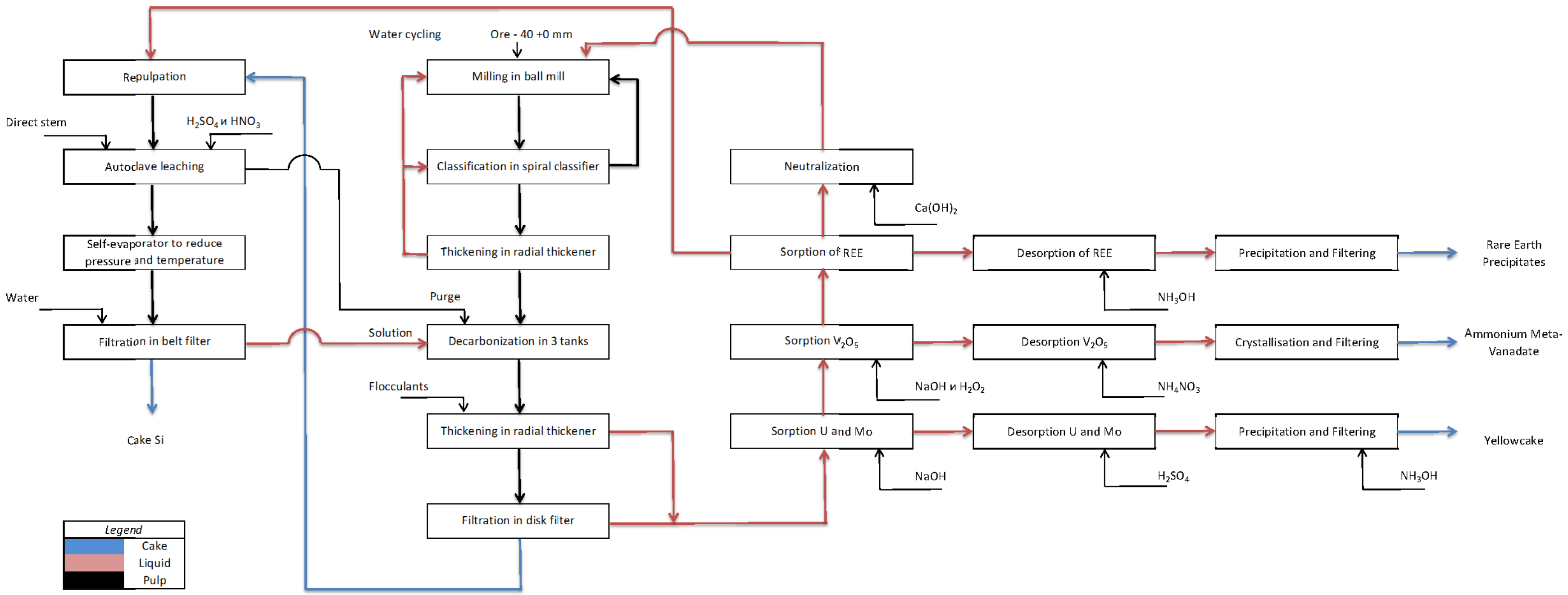


Figure 7-1: Process Block Diagram

7.3.2 CRUSHING AND GRINDING

Run-of-mine (ROM) ore will be crushed from 350 mm to 20 mm using a standard crushing flowsheet. is

Figure 7-2: Mill and Gravity Classifier

Output from the mill is directed to a spiral classifier (circulating load 100 %), milled ore -0.2 mm (100 %) from the spiral classifier is sent to a thickener of 6 m diameter which provides acceptable thickening without the use of flocculants. The mill and classifier are illustrated in Figure 7-2.

7.3.3 DE-CARBONISATION

After settling by radial thickener, the dense slurry is transfered to the decarbonising process using a diaphragm pump. Decarbonisation is carried out in three tanks (i.e. acid wash reactors) with two bladed stirrer mechanisims with rotational stiring speed of 240 revolutions/minute.

Acid solution from the belt filter is recirculated (70 g/l) to the first reactor with the addition of sulfuric acid and a temperature of 60 - 70 °C.

The reactor tanks for decarbonisation are brick lined and steam heated, with the injection point being the first reactor.

The optimum regimes of decarbonisation are shown in Table 7-2.

Table 7-2: Optimum Regimes of Decarbonization

Time in process [h]	Density of slurry [kg/m^3]	Screening less	Recovery V_2O_5 [%]	Recovery U [%]
1.9	1,211.2	0.2 mm	25.9	89

During the neutralisation stage, acid consuming minerals are reacted with sulphuric acid in order to prevent them consuming acid during the autoclave process. The acid used during the neutralisation stage is sourced from the autoclave circuit. Acid generated during the autoclave process, which would otherwise require further treatment prior to disposal, is consumed during the neutralisation stage. During the process carbon dioxide is produced.

7.3.4 THICKENING AND FILTRATION

After decarbonisation, the pulp is sent to a second radial thickener with an area of 4.9 m^2.

After settling, the pulp goes to a disc filter (porous ceramic) with a filtration area of 12 m^2. The cake from the disc filter is removed to the autoclave by conveyor (moisture 7-10 %). The liquid goes to desorption.

The thickener and disc filter are illustrated in Figure 7-3 and Figure 7-4 respectively.

Figure 7-3: Thickener

Figure 7-4: Disc Filter

7.3.5 AUTOCLAVE LEACHING CIRCUIT

The autoclave circuit consists of a single autoclave operating with a self-evaporator to allow for the reduction in pressure without boiling of the fluids. Cake from the disc filter is first repulped before being pumped into the autoclave with high-pressure peristaltic pumps. Acid mixture is also pumped into the autoclave.

The autoclave was designed to work in different modes with variable pressure and temperature in order to test and optimise the regimes. Currently, a pressure of 0.8 Mpa is maintained. Autoclave heating is by steam at 270 to 320 °C, maintaining the temperature in the autoclave at between 125 to 150 °C. Pulp density is 1,500 kg/m^3.

After the autoclave, pulp density is maintained at 1,500 kg/m^3 and the output temperature of the slurry is 85-90 °C. After the self-evaporator, the pulp is cooled by a pipe-in-pipe heat exchanger before passing to the belt filter. Because the density of the pulp is enough for filtration, there is no need for further thickening.

Pulp from the autoclave is passed to the belt filter with a temperature of 70-80 °C. The filtration area is 16 m^2. The cake is passed out of the plant by conveyor as the product - carbon-silica flux. The solutions pass to the decarbonization process for neutralization and then sorption processes.Table 7-3 outlines the optimum regime of autoclave leaching. The autoclave unit is shown in Figure 7-5

Table 7-3: Optimum Regimes of Autoclave Leaching

Time in autoclave (h)	Pressure [kg/cm^{2}]	T, ^{0}C	Density (kg/m3)	Recovery V (%)	Screening less
1.6	5.0	130.0	1,430	91	0.2 mm

Figure 7-5: Autoclave

7.3.6 ADSORPTION CIRCUIT

The recovery of vanadium, uranium, molybdenum and rare earths is undertaken using three separate adsorption circuits. Solution from the autoclave circuit is initially fed to the uranium-molybdenum adsorption circuit with the exit solution being fed to the vanadium circuit. The solution exiting the vanadium circuit is fed to the rare earth adsorption circuit while the solution exiting this stage is sent to the “alum sedimentation” stage. Figure 7-6 shows the current adsorption circuit arrangement.

Figure 7-6: Adsorption Circuit

After the adsorption process there is, for each of the three process streams, subsequent desorption processes, the process steps for each mineral assemblage are described separately.

7.3.6.1 URANIUM AND MOLYBDENUM ADSORPTION

Pregnant solution is fed through two columns, operated in parallel, containing the resin “Ambersep 920”. The solution is fed until the resin loading is completed at which point the feed solution will be diverted to the second column. Uranium and molybdenum is stripped from the resin using a solution containing 1 % H_2SO_4.

7.3.6.2 VANADIUM ADSORPTION

The pregnant liquor from the uranium column is fed into three resin columns operated in series. The columns are operated in a counter current arrangement with fresh resin being loaded into the last column and transported from column to column until it exits the first column. The solution flows in the opposite direction to that of the resin.

In order to separate vanadium from iron (Fe^{3+}) step-by-step pH adjustment is undertaken between each of the adsorption columns. Hydrogen peroxide (H_2O_2) is also added to the process in order to convert the vanadium from oxidation state V (IV) to V (V) so that it is amenable to adsorption onto the resin.

In the first column the pH is maintained at pH 1.8 with H_2O_2 being added. The exit solutions from each column flow into holding tanks where the pH is adjusted accordingly using sodium carbonate (Na_2CO_3).

The overall contact time of the solution with the resin is less than one hour in order to prevent the reduction of V (V) to V (IV). The loaded resin contains between 400 and 450 kg of V_2O_5 per tonne of resin.

7.3.6.3 RARE EARTHS ADSORPTION

The solution leaving the vanadium adsorption circuit is transferred to the rare earth adsorption circuit. The loaded resin contains approximately 38.2 kg of rare earths per tonne. Prior to stripping the rare earth from the resin, the solution flow is diverted to an alternative column.

7.3.6.4 SEDIMENTATION OF ALUM

Potassium and aluminium in the ore are precipitated as potassium alum and separated from the solution using a thickener. The thickener overflow reports to the autoclave circuit. Additional aluminium oxide is precipitated after the neutralisation circuit. It is envisaged that higher recovery of alum can be obtained by adding further (purchased or recycled) potassium to the circuit.

7.3.6.5 CARBON - SILICA

The filter cake generated from the autoclave circuit is fed to a washing circuit for the removal of residual sulphuric acid and filtered. The resulting carbon-silica is currently removed to tailings as it has been derived from the upper oxidised ore where the carbon level is low. In the 1Mtpa circuit treating high carbon ore, the carbon-silica will be further treated to make saleable products according to demand. Market studies and test-work have been carried out to determine the market and suitability of the product for briquetting and use for smelting ferro-silicon. The market and product study found that briquetting could be carried out using pressure alone, resulting in high quality briquettes which could withstand transport shocks and smelting heat tests. The carbon and silica were found to be of an ideal form to substitute for 100% of the silica and approximately 50% of the carbon requirement in smelting arrangements. Further market studies will underpin the optimal product specifications. See also under 7.6.5.

7.3.6.6 URANIUM DESORPTION AND PROCESSING

Loaded resin containing uranium and molybdenum (up to 40 kg/t resin) is washed with an ammonium nitrate and sulphuric acid solution (NH_4NO_3 100g/l + H_2SO_4 10g/l). Desorption processing takes five hours.

The uranium and molybdenum within the washed solution is then precipitated with ammonia, aqua ammonia, caustic soda or ammonium carbonate solutions to produce ‘yellow cake’.

7.3.6.7 VANADIUM DESORPTION AND PROCESSING

The loaded resin contains up to 400-500 kg/t of V_2O. Vanadium desorption is undertaken with a solution of ammonia nitrate (150-200 g/l) at a pH of between 7.5 and 8.5 and at a temperature of between 50 to 60 °C. The material produced from the desorption unit is directed to the resin reloading unit and four volumes of the column effluent are recirculated until the ammonium meta-vanadate (MVA) is fully recovered.

The resin and the washed solution are mechanically screened in order to separate the MVA crystal slurry from the resin. The resin is then reloaded into the column and the MVA slurry is sent for filtration. At the filtration stage, ammonium meta-vanadate is separated from the solution and directed to the recrystallisation processing stage. The barren solution is used for making desorption solution.

Recrystallisation of the MVA is undertaken in water at 80-90 °C and with a V_2O_5 concentration of 20 to 25 g/l at pH 7.0-7.5. The barren solution is re-circulated and the MVA crystals are directed to the thermal breakdown circuit.

Thermal breakdown is done in a rotary furnace at a temperature of 550 °C. The vanadium oxide powder is then bagged ready for shipment.

7.3.6.8 DESORPTION OF RARE EARTH METAL.

The loaded resin is washed with ammonia solution at pH 7.5-8.5. The resin and the washed solution are separated from one another using a screen. The washed solution is filtered in order to remove rare earth precipitates. The precipitated residue contains 91 % rare earth metal oxides.

7.3.6.9 NEUTRALISATION

The process liquor is recycled and used again in the hydrometallurgical process. The recycling circuit involves neutralisation and clarification of the liquors and is attained by mixing with quick lime (Size - 1 mm, flow rate - 12 kg/t of ore). The neutralisation of the acidic solution is conducted in a tank with mechanical stirring whilst simultaneously feeding lime milk. The process results in a liquor with pH 7.5-8.0. The batch processing time is 2-3 hours.

The pulp from the clarification process is pumped to a ‘mud’ settling impoundement and held for 3-5 days. During this period the precipitate (slurry) settles with gypsum $CaSO_4 \cdot 1.5H_2O$, calcium phosphate $Ca_3(PO_4)_2$, iron and aluminum hydrates being the main solid products. The clarified solution (with concentration: Fe 0.01 g/l; Al 0.0027 g/l, NO_3 4.94 g/l, K 1.37 g/l and salinity 10.91 g/l) is then decanted and is used in the milling and classification stages.

7.4 METALLURGICAL RECOVERY DURING OPERATIONS

Recovery of vanadium occurs principally in two stages of the process, in decarbonisation and in the autoclave leaching circuit.

Overall recovery into solution is 93.3 %. The vanadium grade is 0.78 % V_2O_5. The average grade in the tailings is 0.052 % V_2O_5. Average grade after decarbonisation (before autoclave leaching) is 0.58 % V_2O_5.

7.4.1 RECOVERY DURING SORPTION-DESORPTION

Further small losses currently take place as vanadium not recovered from solutions in the first pass across the resin can be lost when surplus water is neutralized and returned to the process water storage dam. However, it is proposed that the expansion to 1Mtpa will use dry milling, thus reducing the water brought into the process and obviating the need to return surplus water. This will result in 100 % of the solutions remaining in cycle so what is not recovered in the first pass across the resin will be captured on subsequent passes. Losses will be small as they will be limited to losses through spillage etc.

The overall recovery of vanadium is therefore predicted to be over 93 % from the 1Mtpa plant.

7.5 ENGINEERING DESIGN FOR 1MTPA EXPANSION

An engineering design has been undertaken by FAR for the treatment of 1Mtpa of mined ore. The main processing sections will be the same as in the current experimental plant, consisting of:

- Crushing and grinding circuit
- Hydrometallurgical facility consisting of:
    - Autoclave circuit
    - Adsorption circuit
    - Desorption circuit
    - Crystallisation and precipitation circuits
- Product handling facilities

- Waste impoundment and management facility

It is proposed that a sulphuric acid plant will be built to make sulphuric acid for the processing facility from sulphur procured from desulphurisation plants for oil production in western Kazakhstan. This sulphur is currently being stored by the producers and is attracting environmental penalties. Waste heat from the acid plant will be used for the generation of steam for the purpose of heating the autoclave unit and the generation of power. However, current prices of sulphuric acid are low and if they remain at this level, the directors of FAR propose to defer the construction of sulphur burner until Phase 2 construction starts.

The process plant will be similar to the existing pilot plant but two significant improvements will be made. Firstly, the use of dry milling will eliminate the requirement for the neutralisation stage and waste water recycle, thus reducing reagent costs and vanadium losses. Secondly, the current filtration systems have proven to be inefficient and the final belt filter will be replaced by disc filters.

7.6 PRODUCT SPECIFICATION

7.6.1 GENERAL

Sales of vanadium have taken place to customers in China, Russia and Taiwan. FAR has acquired commercially available studies on the world vanadium market and has investigated the potential users of the various by-products, their markets and product specification requirements.

The Kazakhstan National Centre for the Integrated Processing of Raw Materials stated that the products conformed to the following specifications:

- Ammonia metavanadate – compliant with standard 38088316–01–2007 (content of V_2O_5 75-77 %);
- Potassium alumen – compliant with standard GOST 4329-77 (content of the basic substance K_2SO_4*Al_2 $(SO_4)_3$*$12H_2O$ at least 98 %);
- Concentrate of rare-earth metals compliant with GOST 25702.0-83* (content of the total rare-earth elements 92 %);
- Uranium concentrate (yellow cake): quality requirements for such products are determined by the agreements of the supplier and the consumer (content of the basic substance 45 % U); and
- Carbon (C) industrial composite mix – compliant with GOST ACTM H-330 (content of 30 % C).

7.6.2 VANADIUM PENTOXIDE

The product specification for the ammonia metavanadate (Standard # ST TOO 38088316-01-2007, UDK MKS 71.060.50) is:

- Grade 65-75 %;
- Moisture 12-15 %;
- P not more than 0.05-0.08 %;
- Fe not more than 0.10-0.15 %;
- S not more than 0.05-0.10 %; and
- U-238 not more than 0.005 % to hold exposition radioactivity in norm.

The grade of metavanadate and impurities produced from the pilot plant operations are as shown in Table 7-4, together with the grade of vanadium pentoxide produced from it. The grade of the product is therefore of good quality. Further upgrade in quality can be achieved by re-crystallisation using purified water if required.

Table 7-4: Grade of Metavanadate and Impurities Produced from Operations

	Metavanadate [%]	V_2O_5 derived [%]
$V_2O_{5/4}$	75.9	98.8
V_2O_4	-	1.24
TiO_2	<0.010	<0.010
Mn	<0.005	<0.005
CaO	0.092	0.097
Fe	0.01	0.025
Si	0.1	0.1
P	0.017	0.024
MgO	0.23	0.24
Cr	<0.005	<0.005
Ni	<0.001	<0.001
Mo	<0.005	<0.005
Na	0.023	0.027
K	0.017	0.017
Pb	<0.005	<0.005
S	<0.005	0.014

7.6.3 URANIUM AND MOLYBDENUM

FAR intends to sell its products to KazAtomProm which has suitable processing facilities and has a statutory first right to purchase uranium products in Kazakhstan.

7.6.4 RARE EARTHS

The rare earth product from Balausa conforms to a Kazakhstan standard. Sales are likely to be made to the Ulba processing plant in Kazakhstan but currently FAR ascribes no value in the cash flow to this by-product as a result of uncertainty over market demand and low prices. In practice, some value is likely to be derived.

7.6.5 "CARBON BLACK" AND FLUX

The ore processing at Balausa involves crushing and milling to <0.2 mm followed by removal of the carbonates and then autoclave sulphuric acid leaching of the metallic content.

After these processes, the remaining material is comprised of around 76.4 % silica, 18.7 % organic material and 4.9 % other (non-deleterious) components. The organic material is composed of 88.7 % carbon, 1.1 % hydrogen, 8.5 % nitrogen and 1.6 % sulphur.

The organic material has unusual characteristics that make it suitable for a number of high-value uses. Physically and chemically, it is similar to carbon black, having a high surface area of 224 m^2/g and calorific value from 7,000 to 9,000 kcal/kg. The silica can also be used for some specialised purposes as a result of being milled and free from deleterious constituents, particularly aluminium.

The organic material cannot easily be separated from the silica by physical means as the two are physically bound together. However, using flotation technology the organic material can be upgraded to around 30 % with a recovery into concentrate of around 45 %, leaving the depleted-carbon tailings for other uses that do not require high-carbon. Fluoric acid or sodium hydroxide leaching can be used to remove the silica, but no detailed analysis of this has yet been undertaken.

FAR has focused on finding uses for the mix of organic material and silica ("carbon-silica tailings"), either at the unconcentrated grade of around 18 % organic or the concentrate grade of 30 % organic, as well as for the remaining low-carbon silica after concentration or pyrolysis of the carbon to make gas or diesel.

Uses for the concentrated tailings include use as a filler in the production of rubber (a carbon black substitute), use as a flux in the production of ferro-silicon (particularly low-aluminium or high-purity ferro-silicon), pyrolysis to produce gas or diesel and the desalination of land. Uses for the un-concentrated or depleted-carbon tailings include pyrolysis, the making of acid-proof or sulphur concrete, and as an additive to make high strength concretes.

The annual production of the silica-carbon tailings will be some 820,000 tonnes per annum from the first stage development of 1Mtpa expansion and around 3.3Mtpa after the planned expansion to 4Mtpa of ore treated.

FAR's base case plan is to sell the leach residue as a smelting flux to ferro-silicon producers. Alternatively, the leach residue can be concentrated and sold as a "carbon black" substitute.

FAR advise that in April 2010 the Combustion Institute tested the tailings from the autoclave testwork for suitability for use in ferro-alloy production. Their conclusion was that the carbon exceeded the properties of coke in terms of calorific value and porosity, and the impurity levels in the silicon were low. They concluded that it was highly suitable after pelletisation or briquetting as a charge for the production of ferro-alloys, with pricing likely to be based on the equivalent price paid by producers for coke and silica.

The Karaganda Chemical-Metallurgical Institute also evaluated a sample of the material to determine its suitability for other applications and has reported that the form of silica is amorphous and the quality of carbon high in calorific content and porosity. They also concluded that it is therefore ideal for use in the manufacture of silica-containing ferro-alloys and ferro-silica.

Studies undertaken by the National Center for Complex Processing of the Republic of Kazakhstan Minerals have shown that FAR’s “carbon black” product can be used as a substitute for carbon black and is recognised as a GOST standard (as tested in 2008 by the Central Laboratory for Certification Testwork of Construction Materials, Almaty).

Carbon black is man-made product (made by combustion of hydrocarbon products under controlled conditions). Therefore, the term carbon black is not completely correct with reference to the current project, but it can be sold as a substitute.

Carbon black is used as a pigment and reinforcer in rubber and plastic products. Various grades of concentrate between 30 % and 90 % (generated from flotation testing) have been tested by the Central Laboratory for the certification of construction materials. The 90 % concentrate meets the requirements of the carbon-black standard GOST 7885-86, ASTM D1765 (equivalent to the western N330) and ISO 1867-86 showing that the carbon content is comparable to carbon black.

Flotation tests were undertaken on leach tailings having a carbon content of about 14 % at a particle size of 80 % passing 74 µm. It was recommended that the flotation process utilises a roughing and scavenger stage at reagent dosages of 400 g/t kerosene, 300 g/t amine and 250 g/t T-80 frother. An amine collector was found necessary for the recovery of carbon that was associated with quartz.

Carbon black is usually mixed with silica when used in the manufacture of rubber and, since the silica in the tailings is of high purity and small particle size, it was shown to be suited for this application.

FAR has tested a concentrate assaying 30 % C in the manufacture of rubber products at the Issyk Technical Rubber Factory. The rubber products were tested for stretch and return, tear resistance, wear and strength and were found to have good properties.

7.6.6 POTASSIUM ALUM

Potassium alum is used in the purification of water, glue spreading, papermaking, painting, dyeing, leather producing, fibre board processing, rubber production and the fermentation of foodstuffs.

The product specification for alum (to GOST 4329-77) is shown in Table 7-5.

Whilst there is currently a good market for potassium alum in China, FAR believes this market may diminish in future as more easily handled alternatives are introduced for water purification as they have in Europe. FAR therefore plans to further refine the alum into alumina and potassium and ammonium sulphate fertilizers.

Table 7-5: Physico-Chemical Indices According to GOST 4329-77

Indices	Standard	
	Pure for analysis	Pure
Mass fraction of potassium alum $AlK(SO_4)_2 \cdot 12 H_2O$, %, not more than	98.0-100.0	Not less than 96.0 %
Mass fraction of insoluble substances, %, not more than	0.005	0.010
Mass fraction of NH_4, %, not more than	0.005	0.010
Mass fraction of chlorides (CL), %, not more than	0.0008	0.0040
Mass fraction of Fe, %, not more than	0.0010	0.0020
Mass fraction of heavy metals (Pb), %, not more than	0.0010	0.0020
Mass fraction of As, %, not more than	0.00005	0.00010
Mass fraction of Na, %, not more than	0.02	Not standardized
pH of preparation solution with 5 % mass fraction	3.0	3.0

7.6.7 EXISTING PROCESSING OPERATION

The pilot plant as described above was modified in 2015/16 to treat purchased concentrates which contain around 4 % vanadium, of which around 1.5 % is extractable in soluble form. The material comes in milled form so the crushing and milling parts of the operation are not required, and the leaching is carried out at atmospheric pressure so the autoclave is also not used. As there is no by-product content, the sorption and adsorption systems formerly used to recover uranium, molybdenum and rare earth metals have been adapted to recover the greater quantities of vanadium that are now produced. The first steps have been taken to convert this plant to treat a wide variety of secondary sources of vanadium including spent catalysts and vanadium-containing ashes.

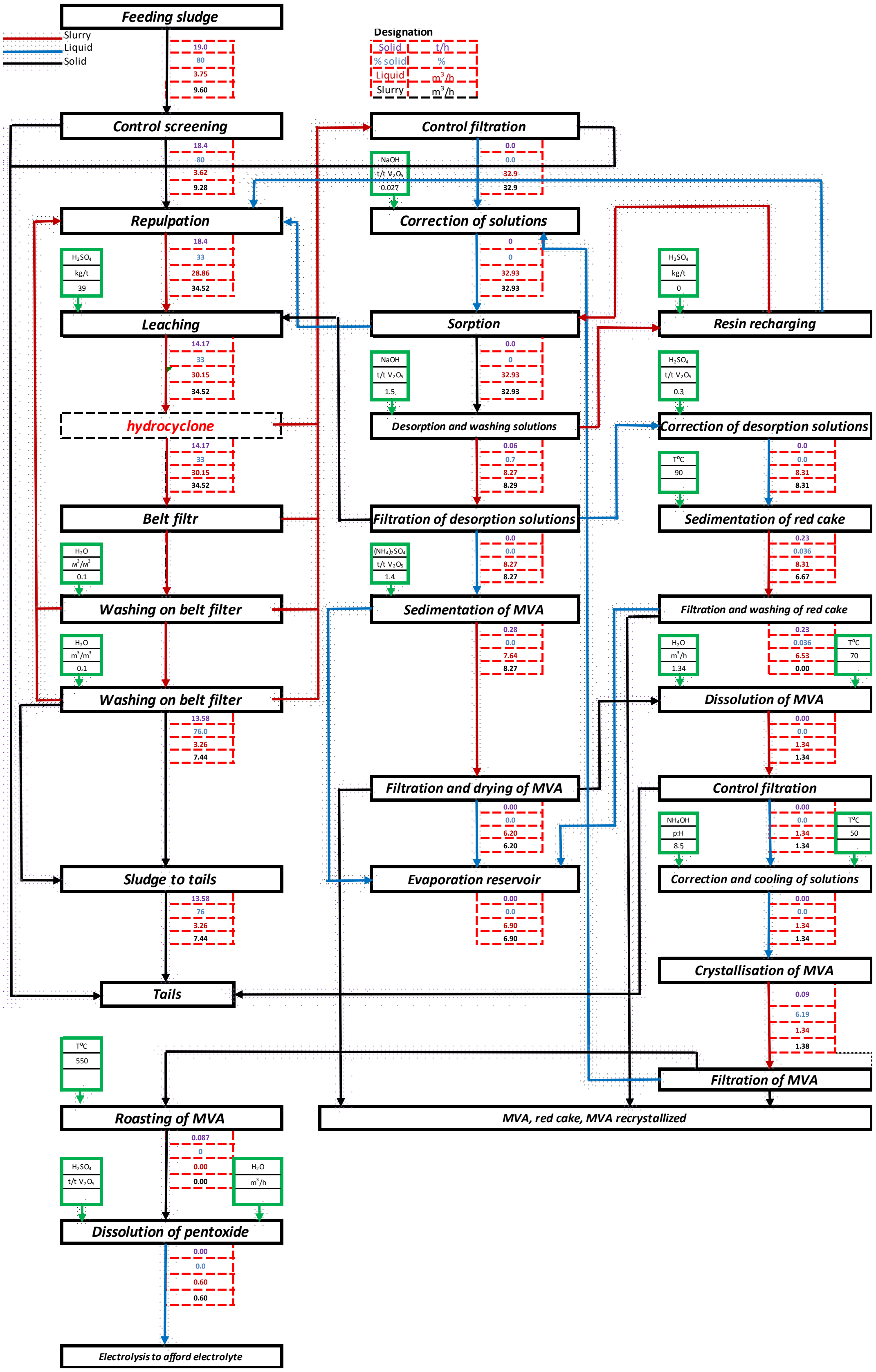


Figure 7-7: Process block flow diagram

SECTION 8 INFRASTRUCTURE

8.1 INTRODUCTION

The infrastructure at the current mine is sufficient for the existing operation which comprises the existing processing operation and small-scale mining. There are plant, engineering, administration and workshop buildings, waste storage facilities and impoundments, a water storage dam, and rock crushing and screening machinery for the sale of aggregates. Within the building suite there is also a chemical assay laboratory, security building and administration building and welfare facilities.

The industrial complex is heated by a central boiler using regionally sourced coal. There are good road and railway links to the region and roadway to the site. The operation produces its own construction aggregates; cement and lime materials are sourced locally.

8.2 RAILWAY SIDING

At Shieli, 70 km from the plant, there is a railway station along the main East-West road and rail transit linking the Russian Baltic, through Kyzylorda, Shymkent and Almaty, into China and on to the East coast. The business plan does not require an extension to the railway to the mine site for the 1Mtpa expansion as input and product volumes are relatively low. There is a possibility to connect the mine to the railway from the other direction when the ferro-silicon operation (which is not part of the expansion proposed in this report) is started.

FAR is currently examining several options to acquire railway sidings in Shieli, together with associated storage buildings and handling equipment. These sidings will be used for the transfer of reagents, materials and capital items from rail to road for onward transfer to site and for the transfer of products from truck to rail.

8.3 OFFICES / BUILDINGS / LABORATORY

These facilities are adequate for the current operations and will be partially upgraded as part of the expansion plans. The upgrade will include high voltage electrical reticulation, the fabric of the buildings and general refurbishment. The company has its main office in the town of Shieli. Figure 8-1 shows the current office building at the factory site.

Figure 8-1: Process Building with Office Block in Foreground

8.4 WORKSHOP AND STORES

The site has modest workshop and operating stores facility which will be upgraded as part of the 1Mtpa expansion. The upgrade will include extra storage racking and the purchase of additional tools.

8.5 DIESEL SUPPLY

There is a well-established diesel storage facility at the Balausa mine (as shown in Figure 8-2). This is adequate for current and future requirements of the business.

Figure 8-2: Diesel Storage

8.6 ELECTRICITY SUPPLY

FAR has built a 22 km overhead power-line from Aksumbe to site which connects with the supply to Aksumbe from Karatau. This line is rated at 35 kV but is currently supplying only 10 kV which is sufficient for the current processing plant operations. The line from Akusmbe to Karatau is owned by the local electricity supply company and is in very poor condition, having wooden poles and is subject to repeated failures, especially after strong winter winds. A separate 110 kV line is immediately adjacent to the Balausa site and FAR plans to connect to a suitable connection point some 2 km distant. This line can provide sufficient reliable and much cheaper power with enough capacity for the expansion of current processing facilities. The line to Aksumbe will continue to be available as back up. Some further upgrade of the 110 kV line will be required to provide power for the Phase 1 development.

When it is constructed, waste heat from the sulphur burner will produce additional power as well as useful heat for steam.

The 10 kV currently used in the pilot plant is reduced to 380 V by on-site transformers for use in the offices and laboratory.

There is a 100 kW diesel generator on site to provide back up to the mains power.

8.7 WATER SUPPLY

8.7.1 CAMP WATER

There is a natural spring near to the accommodation supplying piped water for washing, showers etc. Although of high quality, it is not approved for drinking so bottled water is used for drinking purposes.

8.7.2 PROCESS WATER

Water consumption in the pilot plant is kept to a minimum through recycling, but for top-up purposes there are two sources:

- A bore-hole supplying up to 6 m^3/h (down to about 2 m^3/h in summer),
- A water collection reservoir that fills with the snow melt in spring and other precipitation.

Underground and surface water is plentiful and FAR plans to drill for more underground water to meet the needs of the expanded plant.

8.8 CAMP

The camp consists of two houses to accommodate personnel and an outside lavatory. It is capable of accommodating up to 77 people during the summer which will be busiest time of the year. This is adequate for the current operation. For the expansion a new accommodation block is to be built, the foundations of which are already in existence. This will be further developed at the time of the Phase 1 development.

Electricity is provided to the camp via a 10 kV overhead line that has been run out from the mine distribution transformer and then broken down to 380 V. Water is usually supplied daily as bottled water which is brought to site from Shieli.

8.9 ROAD

All materials and equipment will be transported to site by road. The road between Shieli and the mine is maintained by KazAtomprom and is currently in good condition, having been resurfaced in 2013 and is well-maintained.

SECTION 9 ENVIRONMENTAL STUDIES AND SOCIAL & COMMUNITY IMPACT

9.1 ENVIRONMENTAL LEGISLATIVE FRAMEWORK

9.1.1 INTRODUCTION

The development of any industrial enterprise requires a series of permissions from various regulatory organisations in the state of Kazakhstan. The common theme throughout is a satisfactory consideration of the environment and that the industrial design of the proposed business will allow the function of the business to be sustainable and not be disruptive in its integration with the local environment.

9.1.2 BACKGROUND TO THE LEGISLATIVE OBLIGATIONS

Details of the requirements of Kazakh legislation are provided in this section.

The Republic of Kazakhstan achieved independence on the 16 December 1991 following the breakup of the Soviet Union. The country is a democratic presidential republic which adopted a Constitution in 1995 stipulating the fundamental principles of the governance of the Republic and the rights of its citizens. The Constitution has evolved through a number of reviews and amendments but the political and economic system has demonstrated its stability and development. The Republic’s other laws and regulations are derived through the Constitution.

In 2007, the Ecology Code came into force in Kazakhstan and it combined all of the main regulations related to ecology issues into one document, including duties and responsibilities of the authorized state bodies and of nature users as well as requirements for the protection of natural resources.

The Ecology Code requires that development activities and projects, which have direct or indirect impact on the environment and socio-economic issues, are subject to the Environment and Social Impact Assessment (ESIA) process. The ESIA process includes the evaluation of environmental and social impacts over the life of the project and requires that studies should establish a baseline prior to commencement of works with regular updates and reviews throughout the project’s life.

Monitoring of environmental and social impacts is required and is regulated by the terms of the licence and by periodic regulatory inspection by the authorities to ensure compliance. Emissions and other limits are set for a number of parameters such as water quality, air quality, soil quality, dust, noise and vibration. The regulatory authorities expect and require full compliance with the achievement and fulfilment of all environmental limiting indices and licence requirements and any non-compliance or violations may lead to severe consequences, including cessation of operations and judicial penalty.

9.1.3 ENGAGEMENT REQUIREMENTS

The government of Kazakhstan has ratified the UNECE Convention on Access to Information, Public Participation in Decision-making and Access to Justice in Environmental Matters, also known as the Aarhus Convention. The Aarhus Convention requires the government to grant the public rights regarding access to information on the environment, including information on the environmental impact of corporate activities, access to which is available only to the relevant environmental authority.

According to legal regulations, information disclosure and dissemination, as well as public consultation, are a part of the development process, especially if the project impacts the environment.

The following legislative acts relate to public participation in decision making within Kazakhstan:

- Environmental Code of RK No. 212-III of 9 January 2007 (as amended on 17 July 2009);
- Instruction of Environmental Impact Assessment, Conduct of Proposed Economical or Other Activities during Development of Pre-planning, Planning, Pre-design and Design Documentation, approved by Order of the Minister of Environmental Protection of RK No.204-p of 28 June 2007;
- Rules on the conduct of Public Hearings, approved by Order of Minister of Environmental, Protection of RK No.135-p of 07 May 2007;
- Rules on Access to Environmental Information Relevant to Environmental Impact Assessment (EIA) Procedure and Decision-Making Process on Proposed Economical and Other Activities, approved by Order of the Minister of Environmental Protection of RK No.233-p of 25 July 2007;
- Rules on the Conduct of Public Hearings while Considering Application for Approval or Change of Tariffs (Prices, Rates) of Entities which are Natural Monopolies. Approved by Decree of RK Government No. 376 of 21 April 2003;

The current legislative system provides guidelines for public consultation and participation in decision-making, although the scale of such activities is dependent on the type and scale of the proposed project and the degree of public interest.

9.1.4 THE OVOS

Prior to 1991, the laws in Kazakhstan provided for State Ecological Expertise (SEE), based on Soviet era laws. This system was also known as the State Environmental Review (SER), and was in the Soviet era controlled under law by the Law of Environmental Protection. The SER processes are mandatory for all projects and are subject to an OVOS, an Assessment of Environmental Impacts; a procedure which is still regulated by the Soviet era OVOS guidelines of 1990 and amended in 1992.

The OVOS process has evolved continuously within the Kazakh legislative system, although its original structure remains; the precepts of an EIA persist and are comparable to and as comprehensive as that of an EIA carried out in the EU. A project in predevelopment or pilot plant phase will carry out a pred OVOS, pre EIA, a detailed screening assessment, which is common place in the EU Environmental Assessment and Protection system.

9.2 LAND OWNERSHIP

FAR leases from the State the land beneath the existing pilot plant and associated facilities, mine buildings and utilities including the route of the overhead power line, plus a buffer zone of 30 m width surrounding the power line and pylons. The land will be returned at the end of the mine life.

9.3 EIS STUDY

9.3.1 INTRODUCTION

The operation is governed by the Kazakhstan system of environmental compliance and as such a Pred OVOS and full OVOS are completed by registered design institutes and company staff. The OVOS (EIS) is similar to that of an EU based EIS. FAR have completed the full OVOS for a proposed 500,000tpa mining operation and this has been approved in full by the relevant authorities.

The business, when at 1Mtpa, will normally operate at full load on a continuous 24 hour basis, all year, with the exception of the peak winter months of January and February. The business will employ approximately 760 full time staff, though a smaller number, of approximately 100, will be working at any one time as a result of the 15 day on-site rotation.

It is anticipated that the OVOS will be updated to incorporate the requirements for a 1Mtpa mining operation and this will be submitted to the relevant authorities for approval during the project development phase.

9.3.2 USE OF RESOURCES

It is estimated that the site currently uses and the 1Mtpa expansion will use the following resources:

- Electricity – currently 250 kW per annum, rising to 5 MW for a 1Mtpa operation.
- Water – 5 m^3 per day, rising to 18 m^3 per day for a 1Mtpa operation. Water is fully recycled but losses arise mainly from evaporation. Any discharges via a planned and constructed passive treatment system are set out in the OVOS and are approved by the regulator.

9.3.3 PROPOSED DISCHARGE DETERMINATION

The main emissions to air arise as fugitive emissions from diesel powered machinery, the chemical processes within the mineral process plant and dust from open pit mining activities.

Other emissions are noise from the machinery and equipment in use, light used to provide safe road and walk ways around the site and road traffic for the import of materials and export of saleable goods.

The ambient concentrations are not affected due to the small scale of the sources and this is expected to continue with expansion to a 1Mtpa process plant.

9.3.4 EMISSIONS TO WATERS

The basis of the operation is that no waste, solid or liquid, is discharged to water from the project site.

Appropriate containment measures (e.g. bunding etc.) are and will be used to prevent contamination of surface water due to leaks or spillages around the workshop/garage in the main building and from the process plant which is contained within this same building.

Groundwater is sampled from three boreholes located to the south of the current tailings pond facility servicing the pilot plant. Samples are taken bi-annually and analysed internally for potassium, chlorine, hydrocarbons, nitrate, sulphate, ammonia, vanadium, and pH. The results for 2008 showed that the pH of the groundwater was neutral.

9.3.5 EMISSIONS TO GROUND

The main emissions to the ground will the stacking of the open pit waste on ground adjacent to the open pit. The mineral process waste will be minimal and will be stacked in a designed impoundment.

The waste rock has been analysed for carbon and oxides of silica, vanadium, aluminium, iron, potassium, titanium, magnesium and calcium. Silica content is high (>80 % by volume) as expected and is inert and benign.

The mine waste material is, where possible, crushed to a range of gravel products and sold as aggregate and collected by the purchaser or delivered by the mine fleet. This minimises the site waste generation.

Domestic sewage will drain to a septic tank for solids settlement, with the supernatant draining to a sump.

9.3.6 WASTE

Small quantities of hazardous wastes such as waste oil and grease, oil filters, cleaning fluids, engine coolant, lab chemicals and lighting tubes/lamps will be generated. Other wastes consist of septic tank sludge, domestic and packaging waste, scrap metal and spent air and water filters. All waste will be collected and sent off-site for disposal by approved and appropriately licensed/permitted recovery/disposal contractors.

9.3.7 NOISE

The proposed project will be designed with various noise control measures so that the noise from the operations does not exceed 55 dBA by day and 45 dBA by night, during normal operation, at the nearest noise sensitive location (NSL). Since there is an absence of non-industrial locations, this is deemed to be the employee accommodation block.

9.3.8 HABITATS

From information studied there are no designated conservation sites within 5 km of the site.

The proposed development is therefore not expected to have any significant adverse effect on any habitat, nor significantly impinge adversely on the environment.

9.4 FUEL HANDLING AND STORAGE

Fuel is stored in above ground tanks and drums at the site. The tanks appear in reasonable condition and are within a bund, as shown in Figure 9-1. The tanks and drums are situated on a concrete base. The fuel pump for refuelling vehicles is situated a considerable distance from the tanks and it is not known whether any containment measures are located beneath the pump or pipeline.

Figure 9-1: Fuel Drums (Foreground) and Tanks (Background) within Mine Waste Bund

Hazardous chemicals are used at the site. Most of these are stored in the containers that they are supplied in. Sulphuric acid and ammonia water are currently stored in large tanks above the plant and HLP, and gravity fed via suspended pipelines to the plant.

The tanks are situated on a concrete base and are well maintained.

9.5 FIRE SAFETY AND FIRST AID

Fire extinguishers and required equipment along with diagrams for contingency situations are located at various points across the mine and plant site. The ‘Mine Rescue’ (independent contractors) has been engaged for emergency situations and has carried out practice drills.

9.6 MINE CLOSURE AND REHABILITATION

The OVOS requires an assessed mine closure plan. Statutory obligations are in place for accumulation of funds to meet the approved business closure plan. Oversight by the regulator is in place.

9.7 HUMAN RESOURCES

The nearest village to the mine site is Aksumbe, 18 km away. The administrative office of the mine is located at Shieli, approximately 70 km from the mine site, being the closest town to the site and location of the nearest railway station. The mine accommodation block is sized for around 70 persons. This will be upgraded as part of the expansion project.

Unemployment in Shieli region is reported to be around 60 %, with below national average monthly salaries. The workforce for the business will be sourced from Aksumbe, Shieli and surrounding areas. Specialists will be sourced from other parts of Kazakhstan.

9.8 INDUSTRIAL DESIGN IMPLICATION

After initial assessment of the site and the likely project parameters, GBM is considers it unlikely that there will be any significant environmental or social issues nor any significant impacts that cannot be mitigated by normal project design, protection and management measures.

SECTION 10 FINANCIAL ANALYSIS

10.1 BACKGROUND

GBM has audited, with the directors of FAR and senior staff from TFB, a cash flow forecast economic model which separately projects the financial results of (1) the expansion of current processing operations and (2) the combined Phase 1 and Phase 2 developments of the Balasausqandiq deposit and processing plant, to produce a combined cash flow.

The expansion of current processing operations is already underway but for the purposes of the financial model account is taken of expenditure scheduled from November 2018 to the end of 2019. It is assumed that operations continue and production steadily rises over this period, reaching full capacity at the end of the first quarter of 2020 and continuing thereafter for the life of the Balasausqandiq operation. For the financial model of Phases 1 and 2, the cash flow forecast model assumes an initial Phase 1 development of the mine and plant production to 1Mtpa, for which commissioning is scheduled for the last quarter of 2020, followed by an increase to 4Mtpa, starting up in the second half of 2023 and continuing through the economic projection to 2043.

In order to construct the model, calculations were made to determine appropriate capital and operating cost estimates. This information has been audited by GBM and found to be reasonable. The estimated accuracy of the cost estimate is ±15 % for phase 1. Phase 2 costs are estimated by industry standard factoring of the phase 1 costs, therefore there is greater variability in the phase 2 estimated costs.

The cash flow models for Balausa include a post-tax Discounted Cash Flow (DCF) analysis and a resultant Net Present Value (NPV). The business post tax IRR is also derived.

Given that a semi-commercial pilot plant has already been constructed and that mining and treatment operations are already taking place, many of the capital and operating costs have been or are already being incurred and this information has been used in the assessment of costs in the cash flow. The Phase 1 expansion will be little different from a scaled-up version of the pilot plant.

Capital and operating cost estimates were prepared in mixed currencies and reported in United States dollars (USD). The exchange rates used are shown in Table 10-1.

Table 10-1: Currency Exchange Rate

Currency	Code	US$ per unit
Euro	EUR	0.8789
Chinese Yuan Renminbi	CNY	6.96
Russian Rouble	RUB	65.56
Kazakhstan Tenge	KZT	366.8

10.2 CAPITAL COSTS

The capital expenditure required for the development of the business has been derived from experience of building and adapting the pilot plant, quotations from vendors, cross referenced against submissions from capable contractors and constructors. The directors of FAR have substantial experience of mineral asset development in Kazakhstan and the capital costs estimates are considered to be realistic.

Table 10-2 shows the estimated initial capital costs for the expansion of the current processing operations, and table 10-3 shows the capital costs for Phase 1.

Table 10-2: Capital costs for expansion of current processing operations

Cost Centre	TOTAL 000 USD	%
Total Capital		
Railway siding, unloading facilities and storage for raw materials	445	4.3
Connection to HV powerline, transformers and reticulation	2,688	26.2
Worker accommodation and office building	477	4.6
Evaporation ponds	343	3.3
Main process building expansion	1,249	12.2
Expansion of main process plant, leaching circuits etc	2,430	23.8
Ovens and other equipment to convert AMV to vanadium pentoxide	668	6.5
Warehouses, laboratories, security etc	474	4.6
Mobile equipment and vehicles	290	2.8
Allowance for unforeseen items	1,200	11.7
TOTAL	**10,264**	**100.0%**

Note that the above individual items include allowance for contingency, construction, installation and commissioning, whilst allowance for unforeseen items is provided separately. At the time of writing this report, over $550,000 had already been spent towards this capital expenditure using funds generated from operations, leaving net remaining capital expenditure of US$9.7million

Table 10-3: Capital and other initial costs for Phase 1 (1Mtpa) mine and processing

Cost Centre	TOTAL [USD000]	%
Total Capital		
Detailed engineering, other preparatory work and owners' project team	9,728	9.7
Additional exploration	1,494	1.5
Buildings and facilities	6,460	6.5
Mining equipment	7,110	7.1
Additional transport equipment	2,629	2.6
Treatment plant equipment	37,261	37.2
Auxiliary treatment plant equipment	5,884	5.9
Geological exploration equipment	1,024	1.0
Tailings impoundment and mining capital works	2,357	2.4
Other infrastructure	3,948	3.9
Corporate overheads	3,071	3.1
First fill	2,399	2.4
Initial spare parts inventory	1,639	1.6
Commissioning support	1,415	1.4
Contingency and allowance for over-run (18% of pre-contingency capital costs)	13,728	13.7
Total	**100,148**	**100.0**

10.2.1 INDIRECT CAPITAL COSTS - BALASAUSQANDIQ

10.2.1.1 OWNERS COSTS

In addition to the corporate overhead costs of USD 3.1 million during the construction phase shown above, $1.1m has been included for the owners' project management team in "Detailed engineering, other preparatory work".

10.2.1.2 EXPLORATION COSTS

GBM also notes that FAR has allowed USD 1 million for the acquisition of geological drill rig and associated equipment and a further USD 1.5 million in its capital cost estimate for the completion of the upgrade of the resource of OB2, OB3, OB4 and OB5 to JORC 2012 indicated standards. Following the completion of such work, there is likely to be considerable upside potential from the further expansion of operations.

10.2.1.3 FIRST FILL

An allowance based on the cost of reagents for 3 months at 1Mtpa have been used as the basis to estimate first fill costs.

10.2.1.4 CONTINGENCY

Contingency is a cost element of the estimate used to cover the uncertainty and variability associated with unforeseeable elements not defined in the project scope. Based on the level of project definition a contingency of 18 % of the initial capital cost has been included as part of the capital cost estimate.

10.2.2 WORKING CAPITAL

Approximately USD 2.8 million has been allowed for working capital costs in addition to the corporate overheads and owners' team project management costs. Full salaries have been assumed during commissioning and the ramp up of production.

10.2.3 PHASED COSTS DISCUSSION

It has been assumed that the bulk of the detailed engineering and other project preparations is carried out in the first six months and that the main capital expenditure will take place during the following 18 months.

It has been assumed that commissioning will start at the end of the second year and that production that year will be 25% of a full year, 80% the second and 100% for the third year of production onwards.

Sustaining capital costs include annual building maintenance, upgrade to a 4Mtpa plant operation and replacement of mobile equipment based on estimated vehicle service life. Similarly, the service life of the mining fleet has been estimated and sustaining capital allows for purchase of new mobile equipment as required. Overhaul and maintenance costs of the process plant and auxiliary equipment are included within the operating costs rather than as sustaining capital.

10.3 OPERATING COSTS - BALASAUSQANDIQ

The main operating cost components in the Balausa project are:

- Mining costs,
- Processing costs (including reagents and transportation), and
- General and administrative costs.

The operating costs for the business have been built up from actual operating experience onsite, procurement prices, and benchmarked against other similar operations. GBM is satisfied that such costs are a reasonable estimate of likely future costs in current terms. A summary of the operating costs is shown in Table 10-3.

Table 10-3: LOM Operating Costs

Description	Life of Mine (LOM) Cost [USD million]	Percentage of LOM OPEX	LOM average cost per ore [USD/t]
GENERAL & ADMINISTRATION	**105**	**4.98%**	**1.37**
General business operations	82	3.89%	1.07
Local employees training	6	0.28%	0.08
Liquidation fund	6	0.28%	0.08
Insurances	1	0.05%	0.01
Development and maintenance of social sphere	10	0.47%	0.13
MINING	**649**	**30.77%**	**8.48**
PROCESS	**1,355**	**64.25%**	**17.70**
Material	38	1.8%	0.50
Reagents	796	37.74%	10.40
Electroenergy	74	3.51%	0.97
Heating	159	7.54%	2.08
Water Supply	1	0.05%	0.01
Repairing of Equipment	151	7.16%	1.97
Lab Chemicals	4	0.19%	0.05
Labour	120	5.69%	1.57
Mobile equipment operation (including fuel, lubricants, labour and maintenance)	12	0.57%	0.16
TOTAL	**2,109**	**100.00%**	**27.55**

The predicted level of costs will make FAR a very low-cost producer. There are several ways of measuring costs per unit of production in a multi-product operation. Offsetting by-product revenues from overall costs would lead to negative costs of vanadium production. Perhaps more appropriately, costs can be arrived at by apportionment of costs pro rata to revenue. The unit costs of production and operating margins are both favourable as detailed in Table 10-4.

Table 10-4: Operating Margins

Description	Processing expansion	1Mtpa ore	4Mtp ore
Operating cost/lb V_2O_5 (where costs are attributed pro-rata to products)	2.76	USD 1.82	USD 1.54
Operating cost/lb V_2O_5 (where by-product revenues are deducted from costs)	2.76	USD (0.80)	USD (1.20)
Operating cost as % of revenue	39.3%	24.2%	20.6 %

Note that operating costs include royalties in the above figures

10.4 FINANCIAL ANALYSIS CRITERIA

A preliminary financial analysis of the Balausa projects has been carried out to estimate the potential economic outcome and its robustness, considering all the possible expenditure which may have an impact on operating and capital costs.

The technical practicality of the project has been investigated and the economic sensitivities tested by the development of a cash flow model with conservative assumptions. The post-tax cash flow forecast model has been prepared based on a mine schedule that is considered attainable.

The main assumptions underlying the base line cash flow model are detailed in Table 10-5.

Table 10-5: Cash Flow Assumptions (current processing expansion)

Assumption	Detail
Currency and inflation	Real terms in 2018 US dollars
Vanadium pentoxide price (Europe)	2018: USD18/lb 2019: USD 13/lb 2020 onwards: $7.50/lb
Metallurgical recovery of V_2O_5	Low grade concentrate 40%; High grade secondary materials 85%
Annual production (V_2O_5)	1,500 tonnes
Kazakhstan corporate tax rate	20 %
Discount rate	10 %

Table 10-6: Cash Flow Assumptions (1 & 4Mtpa project)

Assumption	Detail
Currency and inflation	Real terms in 2018 US dollars
Vanadium pentoxide price (Europe)	7.50 USD/lb of V_2O_5
Carbon-silica flux price	35 USD/t

Assumption	Detail
Annual treatment (tonnes of ore) Year 2 from start of construction (second half) Year 3 Year 4 Year 5 Years 6 – 23	 250,000 800,000 1,000,000 2,500,000 4,000,000
Flux recovery from ore treated	73.8 %
Metallurgical recovery of V_2O_5	91.1 %
Annual production (V_2O_5 only) Year 2 Year 3 Year 4 Year 5 Years 6 – 23	 1,401 tonnes 4,483 5,604 14,009 22,414
Total revenue from each tonne treated	USD 135.25
Kazakhstan corporate tax rate	20 %
Discount rate	10 %

10.5 ASSUMPTIONS AND EXCLUSIONS

All the cost estimates and product sales prices used in the model were taken without inflation and are considered real, not nominal.

For the project to expand the current processing plant, the costs of building the existing operating plant up to its current state have been ignored. It has also been assumed that the bulk of the expansion can be carried out without interrupting operations by building and largely equipping the extension of the factory building alongside the existing, so that only a short shutdown is required to move and install various plant items in the original building.

For the 1Mtpa mine and processing plant, the cost estimate has been prepared beginning from the point of the start of detailed engineering. Therefore, project development costs which have already been incurred are excluded from this cost estimate. Major project development components which have already been incurred and are therefore not included are:

- Pilot Plant capital or operating costs;
- Expansion costs already undertaken
- Metallurgical testwork;
- EIA;
- PFS / BFS engineering and studies;
- Exploration drilling to date;

- Project development social and environmental programs;
- Land acquisition and right of way costs; and
- Permits, licences, bonds or legal and administrative costs associated with government mining and environmental regulations. This includes reporting requirements during operation and related administrative costs.

Additionally, no allowance has been made for:

- Cost escalation;
- Currency fluctuations;
- Currency hedging;

It has been assumed that the mine will produce a total of 76,550 tonnes of ore in the years up to 2043, the currently approved exploitation period. The current JORC (2012) reserves for OB1 amount to some 23m tonnes and an additional 3 million tonnes are expected from the near-surface oxide cap and other inferred resources. The remaining 50.6 million tonnes is a conservative estimate of what might reasonably be expected to be derived from the other four ore-bodies for which the JORC (2012) exploration potential has been assessed as being in the range from 77.3 million tonnes to 103.8 million tonnes. These ore-bodies are outcropping and their known features are sufficient to have a good degree of confidence that after further resource definition, will prove to minable with costs similar to those of OB1. The costs of this further exploration have been allowed for in the financial analysis.

FAR have prepared a feasibility study which includes the additional 50.6 million tonnes which was prepared for locally required approval purposes under the GKZ system of resource estimation. Whilst under the JORC system of resource evaluation these reserves have been classified as Exploration Potential, GBM considers that this feasibility study can be taken as evidence that there are good reasons to consider such Exploration Potential to be Production Target and to be included in the financial evaluation.

GBM has considered whether the additional cash inflows derived from such Production Target should be subject to further discount to reflect the greater uncertainty relating to such material and have concluded that since the total of such Production Target material is some 35% less than the lowest point in the Exploration Potential range, and having regard to the disposition of such material in the geological models examined, the estimates derived are already conservatively estimated and no further discount should be applied.

10.6 FINANCIAL MODEL RESULTS

The main aspects of the cash flow model are presented in Table 10-7.

Table 10-6: Main Aspects of Cash Flow Model (processing expansion)

Item	Value [USD]
Capital costs remaining	9.7 million
Funding requirement net of operating cash inflows	5.0 million
Annual operating costs	11.3 million
Annual revenue	23.6 million
Annual operating cash generation	9.7 million
Annual operating cash generation after tax incentive agreement expires in 2027	7.8 million
Base case post tax NPV (10 %)	**73 million**
Base case post tax asset IRR	**242 %**

Table 10-7: Main Aspects of Cash Flow Model (Phases 1 & 2 only)

Item	Throughput	Value [USD]
Phase 1, 1Mtpa, start-up 2020		
Total funding required	Phase 1	100 million
Funding required net of operating inflows	Phase 1	86 million
Annual operating costs & royalty	1Mtpa	32 million
Annual revenue	1Mtpa	135 million
Annual operating cash generation	1Mtpa	103 million
Phase 2, 4Mtpa, start-up 2023		
Total funding for stage	Phase 2	225 million
Funding required net of operating inflows	Phase 2	nil
Annual op. costs & royalty	4Mtpa	110 million
Annual revenue	4Mtpa	541 million
Annual operating cash generation	4Mtpa	430 million
Combined phases 1 & 2 (but excluding expanded processing)		
Base case post tax NPV (10 %)		**1,978 million**
Base case post tax asset IRR		**89 %**

10.7 SENSITIVITIES AND SCENARIOS

A sensitivity analysis was performed on the base case for key cost variables as discussed in Section 10.7.1.

Additionally, upside potential for the project was examined by considering the benefits of revenue gained from smelting of the autoclave tails to produce ferro-silicon. The results of this scenario are presented in Section 10.7.2.

10.7.1 BASE CASE SENSITIVITY

Sensitivities to changes in product revenues and the main operating cost components are provided in Table 10-8 and Table 10-9.

Table 10-8: Sensitivities (Expansion of current processing)

Sensitivity	Base Case	Change %	Sensitivity Value	NPV USD million
Base Case				73
Vanadium pentoxide price/lb	$24/$13/$10/$7.50	-10 %	$21.6/$11.7/$9/$6.75	61
Vanadium pentoxide price/lb (2020 on)	$10.00/$7.50/lb	+47% (2021 on)	$11.00	132
Reduced raw material feed	30,564 t/a	-10 %	27,508 t/a (2021 on)	70
Concentrate price increase	various	+10%	various	62
Operating price increase (excl. Concentrate)	USD 1.8 million	+10 %	USD 2.0 million	68
Capital cost increase	USD 9.7 million	+10 %	USD 10.7 million	72
Discount Rate	10 %	+20 %	12 %	61

Table 10-9: Sensitivities (Balasausqandiq Phases 1 & 2 only)

Sensitivity	Base Case	Change %	Sensitivity Value	NPV USD million
Base Case				1,962
Vanadium pentoxide price reduction	USD 7.50/lb	-10 %	USD 6.75/lb	1,784
Vanadium pentoxide price	$7.50/lb	+47%	$11.00	2,875
Carbon-silica flux price reduction	USD 35.00/tonne	-10 %	USD 31.50/tonne	1,924
Vanadium pentoxide recovery	91%	-10 %	82%	1,797
Carbon-silica flux recovery	74%	-10 %	66%	1,924
Mining operating cost increase	USD $8.48/t	+10 %	USD 9.33/t	1,959
Treatment operating price increase	USD 16.55/t	+10 %	USD 18.21t	1,943
Capital cost increase (phases 1&2)	USD 326 million	+10 %	USD 359 million	1,951

Sensitivity	Base Case	Change %	Sensitivity Value	NPV USD million
Discount Rate	10 %	+20 %	12 %	1,610

GBM notes the low sensitivity to operating cost increases which are expected as mining and treatment costs amount to only around 8 % and 18 % of revenue respectively. Similarly, the project is relatively insensitive to capital costs which amount to only a small proportion of NPV. Not surprisingly, the biggest sensitivities are to the vanadium pentoxide price and recovery where vanadium accounts for 69 % of revenue.

10.7.2 FERRO-SILICON SCENARIO

FAR has considered many uses for the carbon-silica tailings from the autoclave treatment. The base case financial analysis provides for this material to be briquetted and sold to smelters but FAR has considered a further scenario where ferro-silicon itself is made. The reasons for the attractiveness of this project are:

- Very low cost gas is available locally which can lead to power generation by a company owned gas turbine generator at under USD 0.02 per kWh. Since power is the main determiner of cost, this will give FAR a significant cost advantage.
- The carbon and silica are ideal materials for the feed, providing 100% of the silica and approximately 50% of the carbon requirement. Labour costs are also low in Kazakhstan.
- Since the aluminium has been removed from the tailings by the vanadium treatment process, the resulting ferro-silicon will be low-aluminium. This type of ferro-silicon has a niche market and typically sells for a premium over ordinary ferro-silicon.

These three factors indicate that such a project would be highly attractive and preliminary projections show that a 150,000tpa operation would add approximately USD 0.4 billion to the project NPV over and above the benefits of selling the carbon-silica flux to smelters.

10.8 MARKET BASED EVALUATION

GBM have considered whether there are other listed companies with sufficient similarity to FAR to allow some inference as to the valuation of FAR by comparison. Since there are few pure-play listed vanadium producing or vanadium project companies, and such companies are based on deposits with very different types of ore to that which FAR group is planning to mine, such comparisons are difficult. With few exceptions, the operations or projects held by such listed companies are concerned with the mining and treatment of titano-vanadiferous magnetite (TVM) which requires a much more complicated processing route including pre-concentration and high temperature roasting, neither of which is required to treat ore from Balasausqandiq. This difference gives Balasausqandiq significant capital and operating cost advantages which will become more significant as the vanadium price returns to more usual levels.

The process for treating Balasausqandiq ore produces a range of by-products which add significant value to the project and which have no parallel for TVM deposits where the iron ore by-product has little or no value. Furthermore, the lack of a pre-concentration stage in the processing route gives a much higher metallurgical recovery of over 90% for Balasausqandiq compared with around 75% typical for TVM. When by-product values and the higher recovery are taken into account, Balasausqandiq's recovered value per tonne treated is much higher than would be expected by comparison on the basis of in situ grade. TVM deposits are often compared by reference not only to in situ grade but also to concentrate grades but these are not applicable to Balasausqandiq.

For these reasons, GBM does not believe that a detailed comparison with any existing quoted primary pure-play vanadium producers can have any useful value in determining the valuation of Ferro-Alloy Resources Limited and we recommend that greater reliance should be put on the analysis of future cash flows and the resulting NPV estimation.

10.9 DISCUSSION

The results of the financial analysis show the benefit of the ore at Balausqandiq compared with other magnetite-based projects. The ore is treated in a single step with a hydrometallurgical process which does not require the capital and operating costs of pre-concentration and high temperature roasting as in the standard processing route for magnetite ores.

The financial return potentially benefits from the availability of low cost sulphur for the production of sulphuric acid. The burning of sulphur to generate acid is an accepted method of producing low cost acid (and heat energy). Kazakhstan is a large producer of oil which mostly requires desulphurisation. The resulting sulphur is accumulating as an environmental hazard and is available for a nominal cost. Transport to site will be at low cost by rail, resulting in a cost price of sulphuric acid of under USD 50 per tonne and significant energy can be recovered from the process to produce power or steam. This will substantially reduce the life-of-mine acid costs but owing to the current low costs and easy availability of acid FAR is proposing to defer the building of the acid plant until Phase 2 is commenced or acid prices rise, whichever is the sooner.

The operation of the autoclave indicates that leach recoveries are over 93 % for vanadium. The adsorption of vanadium onto resin and its subsequent recovery has been demonstrated. The base case financial model has used a more conservative estimate of recovery of 91.1 %.

SECTION 11 FAR DEVELOPMENT STRATEGY

Although not yet fully explored it is considered likely that the Balasausqandiq deposit is large enough to support an operation of up to 10Mtpa of ore, producing about 55,000 tonnes of vanadium pentoxide over the life of mine. This would be some 36 % of the world market and create oversupply, as well as producing vast quantities of carbon-silica flux for which it might take time to develop a sufficient market.

If such a plant were built, there would be a danger of a sharp reduction in the market price of vanadium and although FAR would most likely be profitable because of its low production cost, it would be a high-risk strategy with consequences that are difficult to forecast.

There would also be engineering risk in building such a plant and starting to mine at a rate of around 55Mtpa (ore and waste).

FAR has therefore sought to develop more slowly, in steps that are more in tune with the natural expansion of the vanadium market that is expected to grow strongly even in the absence of any significant demand for vanadium for flow batteries. If such demand materialises, then FAR will be able to increase its rate of expansion.

Although the marketing position argues for caution, there are economies of scale that mean that larger plants are more attractive provided the vanadium price is not significantly affected. FAR decided that a two-step solution, starting at 1Mtpa, would capture most of the economies of scale and would then allow a subsequent expansion, currently envisaged to be to 4Mtpa, to be paid for out of earnings and debt. Vanadium production from 1Mtpa will be about 5,600 tonnes per annum of pentoxide, less than half the predicted annual growth of the market, and 4Mtpa would produce an additional 16,800 tonnes – figures which are less than the currently estimated deficit and expected growth.

The 1Mtpa expansion followed by a 4Mtpa expansion is an integrated plan aimed at limiting engineering and market risk, whilst still benefiting from the economies of scale of the large deposit size. GBM considers this two-step development to be soundly based and a sensible, low risk development plan.

Either Phase 1 or Phase 2 could be scaled up if the vanadium market warrants it and capital is available. For example, if vanadium redox battery technology takes off as some forecasters believe it will, the effect on the size of the market could be dramatic and could justify a larger expansion at this stage of the project.

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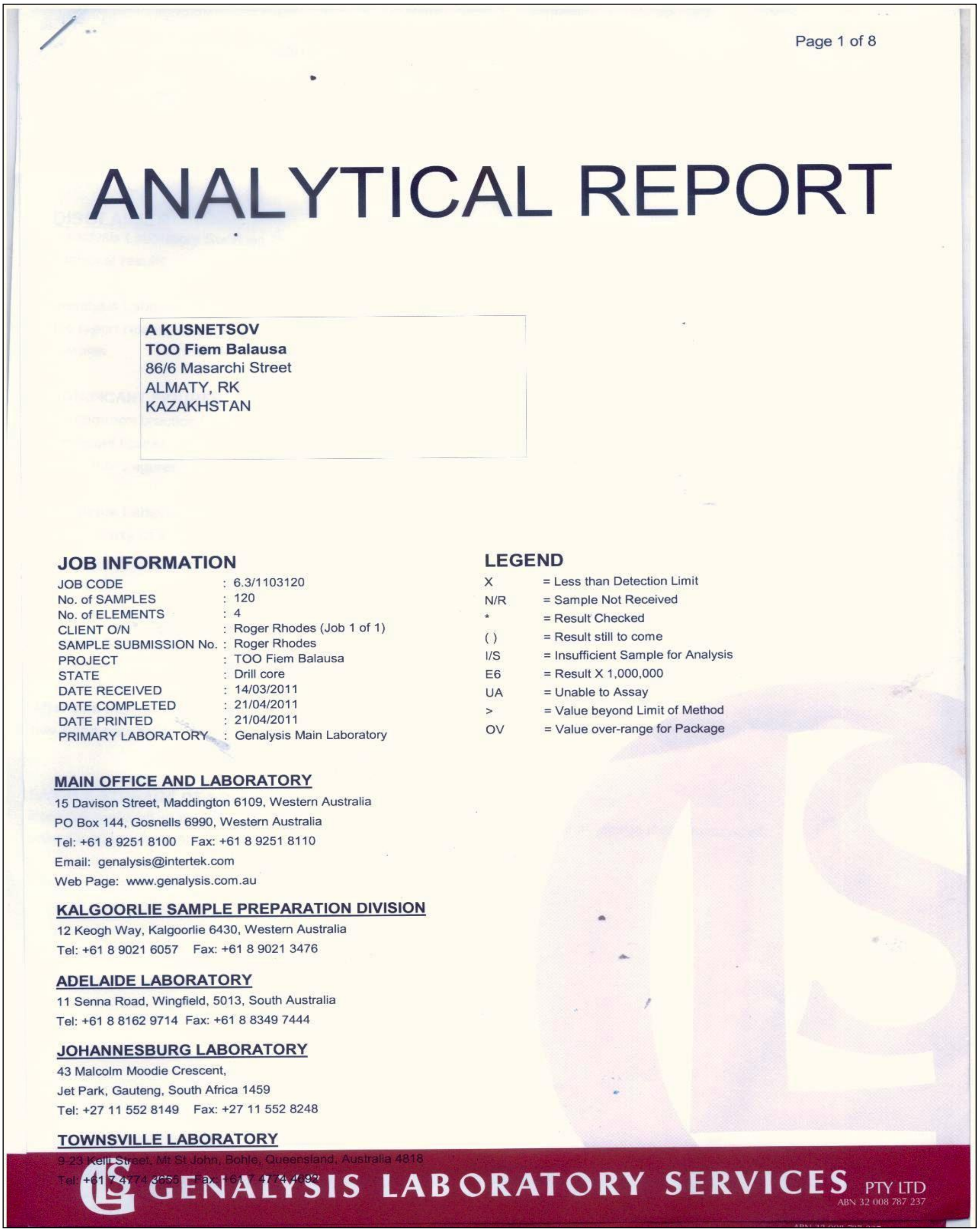
APPENDIX A. ASSAY CERTIFICATES

In this first batch of results from Intertek it was found that the multi-acid digestion approach was similar to the sodium peroxide fusion method for vanadium, molybdenum and uranium, see first page of analytical report “Assay certificates – 1st batch from Intertek”

A.1 ASSAY CERTIFICATES – 1ST BATCH FROM INTERTEK



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ANALYTICAL REPORT

A KUSNETSOV
TOO Fiem Balausa
86/6 Masarchi Street
ALMATY, RK
KAZAKHSTAN

JOB INFORMATION

JOB CODE	: 6.3/1103120
No. of SAMPLES	: 120
No. of ELEMENTS	: 4
CLIENT O/N	: Roger Rhodes (Job 1 of 1)
SAMPLE SUBMISSION No.	: Roger Rhodes
PROJECT	: TOO Fiem Balausa
STATE	: Drill core
DATE RECEIVED	: 14/03/2011
DATE COMPLETED	: 21/04/2011
DATE PRINTED	: 21/04/2011
PRIMARY LABORATORY	: Genalysis Main Laboratory

LEGEND

X	= Less than Detection Limit
N/R	= Sample Not Received
*	= Result Checked
( )	= Result still to come
I/S	= Insufficient Sample for Analysis
E6	= Result X 1,000,000
UA	= Unable to Assay
>	= Value beyond Limit of Method
OV	= Value over-range for Package

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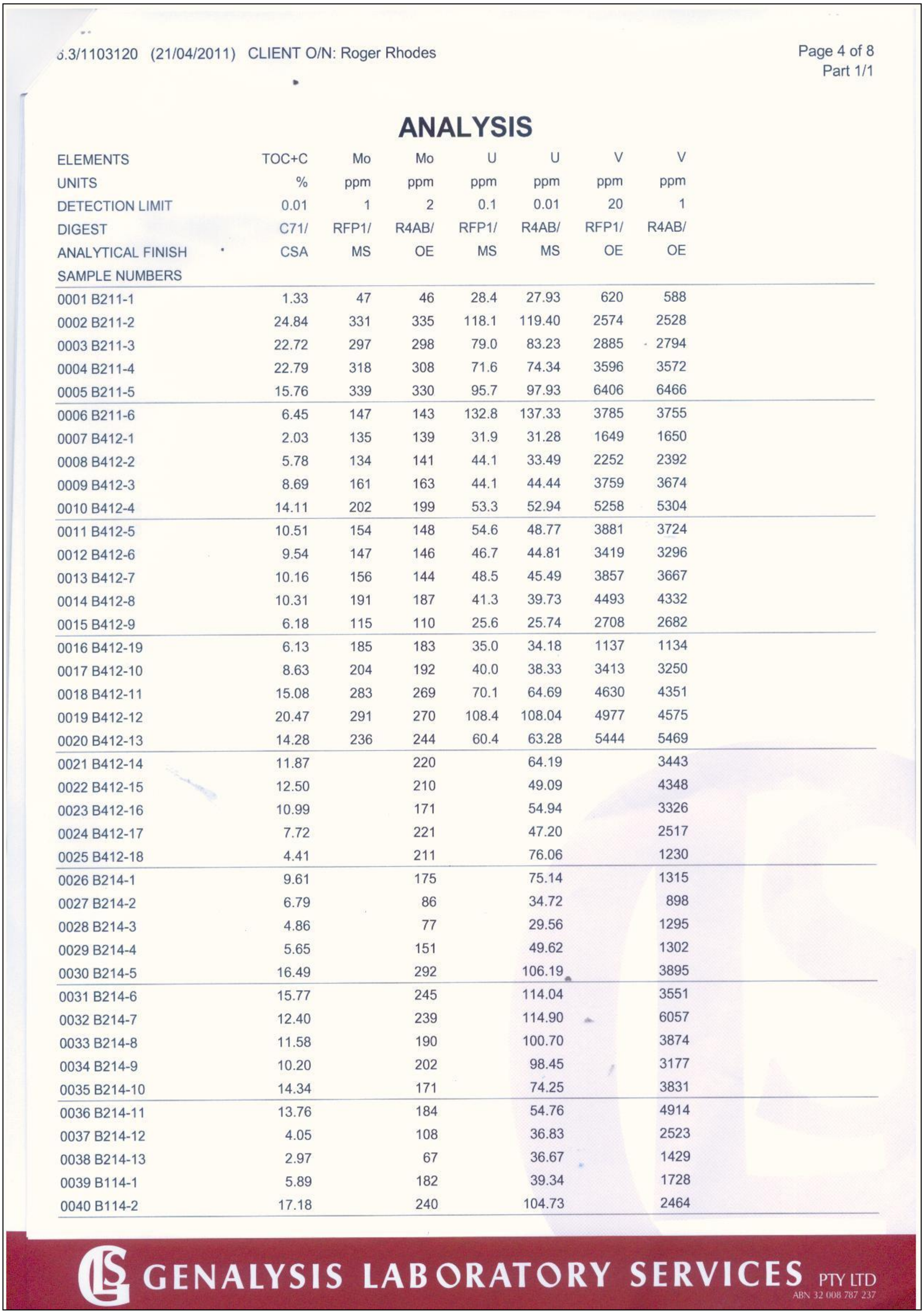

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ANALYSIS

ELEMENTS	TOC+C	Mo	Mo	U	U	V	V
UNITS	%	ppm	ppm	ppm	ppm	ppm	ppm
DETECTION LIMIT	0.01	1	2	0.1	0.01	20	1
DIGEST	C71/	RFP1/	R4AB/	RFP1/	R4AB/	RFP1/	R4AB/
ANALYTICAL FINISH	CSA	MS	OE	MS	MS	OE	OE
SAMPLE NUMBERS							
0001 B211-1	1.33	47	46	28.4	27.93	620	588
0002 B211-2	24.84	331	335	118.1	119.40	2574	2528
0003 B211-3	22.72	297	298	79.0	83.23	2885	2794
0004 B211-4	22.79	318	308	71.6	74.34	3596	3572
0005 B211-5	15.76	339	330	95.7	97.93	6406	6466
0006 B211-6	6.45	147	143	132.8	137.33	3785	3755
0007 B412-1	2.03	135	139	31.9	31.28	1649	1650
0008 B412-2	5.78	134	141	44.1	33.49	2252	2392
0009 B412-3	8.69	161	163	44.1	44.44	3759	3674
0010 B412-4	14.11	202	199	53.3	52.94	5258	5304
0011 B412-5	10.51	154	148	54.6	48.77	3881	3724
0012 B412-6	9.54	147	146	46.7	44.81	3419	3296
0013 B412-7	10.16	156	144	48.5	45.49	3857	3667
0014 B412-8	10.31	191	187	41.3	39.73	4493	4332
0015 B412-9	6.18	115	110	25.6	25.74	2708	2682
0016 B412-19	6.13	185	183	35.0	34.18	1137	1134
0017 B412-10	8.63	204	192	40.0	38.33	3413	3250
0018 B412-11	15.08	283	269	70.1	64.69	4630	4351
0019 B412-12	20.47	291	270	108.4	108.04	4977	4575
0020 B412-13	14.28	236	244	60.4	63.28	5444	5469
0021 B412-14	11.87		220		64.19		3443
0022 B412-15	12.50		210		49.09		4348
0023 B412-16	10.99		171		54.94		3326
0024 B412-17	7.72		221		47.20		2517
0025 B412-18	4.41		211		76.06		1230
0026 B214-1	9.61		175		75.14		1315
0027 B214-2	6.79		86		34.72		898
0028 B214-3	4.86		77		29.56		1295
0029 B214-4	5.65		151		49.62		1302
0030 B214-5	16.49		292		106.19		3895
0031 B214-6	15.77		245		114.04		3551
0032 B214-7	12.40		239		114.90		6057
0033 B214-8	11.58		190		100.70		3874
0034 B214-9	10.20		202		98.45		3177
0035 B214-10	14.34		171		74.25		3831
0036 B214-11	13.76		184		54.76		4914
0037 B214-12	4.05		108		36.83		2523
0038 B214-13	2.97		67		36.67		1429
0039 B114-1	5.89		182		39.34		1728
0040 B114-2	17.18		240		104.73		2464

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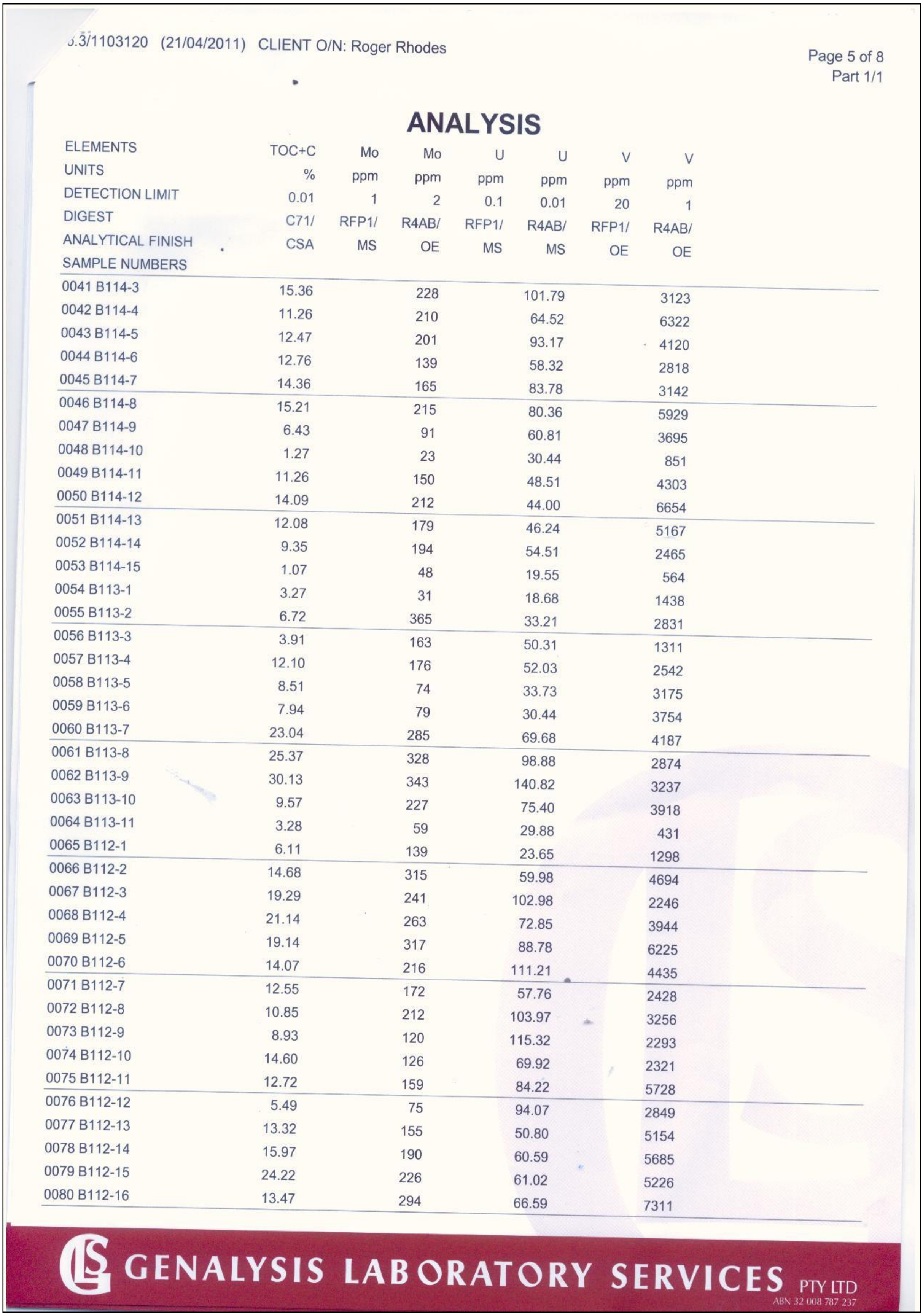

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ANALYSIS

ELEMENTS	TOC+C	Mo	Mo	U	U	V	V
UNITS	%	ppm	ppm	ppm	ppm	ppm	ppm
DETECTION LIMIT	0.01	1	2	0.1	0.01	20	1
DIGEST	C71/	RFP1/	R4AB/	RFP1/	R4AB/	RFP1/	R4AB/
ANALYTICAL FINISH	CSA	MS	OE	MS	MS	OE	OE
SAMPLE NUMBERS							
0041 B114-3	15.36		228		101.79		3123
0042 B114-4	11.26		210		64.52		6322
0043 B114-5	12.47		201		93.17		4120
0044 B114-6	12.76		139		58.32		2818
0045 B114-7	14.36		165		83.78		3142
0046 B114-8	15.21		215		80.36		5929
0047 B114-9	6.43		91		60.81		3695
0048 B114-10	1.27		23		30.44		851
0049 B114-11	11.26		150		48.51		4303
0050 B114-12	14.09		212		44.00		6654
0051 B114-13	12.08		179		46.24		5167
0052 B114-14	9.35		194		54.51		2465
0053 B114-15	1.07		48		19.55		564
0054 B113-1	3.27		31		18.68		1438
0055 B113-2	6.72		365		33.21		2831
0056 B113-3	3.91		163		50.31		1311
0057 B113-4	12.10		176		52.03		2542
0058 B113-5	8.51		74		33.73		3175
0059 B113-6	7.94		79		30.44		3754
0060 B113-7	23.04		285		69.68		4187
0061 B113-8	25.37		328		98.88		2874
0062 B113-9	30.13		343		140.82		3237
0063 B113-10	9.57		227		75.40		3918
0064 B113-11	3.28		59		29.88		431
0065 B112-1	6.11		139		23.65		1298
0066 B112-2	14.68		315		59.98		4694
0067 B112-3	19.29		241		102.98		2246
0068 B112-4	21.14		263		72.85		3944
0069 B112-5	19.14		317		88.78		6225
0070 B112-6	14.07		216		111.21		4435
0071 B112-7	12.55		172		57.76		2428
0072 B112-8	10.85		212		103.97		3256
0073 B112-9	8.93		120		115.32		2293
0074 B112-10	14.60		126		69.92		2321
0075 B112-11	12.72		159		84.22		5728
0076 B112-12	5.49		75		94.07		2849
0077 B112-13	13.32		155		50.80		5154
0078 B112-14	15.97		190		60.59		5685
0079 B112-15	24.22		226		61.02		5226
0080 B112-16	13.47		294		66.59		7311

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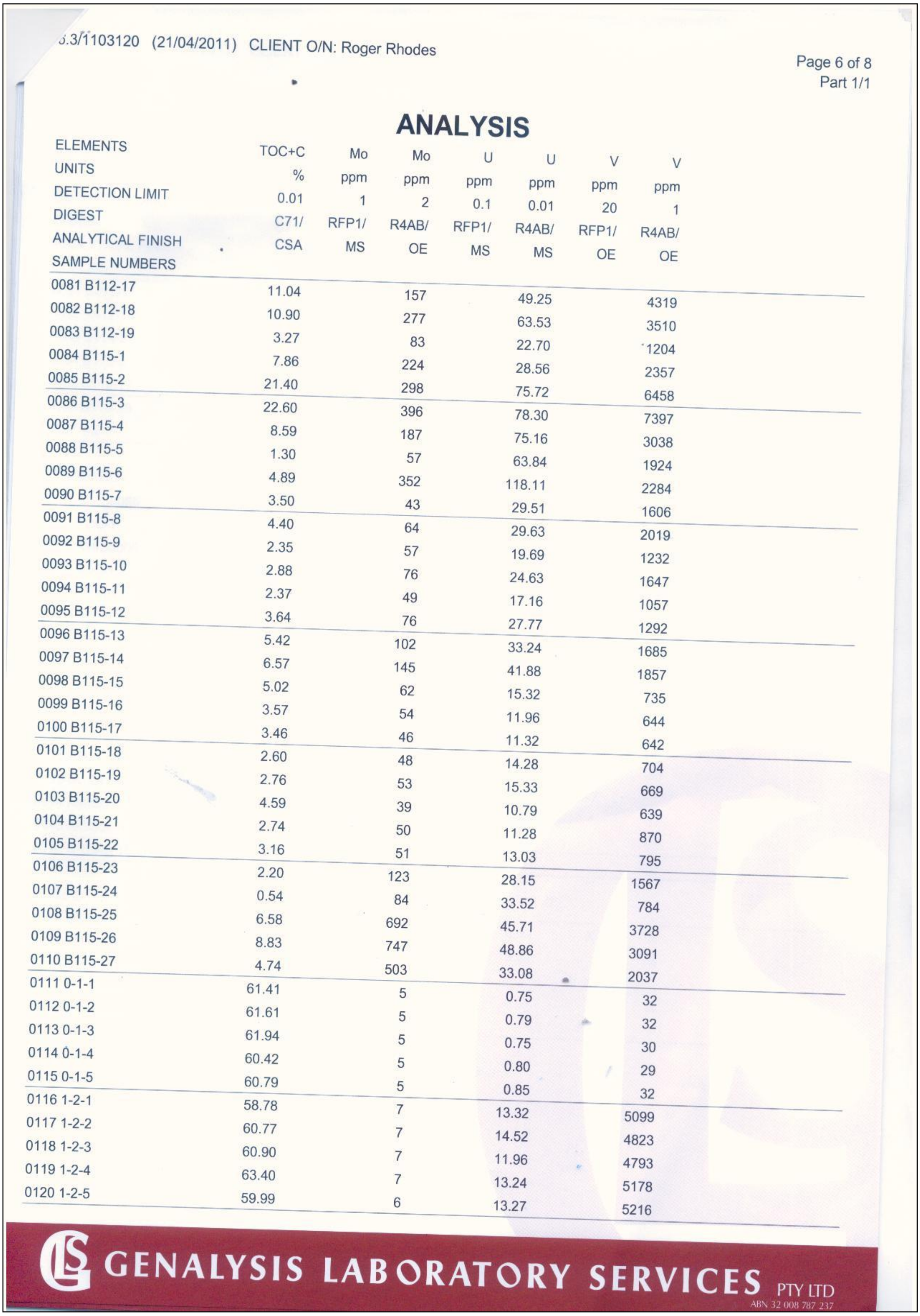

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ANALYSIS

ELEMENTS	TOC+C	Mo	Mo	U	U	V	V
UNITS	%	ppm	ppm	ppm	ppm	ppm	ppm
DETECTION LIMIT	0.01	1	2	0.1	0.01	20	1
DIGEST	C71/	RFP1/	R4AB/	RFP1/	R4AB/	RFP1/	R4AB/
ANALYTICAL FINISH	CSA	MS	OE	MS	MS	OE	OE
SAMPLE NUMBERS							
0081 B112-17	11.04		157		49.25		4319
0082 B112-18	10.90		277		63.53		3510
0083 B112-19	3.27		83		22.70		1204
0084 B115-1	7.86		224		28.56		2357
0085 B115-2	21.40		298		75.72		6458
0086 B115-3	22.60		396		78.30		7397
0087 B115-4	8.59		187		75.16		3038
0088 B115-5	1.30		57		63.84		1924
0089 B115-6	4.89		352		118.11		2284
0090 B115-7	3.50		43		29.51		1606
0091 B115-8	4.40		64		29.63		2019
0092 B115-9	2.35		57		19.69		1232
0093 B115-10	2.88		76		24.63		1647
0094 B115-11	2.37		49		17.16		1057
0095 B115-12	3.64		76		27.77		1292
0096 B115-13	5.42		102		33.24		1685
0097 B115-14	6.57		145		41.88		1857
0098 B115-15	5.02		62		15.32		735
0099 B115-16	3.57		54		11.96		644
0100 B115-17	3.46		46		11.32		642
0101 B115-18	2.60		48		14.28		704
0102 B115-19	2.76		53		15.33		669
0103 B115-20	4.59		39		10.79		639
0104 B115-21	2.74		50		11.28		870
0105 B115-22	3.16		51		13.03		795
0106 B115-23	2.20		123		28.15		1567
0107 B115-24	0.54		84		33.52		784
0108 B115-25	6.58		692		45.71		3728
0109 B115-26	8.83		747		48.86		3091
0110 B115-27	4.74		503		33.08		2037
0111 0-1-1	61.41		5		0.75		32
0112 0-1-2	61.61		5		0.79		32
0113 0-1-3	61.94		5		0.75		30
0114 0-1-4	60.42		5		0.80		29
0115 0-1-5	60.79		5		0.85		32
0116 1-2-1	58.78		7		13.32		5099
0117 1-2-2	60.77		7		14.52		4823
0118 1-2-3	60.90		7		11.96		4793
0119 1-2-4	63.40		7		13.24		5178
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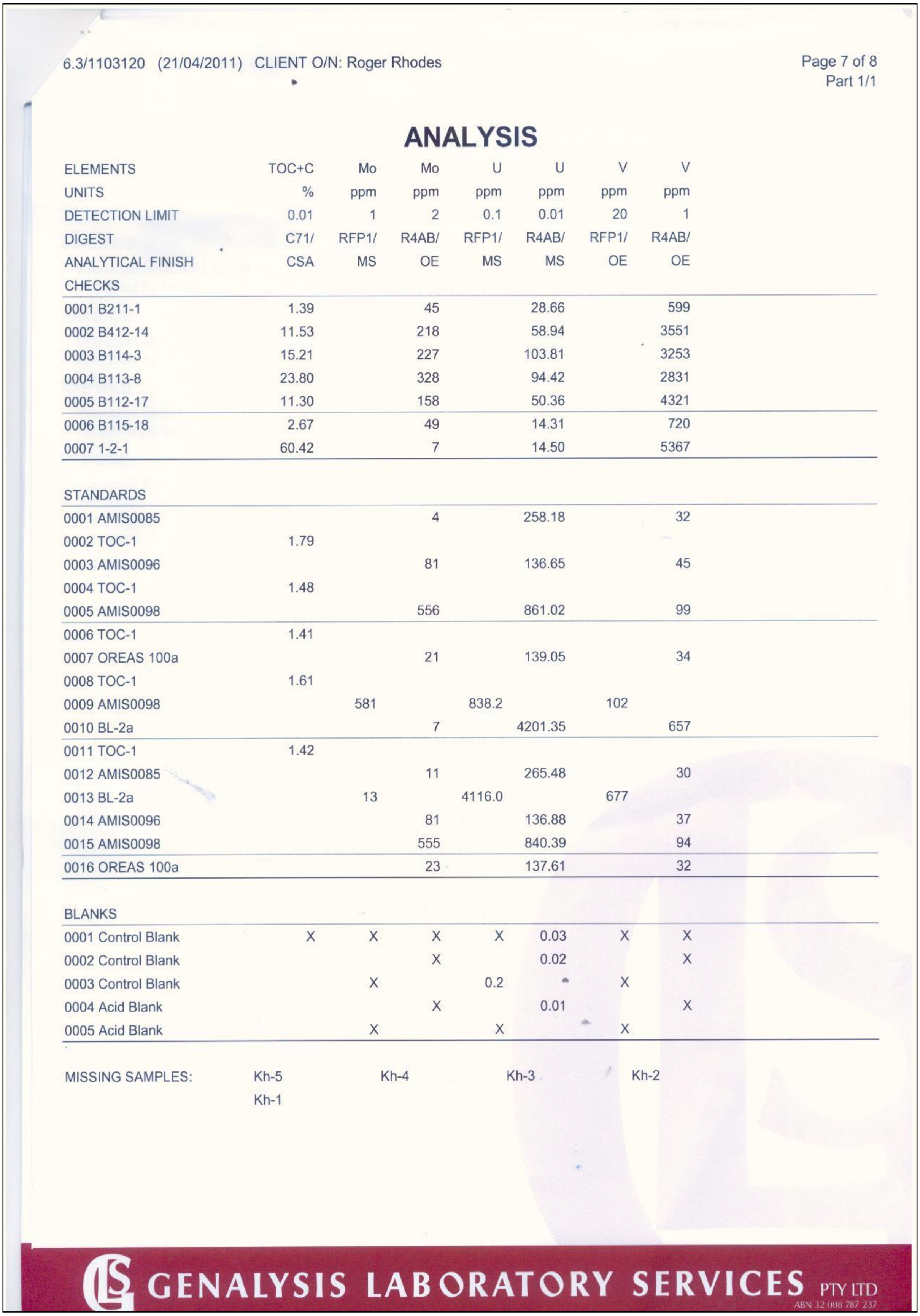

6.3/1103120 (21/04/2011) CLIENT O/N: Roger Rhodes

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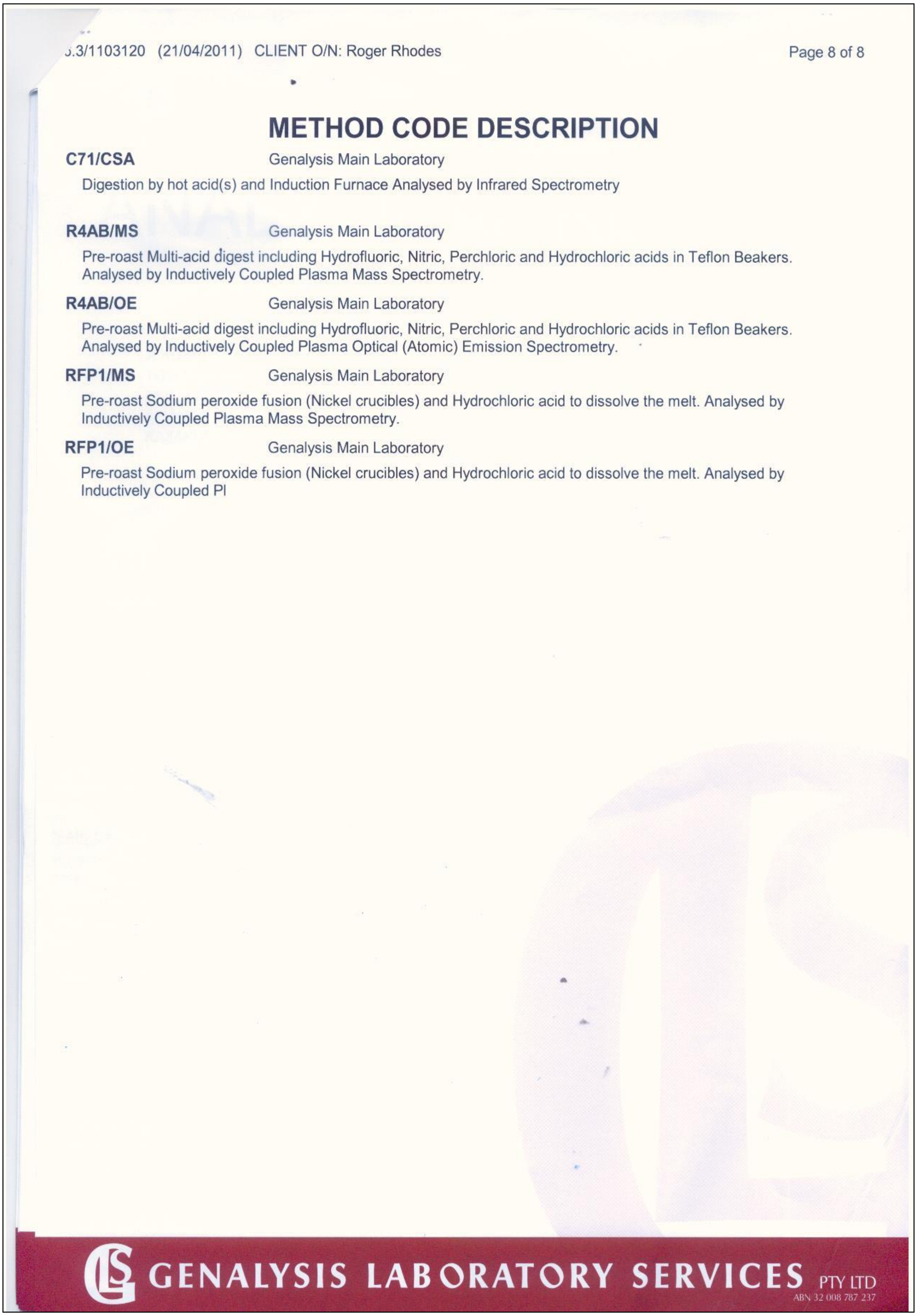
Part 1/1

ANALYSIS

ELEMENTS	TOC+C	Mo	Mo	U	U	V	V
UNITS	%	ppm	ppm	ppm	ppm	ppm	ppm
DETECTION LIMIT	0.01	1	2	0.1	0.01	20	1
DIGEST	C71/	RFP1/	R4AB/	RFP1/	R4AB/	RFP1/	R4AB/
ANALYTICAL FINISH	CSA	MS	OE	MS	MS	OE	OE
CHECKS							
0001 B211-1	1.39		45		28.66		599
0002 B412-14	11.53		218		58.94		3551
0003 B114-3	15.21		227		103.81		3253
0004 B113-8	23.80		328		94.42		2831
0005 B112-17	11.30		158		50.36		4321
0006 B115-18	2.67		49		14.31		720
0007 1-2-1	60.42		7		14.50		5367
STANDARDS							
0001 AMIS0085			4		258.18		32
0002 TOC-1	1.79						
0003 AMIS0096			81		136.65		45
0004 TOC-1	1.48						
0005 AMIS0098			556		861.02		99
0006 TOC-1	1.41						
0007 OREAS 100a			21		139.05		34
0008 TOC-1	1.61						
0009 AMIS0098		581		838.2		102	
0010 BL-2a			7		4201.35		657
0011 TOC-1	1.42						
0012 AMIS0085			11		265.48		30
0013 BL-2a		13		4116.0		677	
0014 AMIS0096			81		136.88		37
0015 AMIS0098			555		840.39		94
0016 OREAS 100a			23		137.61		32
BLANKS							
0001 Control Blank	X	X	X	X	0.03	X	X
0002 Control Blank			X		0.02		X
0003 Control Blank		X		0.2		X	
0004 Acid Blank			X		0.01		X
0005 Acid Blank		X		X		X	

MISSING SAMPLES: Kh-5 Kh-4 Kh-3 Kh-2
Kh-1

GENALYSIS LABORATORY SERVICES PTY LTD
ABN 32 008 787 237



.3/1103120 (21/04/2011) CLIENT O/N: Roger Rhodes

Page 8 of 8

METHOD CODE DESCRIPTION

C71/CSA Genalysis Main Laboratory

Digestion by hot acid(s) and Induction Furnace Analysed by Infrared Spectrometry

R4AB/MS Genalysis Main Laboratory

Pre-roast Multi-acid digest including Hydrofluoric, Nitric, Perchloric and Hydrochloric acids in Teflon Beakers. Analysed by Inductively Coupled Plasma Mass Spectrometry.

R4AB/OE Genalysis Main Laboratory

Pre-roast Multi-acid digest including Hydrofluoric, Nitric, Perchloric and Hydrochloric acids in Teflon Beakers. Analysed by Inductively Coupled Plasma Optical (Atomic) Emission Spectrometry.

RFP1/MS Genalysis Main Laboratory

Pre-roast Sodium peroxide fusion (Nickel crucibles) and Hydrochloric acid to dissolve the melt. Analysed by Inductively Coupled Plasma Mass Spectrometry.

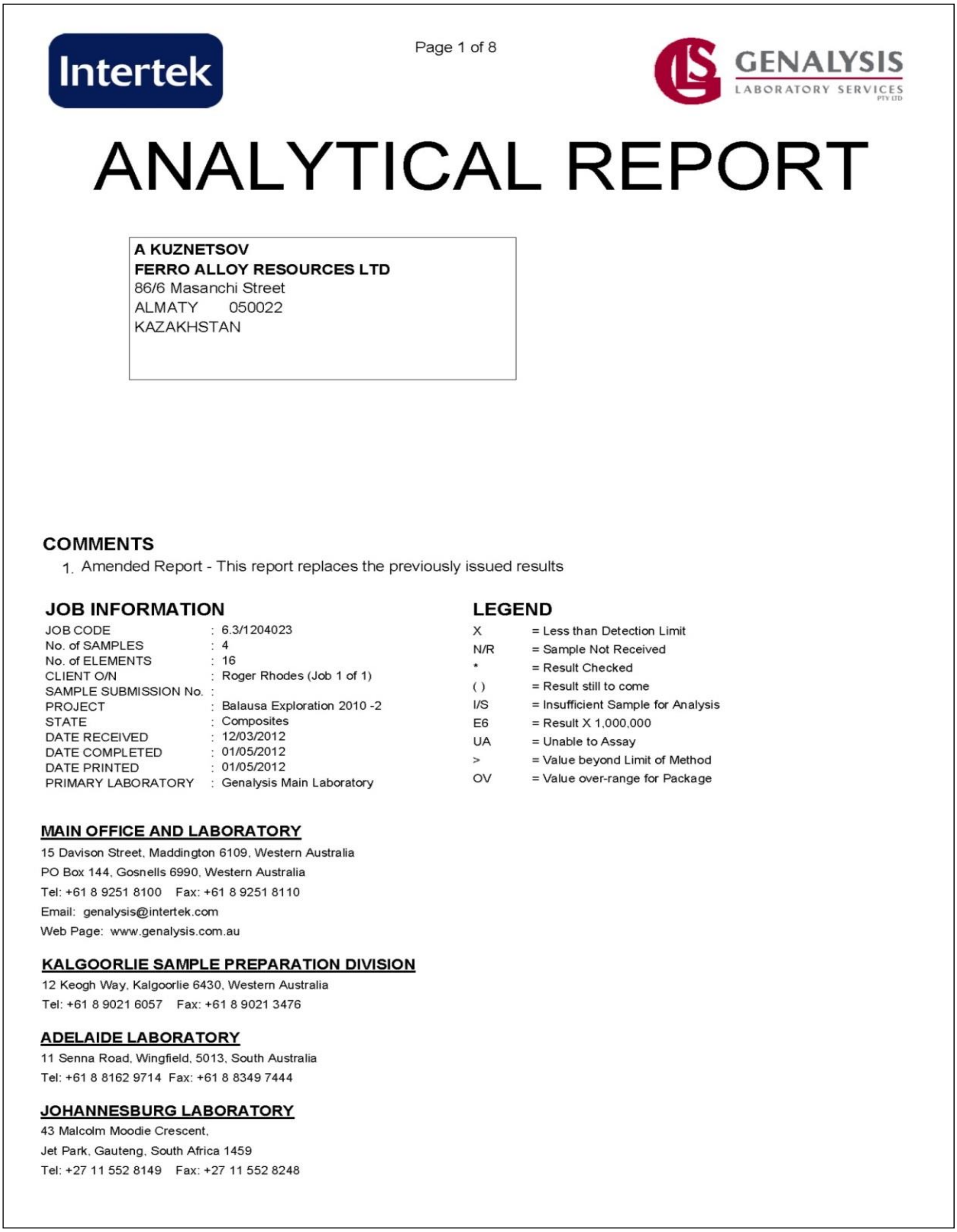
RFP1/OE Genalysis Main Laboratory

Pre-roast Sodium peroxide fusion (Nickel crucibles) and Hydrochloric acid to dissolve the melt. Analysed by Inductively Coupled Pl

GENALYSIS LABORATORY SERVICES PTY LTD
ABN 32 008 787 237

A.2 REE TEST CERTIFICATE RESULTS FROM INTERTEK

Showing 4-acid digestion v peroxide fusion results



Page 1 of 8

GENALYSIS
LABORATORY SERVICES PTY LTD

ANALYTICAL REPORT

A KUZNETSOV
FERRO ALLOY RESOURCES LTD
86/6 Masanchi Street
ALMATY 050022
KAZAKHSTAN

COMMENTS

1. Amended Report - This report replaces the previously issued results

JOB INFORMATION

JOB CODE : 6.3/1204023
No. of SAMPLES : 4
No. of ELEMENTS : 16
CLIENT O/N : Roger Rhodes (Job 1 of 1)
SAMPLE SUBMISSION No. :
PROJECT : Balausa Exploration 2010 -2
STATE : Composites
DATE RECEIVED : 12/03/2012
DATE COMPLETED : 01/05/2012
DATE PRINTED : 01/05/2012
PRIMARY LABORATORY : Genalysis Main Laboratory

LEGEND

X = Less than Detection Limit
N/R = Sample Not Received
* = Result Checked
( ) = Result still to come
I/S = Insufficient Sample for Analysis
E6 = Result X 1,000,000
UA = Unable to Assay
> = Value beyond Limit of Method
OV = Value over-range for Package

MAIN OFFICE AND LABORATORY
15 Davison Street, Maddington 6109, Western Australia
PO Box 144, Gosnells 6990, Western Australia
Tel: +61 8 9251 8100 Fax: +61 8 9251 8110
Email: genalysis@intertek.com
Web Page: www.genalysis.com.au

KALGOORLIE SAMPLE PREPARATION DIVISION
12 Keogh Way, Kalgoorlie 6430, Western Australia
Tel: +61 8 9021 6057 Fax: +61 8 9021 3476

ADELAIDE LABORATORY
11 Senna Road, Wingfield, 5013, South Australia
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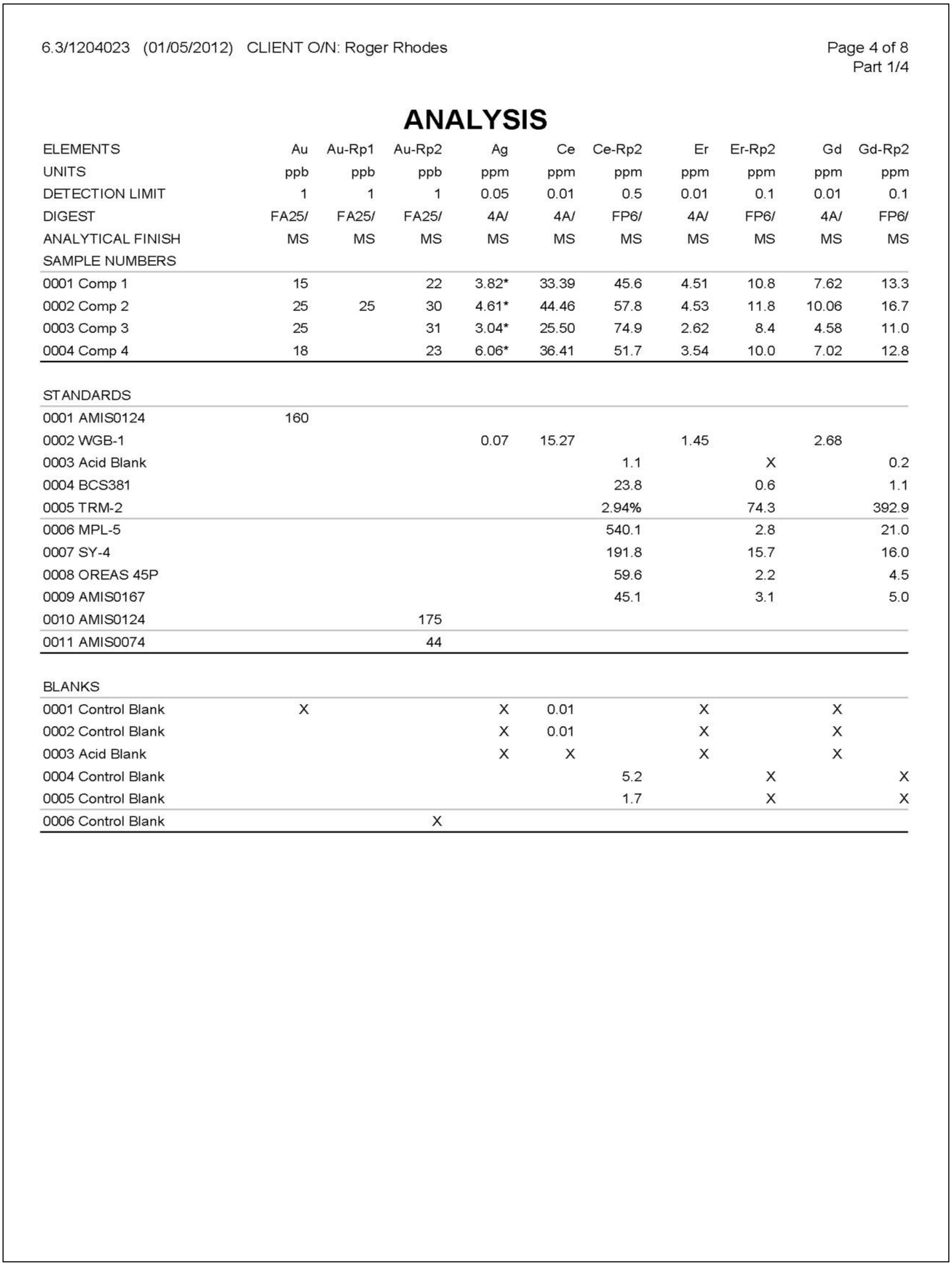
JOHANNESBURG LABORATORY
43 Malcolm Moodie Crescent,
Jet Park, Gauteng, South Africa 1459
Tel: +27 11 552 8149 Fax: +27 11 552 8248



6.3/1204023 (01/05/2012) CLIENT O/N: Roger Rhodes

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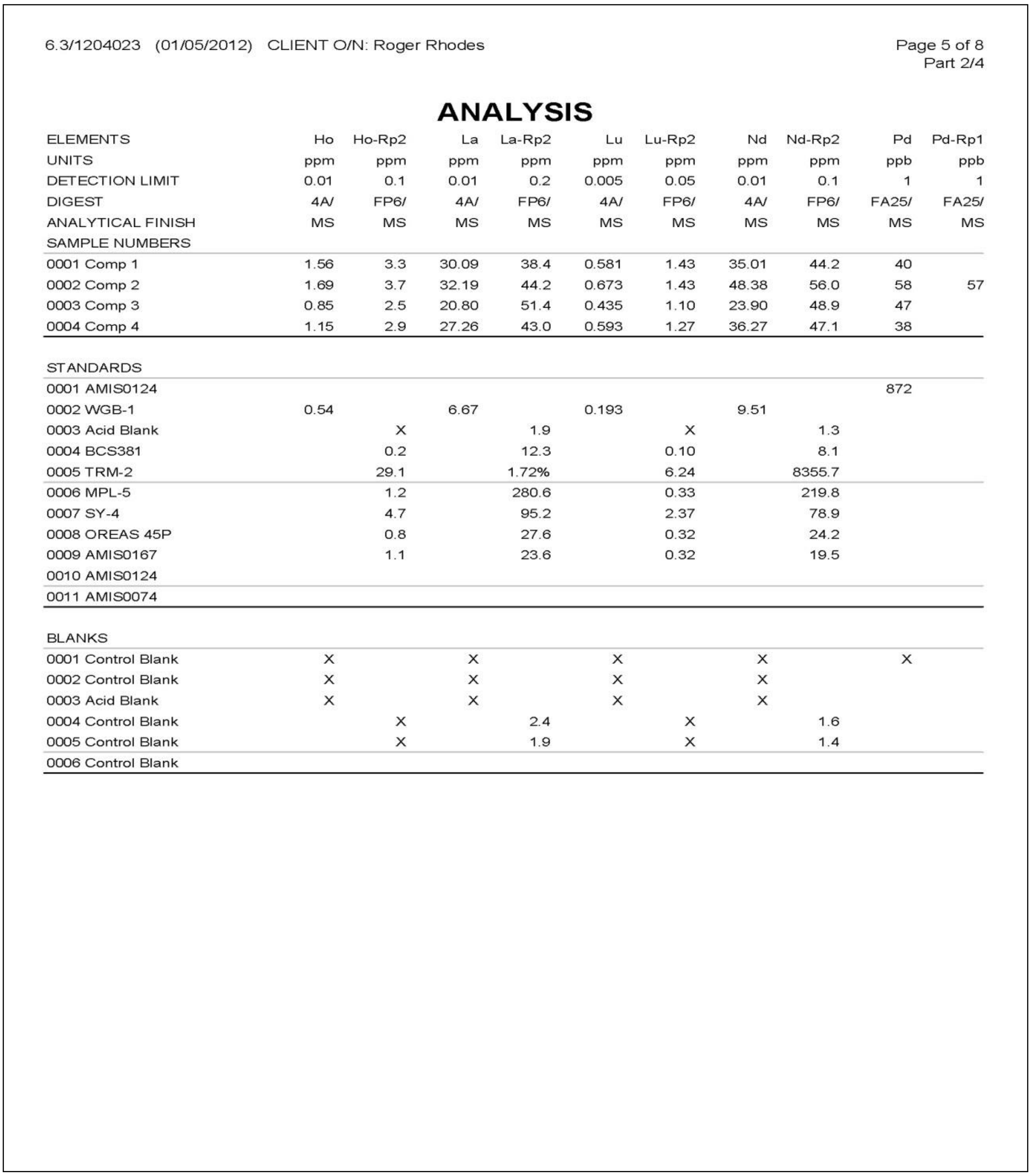
Part 1/4

ANALYSIS

ELEMENTS	Au	Au-Rp1	Au-Rp2	Ag	Ce	Ce-Rp2	Er	Er-Rp2	Gd	Gd-Rp2
UNITS	ppb	ppb	ppb	ppm	ppm	ppm	ppm	ppm	ppm	ppm
DETECTION LIMIT	1	1	1	0.05	0.01	0.5	0.01	0.1	0.01	0.1
DIGEST	FA25/	FA25/	FA25/	4A/	4A/	FP6/	4A/	FP6/	4A/	FP6/
ANALYTICAL FINISH	MS	MS	MS	MS	MS	MS	MS	MS	MS	MS
SAMPLE NUMBERS										
0001 Comp 1	15		22	3.82*	33.39	45.6	4.51	10.8	7.62	13.3
0002 Comp 2	25	25	30	4.61*	44.46	57.8	4.53	11.8	10.06	16.7
0003 Comp 3	25		31	3.04*	25.50	74.9	2.62	8.4	4.58	11.0
0004 Comp 4	18		23	6.06*	36.41	51.7	3.54	10.0	7.02	12.8
STANDARDS										
0001 AMIS0124	160									
0002 WGB-1				0.07	15.27		1.45		2.68	
0003 Acid Blank						1.1		X		0.2
0004 BCS381						23.8		0.6		1.1
0005 TRM-2						2.94%		74.3		392.9
0006 MPL-5						540.1		2.8		21.0
0007 SY-4						191.8		15.7		16.0
0008 OREAS 45P						59.6		2.2		4.5
0009 AMIS0167						45.1		3.1		5.0
0010 AMIS0124			175							
0011 AMIS0074			44							
BLANKS										
0001 Control Blank	X			X	0.01		X		X	
0002 Control Blank				X	0.01		X		X	
0003 Acid Blank				X	X		X		X	
0004 Control Blank						5.2		X		X
0005 Control Blank						1.7		X		X
0006 Control Blank			X							



6.3/1204023 (01/05/2012) CLIENT O/N: Roger Rhodes

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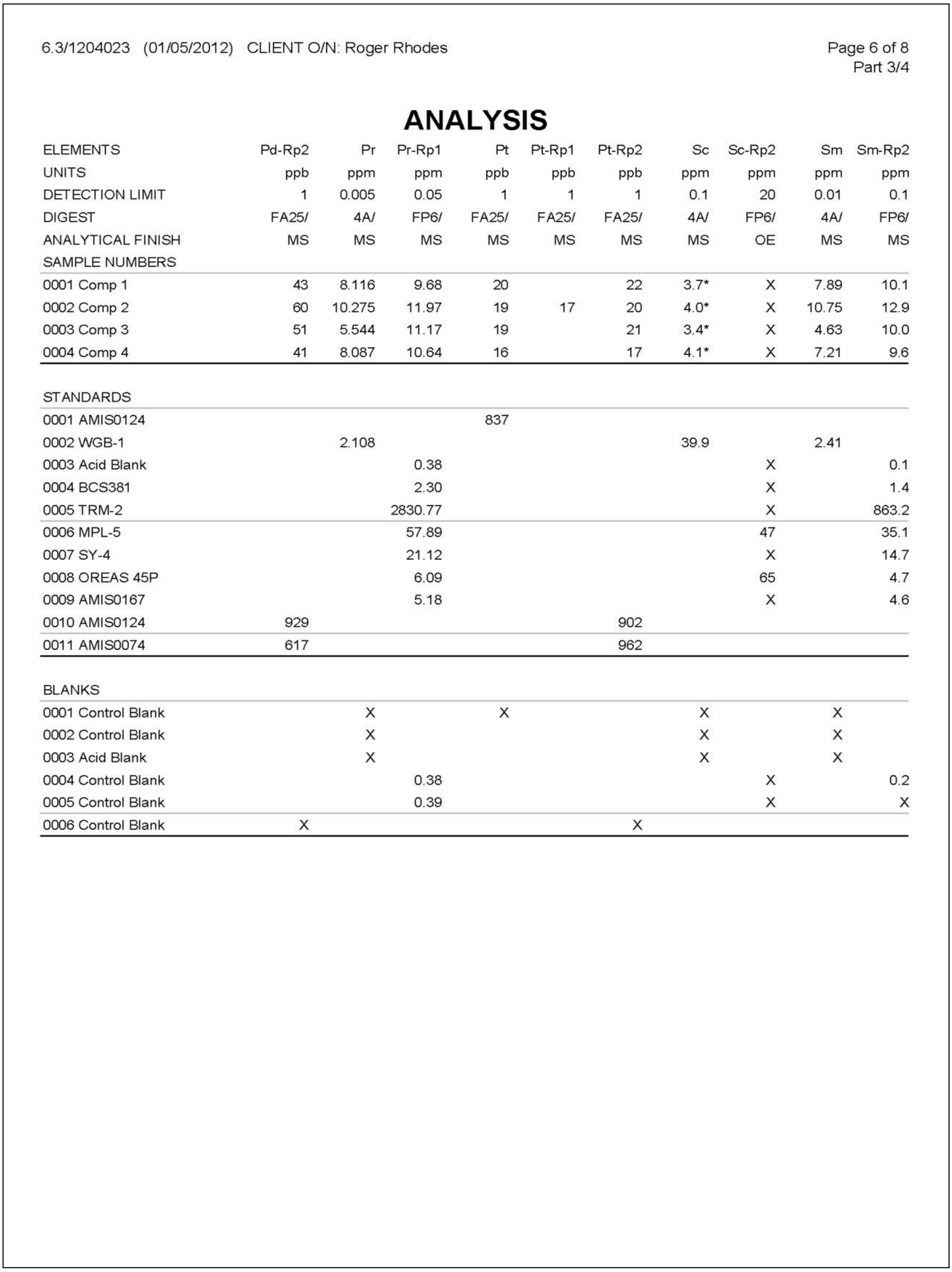
Part 2/4

ANALYSIS

ELEMENTS	Ho	Ho-Rp2	La	La-Rp2	Lu	Lu-Rp2	Nd	Nd-Rp2	Pd	Pd-Rp1
UNITS	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppb	ppb
DETECTION LIMIT	0.01	0.1	0.01	0.2	0.005	0.05	0.01	0.1	1	1
DIGEST	4A/	FP6/	4A/	FP6/	4A/	FP6/	4A/	FP6/	FA25/	FA25/
ANALYTICAL FINISH	MS	MS	MS	MS	MS	MS	MS	MS	MS	MS
SAMPLE NUMBERS										
0001 Comp 1	1.56	3.3	30.09	38.4	0.581	1.43	35.01	44.2	40	
0002 Comp 2	1.69	3.7	32.19	44.2	0.673	1.43	48.38	56.0	58	57
0003 Comp 3	0.85	2.5	20.80	51.4	0.435	1.10	23.90	48.9	47	
0004 Comp 4	1.15	2.9	27.26	43.0	0.593	1.27	36.27	47.1	38	
STANDARDS										
0001 AMIS0124									872	
0002 WGB-1	0.54		6.67		0.193		9.51			
0003 Acid Blank		X		1.9		X		1.3		
0004 BCS381		0.2		12.3		0.10		8.1		
0005 TRM-2		29.1		1.72%		6.24		8355.7		
0006 MPL-5		1.2		280.6		0.33		219.8		
0007 SY-4		4.7		95.2		2.37		78.9		
0008 OREAS 45P		0.8		27.6		0.32		24.2		
0009 AMIS0167		1.1		23.6		0.32		19.5		
0010 AMIS0124										
0011 AMIS0074										
BLANKS										
0001 Control Blank	X		X		X		X		X	
0002 Control Blank	X		X		X		X			
0003 Acid Blank	X		X		X		X			
0004 Control Blank		X		2.4		X		1.6		
0005 Control Blank		X		1.9		X		1.4		
0006 Control Blank										



6.3/1204023 (01/05/2012) CLIENT O/N: Roger Rhodes

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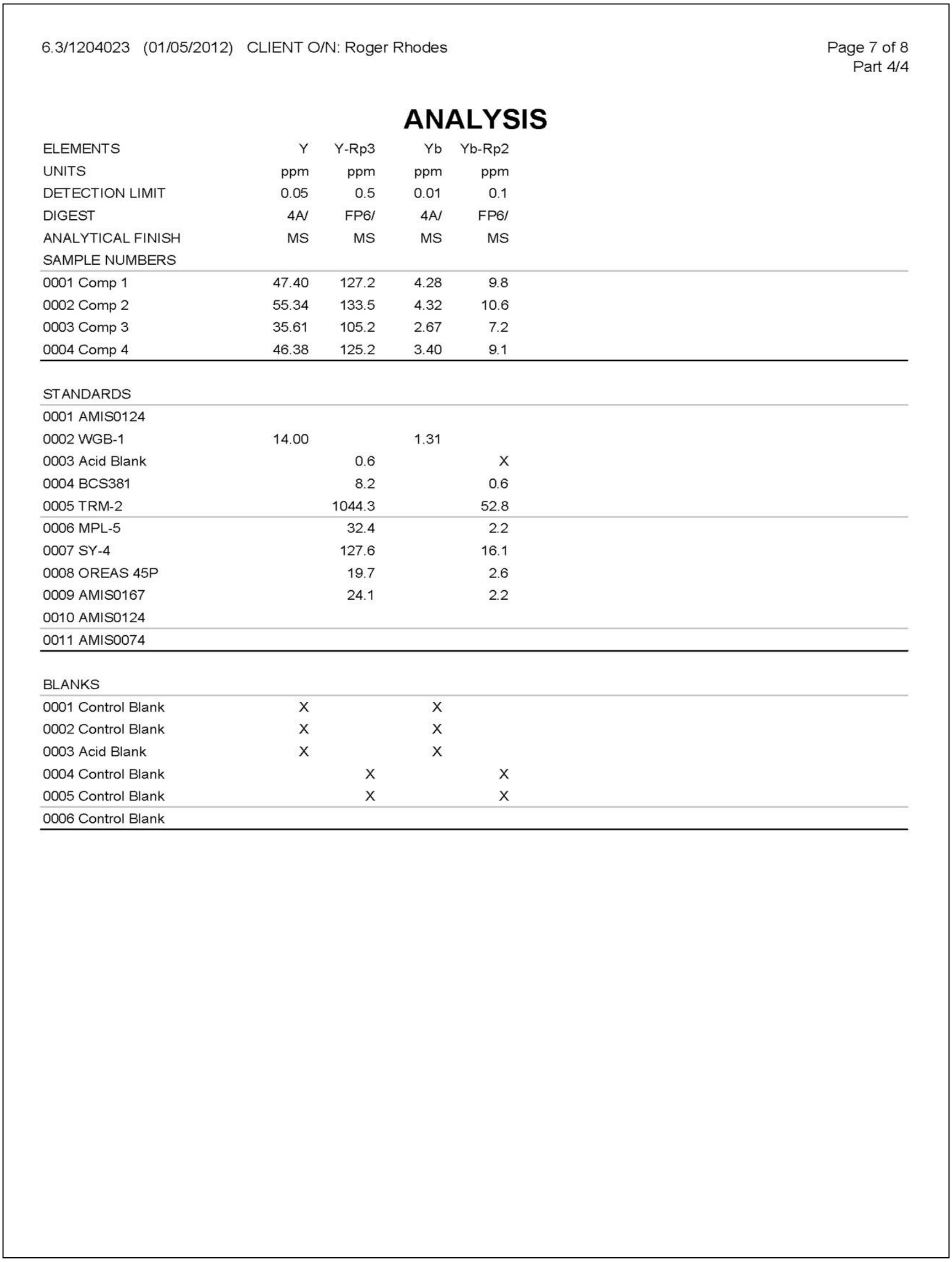
Part 3/4

ANALYSIS

ELEMENTS	Pd-Rp2	Pr	Pr-Rp1	Pt	Pt-Rp1	Pt-Rp2	Sc	Sc-Rp2	Sm	Sm-Rp2
UNITS	ppb	ppm	ppm	ppb	ppb	ppb	ppm	ppm	ppm	ppm
DETECTION LIMIT	1	0.005	0.05	1	1	1	0.1	20	0.01	0.1
DIGEST	FA25/	4A/	FP6/	FA25/	FA25/	FA25/	4A/	FP6/	4A/	FP6/
ANALYTICAL FINISH	MS	MS	MS	MS	MS	MS	MS	OE	MS	MS
SAMPLE NUMBERS										
0001 Comp 1	43	8.116	9.68	20		22	3.7*	X	7.89	10.1
0002 Comp 2	60	10.275	11.97	19	17	20	4.0*	X	10.75	12.9
0003 Comp 3	51	5.544	11.17	19		21	3.4*	X	4.63	10.0
0004 Comp 4	41	8.087	10.64	16		17	4.1*	X	7.21	9.6
STANDARDS										
0001 AMIS0124				837						
0002 WGB-1		2.108					39.9		2.41	
0003 Acid Blank			0.38					X		0.1
0004 BCS381			2.30					X		1.4
0005 TRM-2			2830.77					X		863.2
0006 MPL-5			57.89					47		35.1
0007 SY-4			21.12					X		14.7
0008 OREAS 45P			6.09					65		4.7
0009 AMIS0167			5.18					X		4.6
0010 AMIS0124	929					902				
0011 AMIS0074	617					962				
BLANKS										
0001 Control Blank		X		X			X		X	
0002 Control Blank		X					X		X	
0003 Acid Blank		X					X		X	
0004 Control Blank			0.38					X		0.2
0005 Control Blank			0.39					X		X
0006 Control Blank	X					X				



6.3/1204023 (01/05/2012) CLIENT O/N: Roger Rhodes

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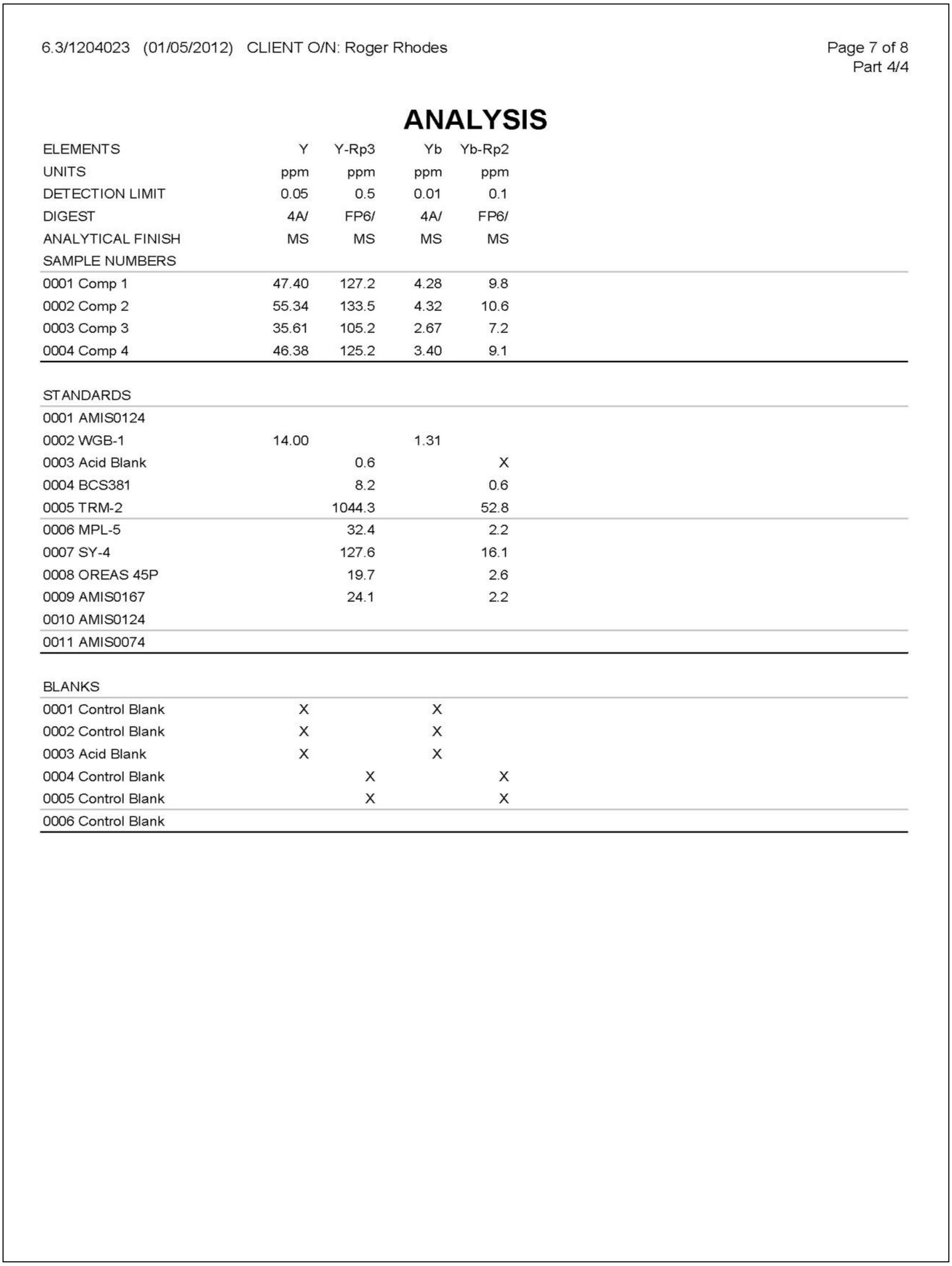
Part 4/4

ANALYSIS

ELEMENTS	Y	Y-Rp3	Yb	Yb-Rp2
UNITS	ppm	ppm	ppm	ppm
DETECTION LIMIT	0.05	0.5	0.01	0.1
DIGEST	4A/	FP6/	4A/	FP6/
ANALYTICAL FINISH	MS	MS	MS	MS
SAMPLE NUMBERS				
0001 Comp 1	47.40	127.2	4.28	9.8
0002 Comp 2	55.34	133.5	4.32	10.6
0003 Comp 3	35.61	105.2	2.67	7.2
0004 Comp 4	46.38	125.2	3.40	9.1
STANDARDS				
0001 AMIS0124				
0002 WGB-1	14.00		1.31	
0003 Acid Blank		0.6		X
0004 BCS381		8.2		0.6
0005 TRM-2		1044.3		52.8
0006 MPL-5		32.4		2.2
0007 SY-4		127.6		16.1
0008 OREAS 45P		19.7		2.6
0009 AMIS0167		24.1		2.2
0010 AMIS0124				
0011 AMIS0074				
BLANKS				
0001 Control Blank	X		X	
0002 Control Blank	X		X	
0003 Acid Blank	X		X	
0004 Control Blank		X		X
0005 Control Blank		X		X
0006 Control Blank				



6.3/1204023 (01/05/2012) CLIENT O/N: Roger Rhodes

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Part 4/4

ANALYSIS

ELEMENTS	Y	Y-Rp3	Yb	Yb-Rp2
UNITS	ppm	ppm	ppm	ppm
DETECTION LIMIT	0.05	0.5	0.01	0.1
DIGEST	4A/	FP6/	4A/	FP6/
ANALYTICAL FINISH	MS	MS	MS	MS
SAMPLE NUMBERS				
0001 Comp 1	47.40	127.2	4.28	9.8
0002 Comp 2	55.34	133.5	4.32	10.6
0003 Comp 3	35.61	105.2	2.67	7.2
0004 Comp 4	46.38	125.2	3.40	9.1
STANDARDS				
0001 AMIS0124				
0002 WGB-1	14.00		1.31	
0003 Acid Blank		0.6		X
0004 BCS381		8.2		0.6
0005 TRM-2		1044.3		52.8
0006 MPL-5		32.4		2.2
0007 SY-4		127.6		16.1
0008 OREAS 45P		19.7		2.6
0009 AMIS0167		24.1		2.2
0010 AMIS0124				
0011 AMIS0074				
BLANKS				
0001 Control Blank	X		X	
0002 Control Blank	X		X	
0003 Acid Blank	X		X	
0004 Control Blank		X		X
0005 Control Blank		X		X
0006 Control Blank				

A.3 KARAGANDA ASSAY CERTIFICATE (TRANSLATION)

KZ.И.10.0109

Testing Centre TOO Tsentrolanaleet

100008, Karaganda, Boulvar Mira, 12; tel/fax: 8(7212) 42-60-39

Laboratory of analytic studies

100008, Karaganda, Boulvar Mira, 12; tel/fax: 8(7212) 42-60-38

Order No 902-129-11

Ordered by TOO Firma Balausa

120700 Kyzylorda obl., Sieli region, Shieli GRP-2

Deposit Balasausqandiq

Method of process: photometric

Date of test: 10.11.2011

Total pages 3

Page 1

TEST REPORT

№ p/p	№ Lab	№ test	V_2O_5, %
1	1	B335-1	0.060
2	2	B335-2	0.105

3	3	B335-3	0.100
4	4	B335-4	0.140
5	5	B335-5	0.210
6	6	B335-6	0.210
7	7	B335-7	0.200
8	8	B335-8	0.106
9	9	B335-9	0.340
10	10	B335-10	0.410
11	11	B335-11	0.540
12	12	B335-12	0.150
13	13	B335-13	0.240
14	14	B335-14	0.210
15	15	B335-15	0.310
16	16	B335-16	0.610
17	17	B335-17	0.590
18	18	B335-18	0.670
19	19	B335-19	0.510
20	20	B335-20	0.400
21	21	B335-21	0.830
22	22	B335-22	0.710
23	23	B335-23	0.560
24	24	B335-24	1.240

25	25	B335-25	1.100
26	26	B335-26	0.770
27	27	B335-27	0.900
28	28	B335-28	0.310
29	29	B335-29	0.620
30	30	B335-30	0.330
31	31	B335-31	1.160
32	32	B335-32	0.490
33	33	B335-33	0.520
34	34	B335-34	0.440
35	35	B335-35	0.180
36	36	B335-36	0.380
37	37	B335-37	0.226
38	38	B335-38	0.380
39	39	B335-39	0.390
40	40	B335-40	0.370
41	41	B335-41	0.330
42	42	B335-42	0.240
43	43	B335-43	0.440
44	44	B335-44	0.190
45	45	B335-45	0.226
46	46	B335-46	0.240

47	47	B335-47	0.180
48	48	B335-48	0.200
49	49	B335-49	0.210
50	50	B335-50	0.290
51	51	B335-51	0.180
52	52	B335-52	0.110
53	53	B335-53	0.180
54	54	B335-54	0.150
55	55	B335-55	0.130
56	56	B335-56	0.140
57	57	B335-57	0.140
58	58	B212,5-1	0.130
59	59	B212,5-2	0.190
60	60	B212,5-3	0.300
61	61	B212,5-4	0.650
62	62	B212,5-5	0.580
63	63	B212,5-6	0.670
64	64	B212,5-7	1.360
65	65	B212,5-8	0.820
66	66	B212,5-9	0.470
67	67	B212,5-10	0.570
68	68	B212,5-11	0.580

69	69	B212,5-12	0.470
70	70	B212,5-13	0.470
71	71	B212,5-14	0.870
72	72	B212,5-15	0.830
73	73	B212,5-16	0.190
74	74	B212,5-17	0.340
75	75	B212,5-18	0.800
76	76	B212,5-19	0.680
77	77	B212,5-20	1.230
78	78	B212,5-21	0.760
79	79	B212,5-22	0.660
80	80	B212,5-23	0.480
81	81	B212,5-24	0.670
82	82	B212,5-25	0.180
83	83	B212,5-26	0.080
84	84	B212,5-27	0.102
85	85	B312,5-1	0.250
86	86	B312,5-2	0.086
87	87	B312,5-3	0.320
88	88	B312,5-4	0.470
89	89	B312,5-5	0.390
90	90	B312,5-6	0.360

91	91	B312,5-7	0.370
92	92	B312,5-8	1.022
93	93	B312,5-9	0.530
94	94	B312,5-10	0.600
95	95	B312,5-11	0.880
96	96	B312,5-12	0.630
97	97	B312,5-13	0.980
98	98	B312,5-14	0.710
99	99	B312,5-15	0.650
100	100	B312,5-16	0.190
101	101	B312,5-17	0.220
102	102	B312,5-18	0.170
103	103	B312,5-19	0.410
104	104	B412,5-1	0.110
105	105	B412,5-2	0.090
106	106	B412,5-3	0.135
107	107	B412,5-4	0.170
108	108	B412,5-5	0.540
109	109	B412,5-6	0.820
110	110	B412,5-7	0.570
111	111	B412,5-8	0.510
112	112	B412,5-9	0.630

113	113	B412,5-10	0.960
114	114	B412,5-11	0.310
115	115	B412,5-12	0.620
116	116	B412,5-13	0.300
117	117	B412,5-14	0.420
118	118	B412,5-15	0.540
119	119	B412,5-16	0.43
120	120	B412,5-17	0.540
121	121	B412,5-18	0.410
122	122	B412,5-19	0.830
123	123	B412,5-20	0.880
124	124	B412,5-21	1.150
125	125	B412,5-22	0.680
126	126	B412,5-23	0.330
127	127	B412,5-24	0.150
128	128	B412,5-25	0.102
129	129	B412,5-26	0.062

This protocol applies only to the samples tested

Chief of laboratory

analytic studies К.R. Tyo

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APPENDIX B. PRE 2008 METALLURGICAL TESTWORK LISTING

During 1998 to 2006, a series of engineering works and heavy industrial testing was undertaken to determine the optimal processing technology and these are detailed in Table 12-1.

Table 12-1: Industrial Testing 1998 to 2006

#	Name	Year
1	Laboratory testing technology of vanadium pentoxide direct pyrometallurgical method for Achpolimetall plant (Kazakhstan);	1999-2000
2	Large-scale technological tests pirometallurgical process for the preparation of vanadium pentoxide on Achpolimetall plant (Kazakhstan);	2000-2001
3	Technological tests produce vanadium pentoxide in parallel with chrome salts on Chilisai plant (Kazakhstan);	2002
4	Technological tests produce vanadium pentoxide in parallel with the rare earths at Ulba Metallurgical Plant (Kazakhstan);	2003
5	Technological tests produce ferrosilicon vanadium- technology electro Chelyabinsk plant and Novokuznetsk ferroalloy plant (Russia);	2003
6	Technological tests use vanadium quartzite as a flux Novodzhambyl phosphorus plant (Kazakhstan);	2012
7	Pilot testing of vanadium quartzite at JSC "Izhora Plants" (St. Petersburg, Russia) direct reduction of vanadium ore solution in cryolite ;	2012
8	Large- development of the technology obtaining standard vanadium ferroalloys FS -45 and FS -65 with parallel dephosphorization pilot production Karaganda Chemical and Metallurgical Institute;	2005
9	Large- test vanadium ore as vanadium flux at the Nizhny Tagil Metallurgical Combine (Russia).	2005
10	Semi- industrial tests and heap leaching quartzites Kara- Tau -based Institute Kazmekhanobr and "Vostokredmet " (Tajikistan) .	2006

12.1.1.1 RESEARCH AND DEVELOPMENT BALAUSA, AUTOCLAVE LEACHING TECHNOLOGY

Table 12-2 sets out in chronological sequence the research and development carried out by FAR to determine optimal industrial chemistry and mineral processing characteristics associated with the pilot plant.

Table 12-2: Research and Development Balausa

#	Name	Contractor	Place	Year
1	Development of technology for the processing of ammonium metavanadate in ferrovanadium FV -80	TOO “Floga”	Almaty Kazakhstan	2007
2	Specifications for ammonium metavanadate produced LLP " The Company " Balausa "	№ 38088316-01-2007, Kometee of standardization of Kazakhstan	Astana Kazakhstan	2007

#	Name	Contractor	Place	Year
3	Research report carbonaceous samples carbon-silicon composite physicochemical methods	Institute of electrochemistry	Almaty Kazakhstan	2008
4	Sanitary-epidemiological conclusion on the safety of radioactive ore Bala Sauskandyk	Kyzil-Orda sanitary inspection	Kyzylorda Kazakhstan	2008
5	Conclusion on carbon black derived from black shales deposit Bala Sauskandyk	Central certification lab of construction materials	Almaty Kazakhstan	2008
6	Pressure leaching vanadium ore Bala Sauskandyk	Stepnogorsk chemical-metallurgical plant	Stepnogorsk Kazakhstan	2009
7	Development sorption technology of complex processing of black shale ores components Big Kara -Tau on large- sample (overall analysis of the results of work performed for the processing of black shale ores Big Kara- Tau deposit Bala Sausqandyk)	National center of mineral resources< lab of vanadium and titanium	Almaty, Kazakhstan	2009
8	Conducting the initial phase of the integrated test enrichment ore Bala Sauskandyk for vacuum aeration installation	National center of mineral resources< lab of vanadium and titanium	Almaty, Kazakhstan	2009
9	Hours of carbon-containing products from tailings leach ore Bala Sausqandyk	Institute Kazmechanobr	Almaty, Kazakhstan	2009
10	Study and development of the technology of carbon from shale Kara- Tau	National center of mineral resources< lab of vanadium and titanium	Almaty Kazakhstan	2009
11	Study and development of technology for integrated extraction of uranium , molybdenum, rare earth elements (REE) , carbon -silicon composite of black shales Kara- Tau with concurrent utilization of flux and potash alum (initial data for designing the plant for processing 500,000 tonnes of black shale ore Bala Sausqandyk)	National center of mineral resources< lab of vanadium and titanium	Almaty Kazakhstan	2010
12	Development of technology for the production of ammonium metavanadate of black shale deposits Bala Sauskandyk	Doctor level of Ayimbetova I. (chief of Balausa lab)	Almaty Kazakhstan	2010
13	Act pilot tests autoclave processing and sulfuric acid sorption from solutions of black shale vanadium vanadium ore.	National center of mineral resources< lab of vanadium and titanium	Almaty Kazakhstan	2010
14	Technological scheme of pilot tests on a representative sample of the deposit Bala Sausqandyk in volume 5000 kg	Central institute of chemical technology	Moskow, Russia, Rosatom	2010
15	Preliminary results of tests node decarbonization, autoclaving and washing and sulfated oxidized black shale ores Bala Sausqanyk	Central institute of chemical technology	Moskow, Russia, Rosatom	2010
16	Initial datas for detail engineering for "The Company " Balausa "	Central institute of chemical technology	Moskow, Russia, Rosatom	2010

#	Name	Contractor	Place	Year
17	Test carbon nanomaterials from ultrafine black shales Kara- Tau (Bala Sausqandyk)	National center of mineral resources< lab of vanadium and titanium	Almaty, Kazakhstan	2010
18	Engineering calculations for tailings ore Bala Sauskandyk	TOO “Bekka”	Semsk, Kazakhstan	2010
19	Development of technology for the production of briquettes from sludge deposits Bala Sauskandyk for ferroalloys	National center of mineral resources< lab of vanadium and titanium	Almaty Kazakhstan	2010
20	Development of autoclave technology of complex processing of black shale ores Kazakhstan (d. Bala Sausqandyk)	National center of mineral resources< lab of vanadium and titanium	Almaty Kazakhstan	2011
21	Conduct pilot testing technology of complex processing black shale deposits Bala Sauskandyk	National center of mineral resources< lab of vanadium and titanium	Almaty Kazakhstan	2011
22	Bond index definition of working procedure of ore deposits Bala Sausqandyk	NII"Kazmekanobre"	Almaty Kazakhstan	2011
23	Laboratory testing of carbon- silicon composite "The Company " Balausa "	RGP "Institute of firing"	Almaty Kazakhstan	2011
24	Autogenous pressure leaching vanadium from black shale deposits Bala Sausqandyk	Central institute of chemical technology	Moskow, Russia, Rosatom	2011
25	Initial data for the installation Designing POX processing of black shales (capacity 15,000 tonnes of ore per annum)	Central institute of chemical technology	Moskow, Russia, Rosatom	2011
26	Research and conduct pilot tests to obtain ferroalloys from black shale deposits of sludge Bala Sausqandyk	Chemical-metallurgical institute	Karaganda, Kazakhstan	2011
27	Development of an integrated processing technology black shale deposits Bala Sausqandyk	National center of mineral resources< lab of vanadium and titanium	Almaty, Kazakhstan	2012
28	Production schedules industrial development ores Bala Sausqandyk	National center of mineral resources< lab of vanadium and titanium	Almaty, Kazakhstan	2012
29	Production of ferro-vanadium , enterprise standard for LLP " The Company " Balausa "	TOO “Floga”	Almaty, Kazakhstan	2013
30	Production schedules for the production of ferro-vanadium FV - 50 of ammonium metavanadate for LLP " The Company " Balausa "	TOO “Floga”	Almaty, Kazakhstan	2013
31	Work organization project area receiving ferrovanadium FV -50 " The Company " Balausa "	TOO “Floga”	Almaty, Kazakhstan	2013
32	Investigation and determination of optimal conditions agglomeration black shale deposits of sludge Bala Sausqandyk by briquetting	Chemical-metallurgical institute	Karaganda, Kazakhstan	2013

APPENDIX C. JORC 2012 – TABLE 1

SECTION 1 SAMPLING TECHNIQUES AND DATA

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* • *Aspects of the determination of mineralisation that are Material to the Public Report.* • *In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.*	• FAR's main sampling technique was by surface core drilling and the quality was acceptable for appropriate assessment of the resource. • Former Soviet-era sampling from the 1940s to 1960s of the surface exposures included continuous cross-strike channel samples and channel sampling from shallow u/g excavations: quality unknown but it showed a vast resource with a strike length of several kilometers (sample results not used for resource estimates). • In the 1970s,Soviet-era shot-drilling and single-barrel diamond drilling had poor core recovery but provided information on the primary zone (95%+ of the total resource) beneath the oxide cap. All holes gamma logged. • In a 1990-91 drilling campaign however, (section 4.5.1.4 of report) the reported soundness of the 1970s V_2O_5 grades were confirmed and FAR's twin drilling also supported the credibility of the historical results.
Drilling techniques	• *Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	• Diamond drilling using double tube core barrels with a minimum core diameter of 47.3mm (NQ) to 63.5mm (HQ) within the target mineralised zone. Gyro inclinometry was used to trace the drillhole after completion, though drill holes were basically straight.

Criteria	JORC Code explanation	Commentary
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.* • *Measures taken to maximise sample recovery and ensure representative nature of the samples.* • *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	• The contractor provided a 90%+ core recovery within the mineralised horizon: core from each run was re-assembled and measured to determine the recovery against the actual run length. This was supervised by FAR's geologist. • The drill site geologist was able determine when the drillhole was near the target depth and the driller was able to reduce the bit pressure when needed and take shorter runs to help maximize recovery in softer zones. • No discernible relationship was found between core recovery and grade and, any detrimental bias considered to be negligible.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* • *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.* • *The total length and percentage of the relevant intersections logged.*	• Core has been geologically logged and is appropriate for resource estimations. A separate core drilling programme for geotechnical and metallurgical samples has yet to be implemented. • The historical drillhole logs (passports) from the 1970s are extremely comprehensive, and include gamma log results and V_2O_5 sample grades through the orebody. GMR used these to confirm that the sample grades in the database are correct. FAR's drillhole logs, include code abbreviations to record the geological features of the core, and also record each sample number with depth interval. A graphical log is also produced, showing rock-type symbols. These logs are stored in a spreadsheet format. Core photography was also undertaken. • 100% of the mineralized zones were logged, but generally, only logged about 2m orthogonally from the contact into the surrounding country rocks, as there is a sharp grade cut-off.

Criteria	JORC Code explanation	Commentary
Sub-sampling Techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.* • *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.* • *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* • *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* • *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.* • *Whether sample sizes are appropriate to the grain size of the material being sampled.*	• FAR's half core samples were cut by a saw, under the supervision of the geologist to ensure unbiased sampling. See section 4.5.2.4 in report. • Non-core appeared to be extremely rare, as based on site visits by the CP and from photographic evidence. • Descriptions of the historic drillhole sampling are not available, but standard Soviet-era practice involves production of half core samples using a saw, for this type of deposit. • FAR's core was oriented and marked, prior to cutting, to produce identical halves. Half core samples were crushed to 2 mm (crushers 1, 2 and 3 are shown in Figure 4-18), mixed and split using a Jones splitter and a 400 g fraction is extracted for pulverization. The rest of the sample was stored as a coarse reject for future additional analyses. The 400 g samples were ground to -200 mesh (74 µm) and 2 x 200 g samples sent to external accredited laboratories in Australia for analyses. • Sample sizes are appropriate to the primary zones' mineralised grain size.
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.*	• The first batch of pulped core samples, weighing 200g, from FAR's exploration programme, totalling 110 samples, was sent to Intertek in March 2011. The CP requested that Intertek test the validity of using a cheaper multi-acid digestion, vis-à-vis the more expensive sodium peroxide fusion approach, as used in 2009 (see section 4.6.2.1 in report). A comparison of results on 20 test samples confirmed that both methods were similarly accurate in determining the total contained V, Mo, and U elements, and the four-acid digestion approach was therefore adopted for all FAR's exploration sample analyses. A further two batches of samples were sent to Intertek: giving a total of 370 samples. To confirm the

Criteria	JORC Code explanation	Commentary
	• *For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.* • *Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	Intertek laboratory results, for external QA/QC assessment, a number of random sample pulp duplicates were re-analysed by the internationally accredited laboratory Ultra Trace, also in Perth, amounting to 14 % (51 samples) of the Intertek samples (see Figure 4-22). Results show excellent precision and accuracy. • FAR installed modern XRF equipment to specifically analyse elements on site. The XRF system is very accurate in quantifying the contained metals and is the preferred technique for the determination of the major oxide elements, as well as some trace elements. However, it needs careful preparation of the samples, plus calibration of standards and matrix corrections and adjustments for drift etc., to avoid false readings. At Balasausqandiq, a press powder technique, vis-à-vis fusion technique, is used but is subject to particle size and matrix effects which increase analytical uncertainty, resulting in lower grade results. However, when the above problems were recognised, sample homogenization improvements by finer sample grinding, for the press powder technique, has improved the XRF accuracy of the results, according to FAR. • Although the CP had made no site visits to the accredited Australian laboratories, extensive telephone discussions with the laboratory managers, showed that their procedures on QA/QC were of a high standard.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.* • *The use of twinned holes.*	• Routine inspection of assay grades with drill logs and remaining half-core, by FAR’s chief geologist and checks by the CP, confirmed that the weighted intersection grades of the primary zone Orebody 1 (OB1) beneath the oxide cap, are relatively uniform, see Table 4-8, and no significant intersections were found. • FAR drilled a twinned hole, to test the validity of grades in an historic drillhole, from the early 1970s, where there was low core recovery: and the assay results corroborated the historic vanadium results -See

Criteria	JORC Code explanation	Commentary
	• *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* • *Discuss any adjustment to assay data.*	section 4.7.5 of report. • FAR retrieved historical data, which included sampling results and drillhole logs, and transcribed for computer storage - this primary data of surface trench sampling, u/g exploration results and diamond drilling were imported into Datamine for downstream computerisation processes. Also, all FAR's generated primary data also computerised and imported into Datamine. This also allowed verification of the data, coupled with routine electronic backups and onto disk drive storage. • No adjustments were made to the assay results.
Location of data points	• *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	• The former Soviet-era maps and plans do not display coordinate data and FAR undertook an extensive topographical re-survey of the Balausa area, which included locations of drillhole collars, sample trenches, adits, shafts and sample pits, and infrastructure. The survey data were compiled in MapInfo, from UMT (WGS84) survey measurements, and from which AutoCAD drawings were generated. These AutoCAD plans were directly imported into Datamine and used for generating topographic wireframe surfaces (digital terrain models). FAR's down-the-hole surveys had an acceptable accuracy. • There were a number of problems in locating the correct historical drillhole standpipe collars for the 1973 exploration holes during field surveys, because of indecipherable or absent identification numbers and, especially, where there was confusion from unknown drillhole collars related to the extensive 1990s drilling campaign: but after diligent examination of the 1973 archived plans and maps, together with the new DM topographic model, it was possible to locate with reasonable certainty, the actual drillhole locations. Based on GMR's independent GPS audit measurements of a number of FAR's surveyed drillhole locations, these were accurate and correlated with the historical plans. However, for a very few drillholes, it was not

Criteria	JORC Code explanation	Commentary
		possible to exactly reconcile the field survey with the historical plans and topographic surface, and this discrepancy was less than a maximum horizontal displacement of 25 m, which was considered to be within acceptable limits for this resource project. .
	• *Specification of the grid system used.*	• FAR's local grid system was made parallel to the old historic grid, as based on the Soviet Gauss Kruger (Pulkovo 42).
	• *Quality and adequacy of topographic control.*	• The final topographic models proved to be of a high standard.
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.*	• Experimental variograms of historical surface trench-grade results in the oxide cap, were generated to help plan the drillhole spacing for the exploration programme. These 1946 oxide surface trench samples provided sufficient density coverage to generate acceptable experimental variograms along strike. Although, as expected for the oxide zone, the nugget effect appears quite high relative to the sill, it does indicate that the historical 50m trench spacing along strike was conservative. See section 4.10 in the report. Extensive variogram modelling of the primary zone drillhole sample grades for V2O5%, show that the nugget effect is typically only 10% of the variogram sill, reflecting a high level of grade consistency. These model parameters were used to help form the search ellipsoids for grade interpolations. See section 4.10.2 in the report for details.
	• *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.*	• The data spacing was sufficient to establish geological and grade continuity of the primary zone for mineral resource estimation and classification accordingly.
	• *Whether sample compositing has been applied.*	• All samples were composited to 2m lengths from an average sample length of 1m.

Criteria	JORC Code explanation	Commentary
Orientation of data in relation to geological structure	- *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* - *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	- Although some minor drillhole orientations can be inconsistent in relation to the mineralised structure, this did not affect the unbiased nature of the sampling, because FAR overcame the complex geometry of the folded structure by generating geometric sub-domains (SW Limb), each with a consistent strike and dip, for the block model, prior to grade estimations. See Figure 4-55 and section 4.12.1 of the report. - The sampling bias was nullified – see the above comments.
Sample security	- *The measures taken to ensure sample security.*	- Soviet-era sample security, although not documented, would have been a high priority and FAR's assessment of analytical results suggest no security issues. FAR's sample security was supervised by the chief geologist, and sample preparation was on site in a secure laboratory. Shipment of pulverized samples to recipient analytical laboratories in Australia and Central Asia were also secure including changing sample numbers and no discrepancies were observed.
Audits or reviews	- *The results of any audits or reviews of sampling techniques and data.*	- Internal audits were conducted by FAR of historical data – see section 4.7.3 of report. Results showed some drillhole discrepancies of up to 25m from the true location and related to the identification of the correct historical drillhole location.

SECTION 2 REPORTING OF EXPLORATION RESULTS

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* • *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	• The mineral tenement is 100% held by FAR through its wholly owned subsidiaries. • There are no land restrictions to the site, other than state observance of subsoil extraction regulations. There is no known security or threat of loss issues with the tenement.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• Refer to section 4.2 of the report for a detailed summary of the historic exploration programmes of the Soviet era to the present.
Geology	• *Deposit type, geological setting and style of mineralisation.*	• This deposit has geological characteristics in common with vanadium deposits in South China, when in primordial times the Karatau mountains were juxtaposed within the same supercontinent. The stratiform vanadium layer is associated with five very large orebodies and their surface expression can be traced for about 40 km. These orebodies are mostly confined to deep synclinal folds, where the primary carbonaceous vanadium rocks at depth are protected from weathering and oxidation processes. From historical data and from FAR's drilling results, the global grades within these orebodies are relatively similar, and this uniformity is testament to the broad stable conditions during mineralogical deposition in a marine basin some

Criteria	JORC Code explanation	Commentary
		510 million years ago (mid-Cambrian).

Drill hole Information	- *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:*
	  ○ *easting and northing of the drill hole collar*
	  ○ *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar*
	  ○ *dip and azimuth of the hole*
	  ○ *down hole length and interception depth*
	  ○ *hole length.*
	- *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*
		- Graphical statistical information on the drillhole samples (Figures 4-44 to 4-50), Figure 4-5 showing a 3D Datamine model profile and a cross section profile of drillholes (Figure 4-39), in the report, all provide critical information on the understanding of the exploration results.
		- Referring to the above comments, tabulations of the drillholes are considered to be unnecessary, but if required these can be obtained from FAR.

Data Aggregation methods	- *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.* - *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* - *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	- No data aggregation was undertaken, with all 2m composited sample results used individually to represent the mineralisation characteristics. Quantile analysis was used to assess the U_3O_8, as the coefficient of variation was very high and this statistic indicated that there were some anomalously high grade samples which caused this high ratio. Results of the quantile analysis suggested that top-cutting to 0.1 % was appropriate and Table 4-52 in the report shows an acceptable coefficient of variation ratio. It also resulted in a change in the correlation coefficient from random to a weak correlation trend with V_2O_5. - Metal equivalents are based on indicated and inferred resources of OB1. These by-products are based on their economic sales value in terms of the equivalent V_2O_5%. See Table 1-6.

Criteria	JORC Code explanation	Commentary
Relationship Between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* • *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	• The main mineralised zone OB1, is basically a narrow synclinal fold structure and approximate true widths of the sub-vertical side walls can be projected from the surface exposures to depth: and this has been confirmed from the drilling. Additionally, vertical drillholes which intercepted the keel area of the syncline showed the mineralised zone to be abnormally thicker, vis-à-vis the flanks. This thickening is also seen at the nose of the syncline. See Figure 4-56 showing 3D perspective of OB1. Although only some of the drill intercepts on the flanks were orthogonal to the mineralised structure, interactive computer modelling has produced an accurate geometric interpretation of OB1.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	• There are a number of diagrams and photos to illustrate the mineralised structures, plus a plan view showing the geology on a Google Earth map (Figure 4-9). Also, refer to Figure 4-10, Figure 4-11, Figure 4-14 and Figure 4-16 showing an unexpected vanadium intersection at 100m depth for OB2.
Balanced reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	• Although all exploration results have not been described, the report does show a balanced perspective and should not lead to any misconceptions.
Other Substantive Exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	• All relevant data have been described and included in the report.

Criteria	JORC Code explanation	Commentary
Further work	- *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* - *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	- Future work recommendations have been described in section 4.20.

SECTION 3 ESTIMATION AND REPORTING OF MINERAL RESOURCES

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	- *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* - *Data validation procedures used.*	- Computerised records of various attributes from resource tables to drillhole geology logs and assays, were compared with a random selection of scans from the historical documents and the level of transcription errors were assessed to be low. Confidence in the correct replication of the data in the computerised format, used to evaluate the deposit, is therefore acceptable. - In Datamine, continual checks of the data, during various stages of computer processing, were compared with primary clean databases.
Site visits	- *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* - *If no site visits have been undertaken indicate why this is the case.*	- 8 to 11 April 2007: Detailed review of operations with technical staff, including field visits with extensive geological examination of orebody related surface exposures. Collected numerous computerised historical documents and maps, plus translated technical reports. Captured and documented GPS locations of the plant and o/p operations. - 4 to 7 June 2009: Reviewed open pit field operations, especially grade control channel sampling and results of analyses in relation to

Criteria	JORC Code explanation	Commentary
		the exposed vanadium layers and geological sample maps. Visited numerous historical sample locations from the 1940s. Established analytical requirements for the FAR exploration programme. • 8 to 11 January 2010: Reviewed progress with technical staff, about diamond drilling, logging, mapping and sampling procedures. Site visits included the new sample preparation laboratory. • 14 to 20 August 2010: Numerous field visits to review the progress of the FAR exploration diamond drilling programme resulting in upgrading a number of procedures to facilitate quality control. Established a requirement for local co-ordinate system to be very accurately correlated with the historical grid and the UMT (WGS84) system. • 8 to 11 December 2010: Continued to monitor the exploration programme and to check that the core sampling and sample preparation procedures are to acceptable international standards.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	• The data used for geological interpretation have been extensive, from both historic and FAR's drilling campaigns and there is a high confidence level in the interpretation for OB1. • The range of the data employed has been extensive, as clearly shown in the report and assumptions were unnecessary. • Because of the consistency in the V_2O_5 grades and the established continuity of the mineralised primary zone, no alternative interpretation has been necessary. • Local changes in dip and strike of the primary zone were overcome by using sub-domains with similar dips and strike, as detailed in the report. • No obvious factors affecting continuity.

Criteria	JORC Code explanation	Commentary
Dimensions	- *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	- OB1 has a strike length of 4.5km and a maximum depth of 200m to the upper contact. Flank thicknesses ranges from 5m to 9m approx. with variable thickening of the mineralised zone at the bottom of the anticline. Note: OB2 to OB5 (see map of these zones Figure 4-9) have been explored during the Soviet era and these have a combined strike length of 21km.
Estimation and modelling techniques	- *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.*	- Ordinary Kriging (OK) was selected as the most appropriate geostatistical approach for estimating the grades into the block model (split into the SW and NE limb submodels) using the modelled variogram parameters as presented in Section 4.10 (see Table 4-55 and Table 4-56). Both V_2O_5 and C were estimated using OK, but for MoO_3 and U_3O_8 only IPD grade interpolations were realistic and IPD for C was required for the SW limb submodel because valid experimental variograms could not be generated due to limited sample assays. Variography results for V_2O_5 exhibited good ranges along strike. For the SW limb, strike 650m, downdip 130m and 15m across the mineralised zone. For the SE limb, strike 550m, downdip 75m and 15m across the mineralised zone. For estimation processing, Datamine software was used.
	- *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* - *The assumptions made regarding recovery of by-products.* - *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).*	- Check estimates using IPD grade interpolations were used to compare with the OK estimations for V_2O_5 and results were very similar: but IPD results were not used for the final resource model. - All products have been estimated as detailed above. - No deleterious elements of economic significance.

Criteria	JORC Code explanation	Commentary
	- *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.*	- Testing of the most appropriate block size for allowing sufficient sensitivity, yet keeping the number of blocks to a low optimal level resulted in a fundamental block size of 40 m x 20 m x 20 m (X, Y & Z directions) and these parent blocks were split at the contacts of the orezone wireframe model, according to splitting criteria, resulting in a block dimension of 5 m along the X (easting) direction, 2.5 m along Y (northing) direction and along the Z (elevation) direction the blocks were split exactly at the wireframe contact, resulting in a minimal block dimension of 0.1 m to a maximum dimension of 20 m (mean length of 10.33 m). No subzone identifiers were required for this block model, though later surface oxide blocks would be removed to form a final primary orebody model. Due to the somewhat uniform grade distributions of the commercial products, these variable block sizes are not expected to induce grade biases.
	- *Any assumptions behind modelling of selective mining units.*	- None
	- *Any assumptions about correlation between variables.*	- Good correlations are found between V_2O_5 and Carbon and a weak correlation between U_3O_8 and V_2O_5 within the primary zone.
	- *Description of how the geological interpretation was used to control the resource estimates.*	- See the Geological interpretation above.
	- *Discussion of basis for using or not using grade cutting or capping.*	- Grade cutting or capping of V_2O_5 is not necessary, as grade levels are consistently similar for OB1 within the primary zone, with a sharp grade drop into the waste rocks of the hanging wall and footwall: see grade/ tonnage curve for total V_2O_5 (Figure 4.67). Note: there was cutting of U_3O_8 grades as previously discussed.

Criteria	JORC Code explanation	Commentary
	• *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	• Statistical checks on the grade estimates, visual checks with drillhole data and a detailed investigation of the swath plots (Figure 4-56 to Figure 4-63) were used to validate the model.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	• The tonnages are determined on a dry basis.
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	• All material above zero was reported.
	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	• The mineralisation is amenable to extraction by conventional open pit mining. The mineral resource does not include any mining dilution.
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	• The mineral resource estimate has been based on extensive metallurgical test work, as described in Section 6 of the report. This has been corroborated by the actual process plant performance from the onsite plant.

Criteria	JORC Code explanation	Commentary
Environmental factors or assumptions	- *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	- The mineral property has been assessed by preliminary EIA in accordance with the State of Kazakhstan requirements.
Bulk density	- *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* - *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.* - *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	- The bulk density of 2.4 was used in the resource estimation: see section 4.6.7 of the report for details.
Classification	- *The basis for the classification of the Mineral Resources into varying confidence categories.*	- The JORC classification of the model was based upon search ellipsoids which were used to define JORC ‘Indicated’ and ‘Inferred’ resources. The processes were quite involved and details are presented in section 4.12.2 (SW Limb estimation), section 4.12.3 (NE Limb Estimation) and section 4.12.4 (Grade Block Model) of the report.
	- *Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values,*	- All these factors were addressed.

Criteria	JORC Code explanation	Commentary
	quality, quantity and distribution of the data). • *Whether the result appropriately reflects the Competent Person's view of the deposit.*	• The results do reflect the Competent Person's view of the deposit.
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	• Internal reviews of the resource have been undertaken.
Discussion of Relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	• The CP is confident in the accuracy of the Mineral resource, as supported by graphical and various statistical investigations. Additionally, strong mineralization continuity and consistent grade distributions support this confidence in the global resource for the primary zone of the OB1 model.

SECTION 4 ESTIMATION AND REPORTING OF ORE RESERVES

(Criteria listed in section 1, and where relevant in sections 2 and 3, also apply to this section.)

Criteria	JORC Code explanation	Commentary

Not Applicable

SECTION 5 ESTIMATION AND REPORTING OF DIAMONDS AND OTHER GEMSTONES

(Criteria listed in other relevant sections also apply to this section. Additional guidelines are available in the ‘Guidelines for the Reporting of Diamond Exploration Results’ issued by the Diamond Exploration Best Practices Committee established by the Canadian Institute of Mining, Metallurgy and Petroleum.)

Criteria	JORC Code explanation	Commentary

Not Applicable