SNOWDEN

FIRE RIVER GOLD

Nixon Fork Project
Alaska, USA

Preliminary Economic Assessment

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1 Summary

Fire River Gold Corp. (FAU) requested Snowden Mining Industry Consultants Inc. (Snowden) prepare a Preliminary Economic Assessment (PEA) of the underground mining at FAU's Nixon Fork Project in Alaska, USA. This assessment focuses on the underground potential only, and does not consider economic potential that could be derived from historical tailings processing in a new Carbon in Leach (CIL) circuit. The economics of this additional feed has not been combined with the economics stated in this report. Details on the economics of processing the historical tailings can be found on SEDAR in a report posted on November 8, 2010, entitled "Technical Report on the Nixon Fork Mine Project Medfra Quadrangle, Alaska" by Flanders, Giroux, and Ravensthorne.

The PEA was to be reported in the form of a Technical Report prepared under the guidelines of the Canadian National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101). This report constitutes a preliminary assessment as provided for under clause 2.3 (3) of NI 43-101.

This assessment is preliminary in nature and includes the assessment of some Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves. There is no certainty that the evaluation reported in this preliminary assessment will be realised. This report is limited in scope to the potential production from the first 24 months of the project.

1.1 Location

The Nixon Fork project is located on the northeastern edge of the Kuskokwim Minerals Belt (KMB) of southwestern Alaska. The KMB covers an area of roughly 190,000 km2 with significant gold and mercury occurrences. In this belt, the Nixon Fork mine is situated between two regional northeast trending structures, the Denali - Farewell fault system to the south and the Nixon Fork - Iditarod fault to the north.

1.2 Geology and mineralisation

The Nixon Fork project is situated in an area of moderate topographic relief with elevations ranging from 300 to 460 meters. Ridges are generally rounded and are forested with black spruce, larch, birch, and alder. Lower elevations are often poorly drained due in part to discontinuous permafrost conditions, often being covered by soft muskeg and stunted black spruce forest. Outcrops are rare in the mine area with a few resistant knobs along ridgelines.

1.3 Exploration and mining history

Mineral exploration and development has occurred at the Nixon Fork project in several discrete phases since it its discovery nearly 100 years ago. From 1920 to 2007, the mine produced (from all operators) 185,300 ounces of gold, 19,566 ounces of silver, and 1,273,066 pounds of copper. The silver and copper production figures are incomplete and should be considered minimum production estimates. Recent underground mining at Nixon Fork occurred during two intervals from 1995 to 1999 and in 2007.

1.4 Drilling and sampling

Exploration and development drilling at Nixon Fork since 1985 included more than 100,000 meters of drilling in more than 1200 surface and underground drill holes. Except for 7,341 meters of RC drilling in 85 holes, all of the drilling at Nixon Fork has been by diamond core drilling. National Instrument 43-101 resources at Nixon Fork were published in 2006.

1.5 Mineral Resources

Mineral resource estimates have been completed by Gary Giroux, MASc., P.Eng. using all available data. These mineral resource estimates are not influenced by any known negative factors. These are all categorized as a Mineral Resources without demonstrated economic viability.

Gary Giroux, is of the opinion that the current estimate for the underground deposits can only be classified as Indicated and Inferred Resources. The estimate conforms to CIM standards for reporting mineral resources and reserves.

The current estimate for lode gold in the Nixon Fork project includes Indicated Resources of 121,690 tonnes grading 26.88 g/t for a total of 105,168 ounces using a 10 g/t cutoff. Inferred Resources total 70,780 tonnes grading 27.80 g/t totalling 63,257 ounces also at a 10 g/t cutoff.

1.6 Preliminary Economic Assessment

An investigation of the underground potential was made to identify potentially economic underground inventories for the next two years. The study concluded that there were significant underground inventories in the Crystal, 3100, Southern Cross, J5, and Mystery areas. On the instruction of FAU, Snowden limited its assessment for the first 24 months of production only. Detailed analysis of these inventories for a variety of extraction methods concluded that using selective mining techniques such as shrinkage or cut and fill presented the best potential for underground production.

An optimized schedule was generated, for the first 24 months of production, so that the inventory and schedule that returned the highest economic value could be reported in this preliminary assessment. For the first 24 months of production Snowden found potentially economic inventories in the Crystal, Mystery and Southern Cross areas, as these areas represented the highest grade for the least development. A schedule of greater duration would likely include additional material from the Mystery zone in the economic inventory.

Conceptual underground designs based on mechanized techniques with decline access have been prepared from the optimization results. The potentially economic inventories derived are shown in Table 1-1. The resource model that this assessment was based upon was depleted for previous workings in the upper part of the Crystal zone only. There is limited historical information about previous workings and Snowden understands that FAU is in the process of determining the full extent of previous mining activity in order to better quantify impact on the inventories identified in this report.

Table 1-1Potentially economic inventories for the two year plan

Area	Tonnes kt	Grade Au g/t
Crystal	87.5	30.6
Southern Cross	1.39	19.2
Mystery	12.4	28.3
Total	**101.3**	**30.2**

After applying appropriate estimates for capital costs, the optimized cash flow yielded the financial outcomes presented in Table 1-2. In Table 1-2 the results are presented for three scenarios (each of which is supported by a different schedule). The first scenario uses the gold selling price requested by FAU of $US1,200/tOz. With the exception of the discussion on sensitivities (Section 18.5.1) this commodity price has been used for the remainder of this report. The second scenario (in Table 1-2) uses a gold selling price of $US1,033/tOz which, as of December 2010, is the three year rolling average gold selling price, and the third scenario uses a gold price of $1,500/tOz as an upside example. The Key Performance Indicators in Table 1-2 (KPI's) reflect the sum of operating and capital expenditures including vendor royalties, but exclude financing, government royalties and taxes. The currency is USD.

Table 1-2 Summary of financial model for two year plan

		Gold price ($US/tOz)		
Item	**Units**	**1,033**	**1,200**	**1,500**
Undiscounted cash flow	$USM	47.81	64.28	93.63
NPV @ 5% discount	$USM	45.30	60.94	88.86
IRR	%	462	549	853
Payback period	Months	4	3	3

From Table 1-2 it can be seen that the IRR's are high, this is because of the high cashflow with respect to the capital. The capital is relatively low because this PEA is assessing the impact of underground production on an existing operation which is financed independently of this assessment. That existing operation involves the upgrade of the plant at Nixon Fork so that it can re-process tailings using leaching. The cost associated with the acquisition and upgrade of the plant and facilities is not part of this assessment. Sensitivity analyses determined that the Project is most sensitive to gold price/process recoveries, moderately sensitive to process and site operating costs and development capacity, and least sensitive to mining and development costs.

1.7 Conclusions and recommendations

It can be concluded from the current study that there is potential for a profitable underground project to be established at Nixon Fork for the first 24 months of production.

It is recommended that FAU continues with its evaluation of the Nixon Fork Project and progresses towards undertaking a Prefeasibility Study to address the remaining material project uncertainties, or commence test mining and processing underground material to demonstrate the economics.

Resource estimation recommendations:

- maintain a substantial ongoing exploration program so that reserves can be replaced as they are depleted by mining.
- determine full extent of previously mined material to appropriately deplete the resource model as uncertainty about depletion of mined material places a high level of uncertainty on the results of this preliminary assessment
- undertake a drilling program so that more of the Resource can be classified as Measured or Indicated which may then be converted into Reserves after completion of a Prefeasibility Study.
- review the resource confidence classification criteria for future Resource estimates and ensure that all aspects affecting confidence in the Resource estimation are considered, including geological understanding, complexity, and continuity, the sample data density and orientation (including sample grades and bulk density data), the data accuracy and precision as established through the QAQC programs, grade continuity including the spatial continuity of mineralisation, the quality of the estimates, and the results of the estimation validation.

Metallurgical

- One of the principle driving forces for the high cut-off grade (COG) at Nixon Fork is the low processing rate (which gives rise to high unit costs). Snowden recommends that Nixon Fork investigate cost effective alternatives to increase the mill throughput. By increasing mill throughput, COG's can be reduced and the size of the resource above cut-off will be substantially increased - for example the mining inventory above a cut-off 10g/t is almost double that of 15g/t.

Other

- Handling the moderate water inflows derived from mining below the water table is important for the sustainable exploitation of these resources. In the schedule this represents about 50% of the mined inventory. It is therefore recommended that FAU proceed with evaluation and the implementation of the identified options for dealing with underground water inflows.

2 Introduction

FAU requested Snowden to prepare a PEA to assess the underground potential of FAU's Nixon Fork Project in Alaska, USA for the first 24 months of production. The PEA was to be reported in the form of a Technical Report prepared under the guidelines of the Canadian National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101). This report constitutes a preliminary assessment as provided for under clause 2.3 (3) of NI 43-101. This assessment focuses on the underground potential only, and does not consider economic potential that could be derived from historical tailings processing in a new CIL circuit. The economics of this additional feed has not been combined with the economics stated in this report. Details on the economics of processing the historical tailings can be found on Sedar in a report posted on November 8, 2010, entitled "Technical Report on the Nixon Fork Mine Project Medfra Quadrangle, Alaska" by Flanders, Giroux, and Ravensthorne

This assessment is preliminary in nature and includes the assessment of some Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the evaluation reported in this preliminary assessment will be realised.

All currency values in this report are in USD unless stated otherwise.

Unless otherwise stated, information and data contained in this report or used in its preparation have been provided by FAU. Nixon Fork is the subject of past Technical Reports prepared November 8th, 2010 by Flanders, Giroux, and Rawsthorne.

The Qualified Persons for preparation of the report are Anthony Finch of Snowden who visited the project site on August 3, 2010, and Gary Giroux of Giroux Consultants Ltd., who visited the site in early May 2010 and Richard Flanders, of Ridgerunner Exwho visited the site in mid-March, 2010.

The responsibilities of each author are provided in Table 2-1.

This report is intended to be used by FAU subject to the terms and conditions of its contract with Snowden. That contract permits filing this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities laws, any other use of this report by any third party is at that party's sole risk.

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Table 2-1 Responsibilities of each co-author

Author	Responsible for section/s
Anthony Finch	1, 2, 3, 17.2, 18 and 19
Richard Flanders	4,5,6,7,8,9,10,11,12,13,14,15
Gary Giroux	17 (except 17.2)
Timothy G Smith	16

3 Reliance on other experts

This report includes findings based on the available information and geologic interpretations as provided by FAU. The authors have relied on this data and have no reason to believe that any material facts have been withheld.

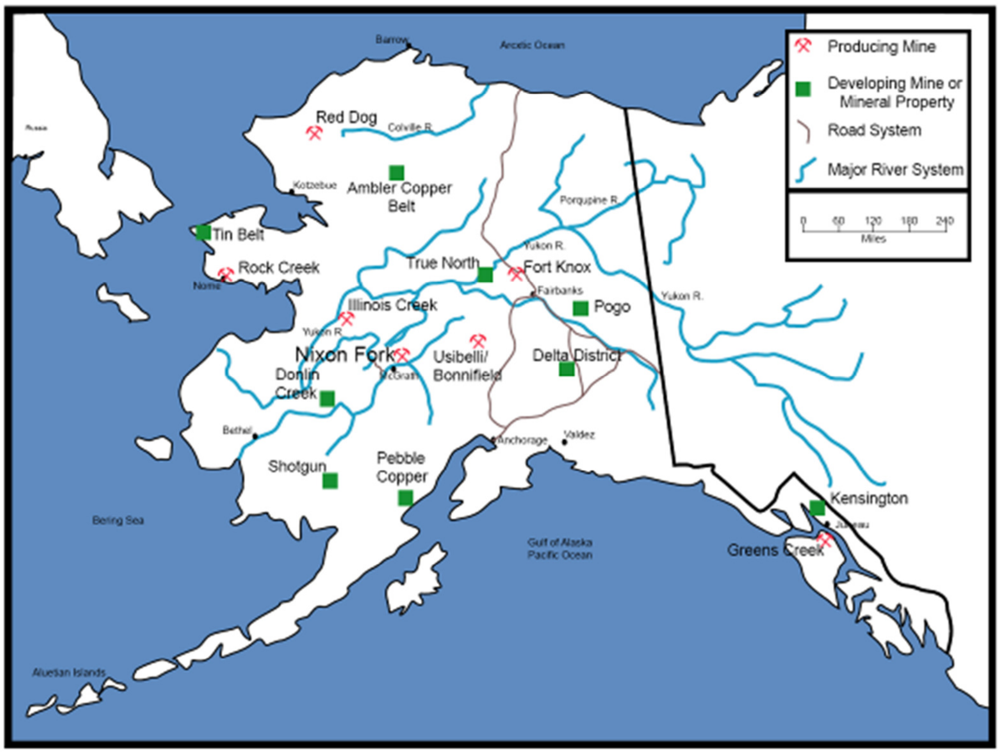
4 Property description and location

4.1 Location

The Nixon Fork Project is located in central Alaska, within the United States of America. Flanders, Giroux & Rawsthorne (2010, p.4) state "the mine is located in the Medfra A4 quadrangle and is centered at 63° 14'N, 154° 46'W, 56 km northeast of McGrath, central Alaska."

The project location is shown in Figure 4-1

Figure 4-1 Project Location (source - Flanders Giroux & Rawsthorne 2010, p.4)



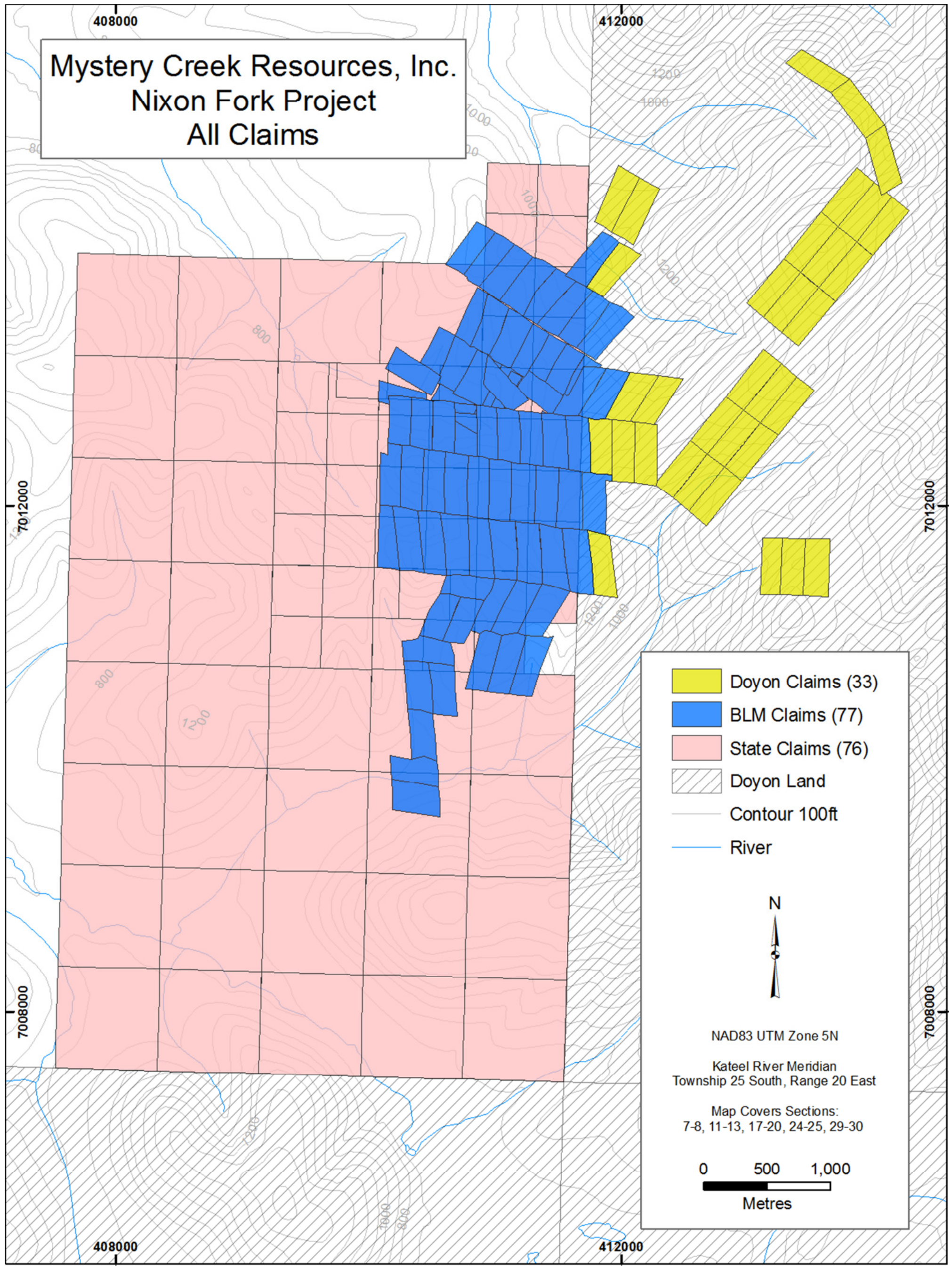
4.2 Type of mineral tenure

The mineral tenure is described in the Technical Report by Flanders, Giroux & Rawsthorne (2010, p.4).

> The Nixon Fork property consists of 95 unpatented federal lode and 15 placer claims and (2,200 acres) an additional 77 State of Alaska mining claims (8,800 acres) located in Township 26 South, Range 21-22 East, Kateel River Meridian.
>
> The claims are registered with the U.S. Bureau of Land Management and the Alaska Division of Mining, Land and Water Management.

The Nixon Fork Mine Claims are shown in Figure 4-2

Figure 4-2 Nixon Fork Mine Claims



4.3 Royalties and agreements

Flanders, Giroux & Rawsthorne (2010, pp. 4-6) report:

> Annual federal mining claim rental payments of $13,750 were paid in 2008 and 2009 and a payment of $15,400 was also paid before August 31, 2010. Annual State mining claim rental payments of $8,795 were paid in 2008 and 2009 and affidavits of annual labor for State mining claims were timely recorded and will become due and payable again before November 30, 2010.

Federal claim rental payments are paid to the US. Bureau of Land Management for claims wholly or partially on federal lands (Range 21 East) and to Doyon, Ltd., the Regional native corporation in this part of Alaska, for federal claims wholly on Doyon land (Range 22 East).

At the time of statehood, the State of Alaska was given the right to select 104 million acres from the "public lands" that were then managed by the Bureau of Land Management (BLM). The State of Alaska began evaluating and selecting lands with the first million acres being lands to generate revenues for a mental health trust that had been established while Alaska was still a territory. The selection process continued until January 1, 1994, which was the statutory end date for completion of selections. Approximately 90 million acres have been tentatively approved (TA'd) for transfer to the State. In the case of the Nixon Fork property, Township 26S Range 21E is a state selected township that has yet to be transferred, and Township 26S Range 22E has been deeded to the native controlled Doyon Ltd.

As a result of this transaction, 33 of the 110 Federal claims are located on land administered by Doyon Ltd, a native corporation that obtained the land through the Alaska Native Claims Settlement Act (ANCSA). Tenure requirements for the Federal claims located on Doyon Land are the same as for those on Bureau of Land Management (BLM). A Federal claim, once granted, is valid until the following August 31. The annual cost to maintain a Federal claim is $140 paid to the BLM. Failure to pay the assessment fee in a timely manner results in the loss of those mineral rights.

There are currently 44 State of Alaska claims staked “at risk” that overlay the federal claims on the BLM ground located in Township 26N, Range 21E which is a TA'd Township. The State claims will only become active if the federal claims are abandoned and the state is conveyed title to the land underlying the claims. There are also 4 State claims on Doyon land, Township 26N Range 22E and an additional 58 expired State claims that form an area of interest clause in the agreement between St Andrew and the owners. Through 2009 State claims required annual assessment work of $100 per 40-acre claim work and an annual rental fee that commenced at $25 per claim, escalating to $55 per claim after 5 years and $130 per claim after the 11th year. All the State claims have been held in excess of 11 years. Starting in September, 2009 the State of Alaska raised these fees to $35, $70, and $170 respectively. These fees were all paid within the required time periods.

A mining license tax (MLT) is payable on all production from State, federal or private lands in Alaska (Borell, 2009). This tax is on a net profits basis with a grace period for the first 3.5 years of production. If annual net income is less than $40,000, there is no MLT. The tax varies from 5% if annual net income is between $40,000 and $100,000 up to 7% if annual net income is above $100,000. In addition, there is also a 3% production royalty calculated on the same net profits basis as the mining license tax that applies to production from State lands. The claimholder may convert the State claims at any time to a lease which is subject to the same rental and production royalties as the claims but grants specific rights of tenure.

Mineral rights in this part of Alaska are administered by the BLM and the State of Alaska. The claims of the Nixon Fork project have not been surveyed by a registered land or mineral surveyor and there is no State or federal law or regulation requiring such surveying.

During late March, 2009, a review of State and federal permits covering the Nixon Fork property was conducted by the Alaska Department of Natural Resources. Representatives of Fire River Gold and their consultants attended this meeting (Freeman, 2009). No significant permit compliance issues or notices of non-compliance outstanding against the permits were identified at that meeting.

On February 4, 2003, Mystery Creek Resources Inc., then a wholly owned subsidiary of St. Andrew Goldfields, entered into a long-term the lease on the Nixon Fork property with the owner, Mespelt & Almasy Mining Company LLC. Provisions of the lease are outlined below:

1. Exclusive and unrestricted 10 year term renewable upon written notice from lessee
2. Lessee has full use of all equipment and facilities on site
3. Exclusive rights to process all surface and underground ores, stockpiles, and tailings
4. Unrestricted access for all exploration, mining, and mineral processing activities
5. Advance minimum royalty of US $36,000 per year
6. Annual work commitments as follows- During 2003 $300,000, During 2004 $700,000, and during 2005 $1,000,000
7. Royalty on precious and platinum group metals based on the price of gold: 2% NSR for gold price less than $300/ounce, 3% NSR for gold price between $300 and $350 per ounce, 4% NSR for gold price between $350 and $400 per ounce, and 5% NSR for gold price greater than $450 per ounce
8. All other metals subject to 4.0% NSR
9. First right of refusal to acquire the property
10. Right to remove all improvements erected or placed thereon by the lessee

    Postle and others (2006) indicated that the expenditure commitments required by the lease as outlined above have been completed and expenditures have exceeded the $2,000,000 required. To the best of the author's knowledge, the lease agreement between Mystery Creek Resources and Mespelt & Almasy Mining Company, LLC, is in good standing. The property is subject to an additional 2% net smelter returns production royalty in favor of unrelated third party interests.

On February 12, 2009 PFN announced that it had exercised an option to acquire from St. Andrew Goldfields Ltd. all of the outstanding shares of Mystery Creek Resources, Inc., a wholly-owned Alaskan subsidiary of St

Andrew Goldfields Ltd. Under terms of the agreement, Pacific North West would acquire all of the assets of Mystery Creek and assume its lease obligations at Nixon Fork, for $500,000. This financial obligation was met by the assigned September 1, 2009 deadline in payments of US$400,000 and US$100,000.

On August 13, 2009, Fire River Gold Corp announced that it had exercised an option to purchase a 100% interest in the Nixon Fork project from Pacific North West Capital (PFN). The terms of the agreement included the following conditions, which were met on time:

- Fire River paid PFN $50,000 on signing of the letter agreement, following the receipt of all necessary approvals.
- Fire River paid PFN $450,000 over a six (6) month period and a total of $2.5 million in Fire River shares at a deemed price of $0.45 per share.
- In addition, Fire River issued PFN one million share purchase warrants at an exercise price of CDN$0.50 for a period of 24 months from the date of issue.
- Fire River refunded all expenses incurred by PFN from May 1st 2009 until the finalization of the transaction, the total of which was not to exceed CDN$1,250,000.

Snowden is not aware of any changes to the aforementioned royalties and agreements.

4.4 Environmental liabilities

Flanders, Giroux & Rawsthorne (2010, p.5) describes the environmental liabilities:

> Two environmental assessments (1991, 1995) were conducted at Nixon Fork prior to commencement of production in 1995, both resulting in a finding of no significant impact. In October, 2005 the BLM published their findings of an environmental assessment conducted by the agency at the Nixon Fork mine site (BLM, 2005). The 2005 environmental assessment also resulted in finding of no significant impact.

5 Accessibility, climate, local resources, infrastructure and physiography

5.1 Topography, elevation and vegetation

Flanders, Giroux & Rawsthorne (2010, p.7) describes the physical geography at Nixon Fork:

> The Nixon Fork project is situated in an area of moderate topographic relief with elevations ranging from 300 to 460 meters. Ridges are generally rounded and are forested with black spruce, larch, birch, and alder. Lower elevations are often poorly drained due in part to discontinuous permafrost conditions, often being covered by soft muskeg and stunted black spruce forest. Outcrops are rare in the mine area with a few resistant knobs along ridgelines. Colluvial and vegetative cover mask bedrock on hillsides and valley bottoms.

5.2 Access

Flanders, Giroux & Rawsthorne (2010, p.7) report

> The Nixon Fork project is accessible via air by charter aircraft from Anchorage, Fairbanks or McGrath, all of which are served by regular scheduled commercial air service. The Nixon Fork airstrip is approximately 1,280 meters long and can handle Hercules C-130 and D-6 fixed wing transport aircraft. Alternative access is by barge on the Kuskokwim River from Bethel, on the coast to the village of Medfra (740 km) and then 16 km north to the mine site via an unimproved dedicated State of Alaska road corridor.

5.3 Climate

Flanders, Giroux & Rawsthorne (2010, p.7) report

> Summer daytime temperatures on the project are typically range from 22oC in the summer to lows of -40oC in winter. Precipitation averages 40 cm per year, much of this as snow during the winter months.

5.4 Local population centres and infrastructure

Flanders, Giroux & Rawsthorne (2010, p.7) reported on the local communities and infrastructure

> The cities of Anchorage (population 280,000) and Fairbanks (50,000) are the major sources of labor, supplies, services and health facilities for the Nixon Fork project. McGrath, with a population of about 350 and other nearby small villages may provide some of the local labor for mine operations. Facilities on site include a camp with accommodation for about 85 persons, a 200 tonne per day gravity flotation plant, assay lab, mechanical shop, offices and support equipment (Bridge Capital, 2008). Electricity for mine operations is generated on-site by diesel-powered generators.

6 History

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

Mineral exploration and development has occurred at the Nixon Fork project in several discrete phases since it its discovery in 1918. The earliest phase was placer exploitation and was followed quickly by the discovery of lode gold. Total placer production, principally from Mystery and Hidden Creek, is estimated at about 15,000 ounces. Total lode production for Au since 1920 amounts to approximately 185,000 ounces (Table 6-1). Mines also produced by-product Ag and copper including 19,566 ounces of silver, and 2,100,000 pounds of copper. (St. Andrew, 2007b, Bundtzen, 1999). The silver and copper production figures are incomplete and should be considered minimum estimates of past production. The following is a chronological summary of the project derived from published and private records available to the authors.

Table 6-1 Total Lode Gold Production - 1920 to 2007

Year	Mined Tonnes	Mined Grade (g/t)	Mined Ounces	Recovered Ounces	Recovery %
1920-1961	28,000	51.4	42,000	42,000	N/A
1995	4,047	92.7	12,062	10,361	85.9%
1996	36,420	37.9	44,378	36,749	82.8%
1997	49,059	31.1	49,053	39,665	80.9%
1998	25,158	58.1	46,993	40,283	85.7%
1999	7,697	51.1	12,644	10,691	84.6%
2007 - Q1	8,198	21.8	5,212	3,374	64.7%
2007 - Q2	7,433	16.0	3,468	2,261	65.2%
TOTALS:	**166,012**	**45.0**	**215,810**	**185,384**	**78.5%**

6.1 Exploration and Mining – 1909 to 1983

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

Gold was discovered on the Nixon Fork in 1909-1910 and caused the first rush of prospectors to the region (Bundtzen and others, 1986). The exact location of these discoveries is uncertain. Placer gold was discovered by F. E. Matthews in June, 1917 on Hidden Creek on the south end of what is now the Nixon Fork property and subsequent placer discoveries soon followed on Ruby, Mystery and Submarine Creeks (Thomas, 1948). Mertie (1936) reported a series of gold fineness values from various creeks in the district. The most productive placer deposits were located on Hidden Creek from Riddle Creek at the upper end to a point on Hidden Creek about one mile below the Riddle Creek – Hidden Creek junction. Gold fineness values on Hidden Creek production from 1925 through 1932 ranged from 892 to 961% and averaged 928%. Silver values ranged from 30 to 68% and averaged 59%. The largest single gold nugget reported from placer mining in the Nixon Fork area was a 4.75 ounce nugget from Hidden Creek (Mertie, 1936). Gold was reportedly fresh, angular and often contained adhering fragments of quartz. Associated heavy mineral concentrates include abundant native bismuth along with scheelite and barite. Other fineness values reported include 961% gold and 33% silver on Birch Gulch and 807% gold and 107% silver on Ruby Creek (Mertie, 1933).

The coarse, fresh nature of the placer gold found in these creeks suggested a nearby lode source which prompted prospecting and staking of the uplands surrounding these creeks in 1918 (Jasper, 1961).What we now call the Nixon Fork lode deposit was discovered by Pearson and Strand in the spring of 1918 and shortly after was leased to Thomas P. Eakin who sank the Crystal shaft (Roehm, 1937, Jasper, 1961). Eakin continued working in 1919, during which he mined and shipped 400 tons of ore to the Tacoma Smelter with an average grade of $90 per ton gold (4.35 opt). In 1919 the Whalen lode prospect (also known as the Whalen – Griffin prospect) was discovered by E.M. Whalen.

In early 1920 the Whalen lode, Pearson and Strand and McGowan and Mespelt prospect were leased to Juneau-based Treadwell-Yukon Company, Ltd. which prospected, developed and mined in various parts of their holdings from 1920 through late 1923 (Martin 1922, Brown, 1926, Roehm, 1937). Following prospecting in 1920, the Treadwell company erected a 10 stamp gravity recovery mill on Ruby Creek below the most promising workings. It sank the Garnet No. 1 and No. 2 shafts and the Recreation shaft above the mill, but derived most of its production in 1922 and 1923 from the Whalen lode. Roehm (1937) reported that the Treadwell company recovered $207,000 from the Whalen lode property (approx. 10,014 oz) and an additional $28,000 worth of gold from the Pearson and Strand property (approx. 1,354 oz). The Treadwell company dropped its options in late 1923 on the Pearson and Strand and McGowan and Mespelt prospects but retained its option on the Whalen lode for an additional year before terminating this option as well (Jasper 1961). Production records from the Whalen prospect from 1922 indicated the bullion produced from that operation had a gold fineness that averaged 812% with a silver fineness that averaged 171% (Mertie, 1936). Production records from the Pearson and Strand prospect from 1922 indicated the bullion produced from that operation had a gold fineness that averaged 740% with a silver fineness that averaged 243% (Mertie, 1936).

The Treadwell-Yukon Company stamp mill on Ruby Creek was described by Mertie (1936). The mill consisted of 10 stamps and accessory equipment and was manufactured by the Alaska Juneau Gold Mining Co. in Juneau, Alaska (Thomas, 1948). The ore from operations was sent through a grizzly and into a jaw crusher, where it was reduced to a 1.5 inch minus pulp. The crushed ore was then sent to the stamp battery and the undersized fraction reported across a set of mercury amalgamation plates. Pulp which passed over the plates, then went to a classifier which sent the coarser material to a ball mill. The pulp leaving the ball mill reported to a second set of mercury amalgamation tables. The remaining waste stream was impounded in a small tailing facility on Ruby Creek where it remains to the present. The waste was reported to contain sulfide material with gold values up to $22 per ton (1.1 opt). The mill was supplied with power from two 70 horsepower boilers and a

125 horsepower steam engine. Mill capacity was 50 tons per 24 hour operating day. Gold was produced on-site using a mercury retort.

In 1924 E.M. Whalen and four others leased the Treadwell mill and processed previously broken ore from the Whalen mine, recovering approximately $80,000 (3,870 ounces) in the process (Jasper, 1961). In 1926 Charles and Adolph Mespelt purchased the Treadwell company mill and the Pearson and Strand prospect. Roehm (1937) reported that work conducted by the Mespelts on the Pearson and Strand prospect between 1926 and 1932 produced an estimated $400,000 (19,351 oz). Production records from the Pearson and Strand prospect from 1926 through 1932 indicated the bullion produced from that operation had a gold fineness that averaged 735% with a silver fineness that averaged 247% (Mertie, 1936).

E.M. Whalen sank a new shaft on the Whalen prospect in 1936 and reportedly produced 50 tons or ore of unknown tenor (Roehm (1937). The Mespelt brothers retained ownership of the Nixon Fork property and continued prospecting and intermittent production from 1926 through 1950 when the property was leased to H.G. Wilcox (Roehm, 1937, Jasper, 1961). By 1952 the Wilcox leased had been terminated and Strandberg and Sons, Inc. acquired a lease on the property and began prospecting work which continued through 1964. The property was returned to the Mespelt brothers in 1964 and, in conjunction with Ted J. Almasy, these parties acquired ownership of all of the lode and placer rights in the Nixon Form mine area (Herreid, 1966).

Wallis and others (2003) reported that for the period 1920 through 1961 average head grades from lode production at Nixon Fork were 1.5 opt gold, 3.0 opt silver and 2% copper. Total production is estimated at 42,000 ounces of gold, 11,282 ounces of silver and 41,440 pounds of copper (Table 6-1). Placer mining is estimated to have amounted to about 15,000 ounces of gold with the majority of that from Hidden and Ruby Creeks and their tributaries.

6.2 Exploration – 1984 to 2002

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

Unless otherwise noted, the following summary of historic exploration, metallurgical testing and mining operations has been derived from Wallis and others (2003), Wallis and Rennie (2005) and Postle and others, (2006). The author did not have access to all of the historical reports written during the period 1984 through 2002 and relied on the above-referenced 43-101 compliant reports for accounts of exploration, development and production during this period.

Since 1984 the Nixon Fork property has been explored by a number of companies including Battle Mountain Gold (Duval Corporation) from 1984 to 1988 and the Nixon Fork Joint Venture, (NFJV) with Central Alaska Gold Co. as operator from 1989 through 1993. Exploration included soil and geophysical surveys, trenching and both reverse circulation and core drilling as listed in Table 6-2. Table 6-2 includes a minor amount of drilling that may have been carried out on adjoining lands not currently held. Nevada Goldfields Inc. (NGI) a wholly owned subsidiary of Consolidated Nevada

Goldfields Corporation (CNGC) acquired the property in July 1993 and carried out additional definition drilling and commissioned Pincock, Allen & Holt Inc. (PAH) to prepare a feasibility study. In 1995 this study was updated, also by PAH, and found that the project was feasible, returning an IRR of 116% at a gold price of $385 per ounce.

Table 6-2 Exploration Drilling - 1985 to 2002

Company	Year	Type	# Holes	Meters
Battle Mt. Gold	1985-1988	Surface RC	85	7,342
Nixon Fork JV	1989	Surface HQ Core	18	1,463
Nixon Fork JV	1990	Surface HQ Core	70	8,874
Nixon Fork JV	1993	Surface HQ Core	23	3,638
Nevada Goldfields	1994	Surface HQ Core	71	5,985
Nevada Goldfields	1994	Underground BQ Core	43	1,764
Nevada Goldfields	1996	Surface HQ Core	69	6,465
Nevada Goldfields	1996	Underground BQ Core	117	7,110
Nevada Goldfields	1997	Surface HQ Core	30	3,012
Nevada Goldfields	1997	Underground BQ Core	163	13,411
Nevada Goldfields	1998	Underground BQ Core	30	4,030
Total			**719**	**63,093**

During 1994, NGI completed 914 meters of underground workings including declines into the Mystery and Crystal zones in addition to both surface and underground drilling. In a report dated December 9, 1994, Derry Michener Booth & Wahl (DMBW) estimated the proven and probable minable reserves to be 117,200 tons averaging 1.32 oz/ton Au containing 154,500 ounces of gold based on a cut-off grade of 0.29 oz/ton Au and reducing all gold assays above 12 oz/ton Au to 12 oz/ton Au. This is an historical estimate and does not conform to the requirements of NI 43-101. Construction of the surface facilities and underground development commenced in March 1995. NGI invested about $34 million in the property during the life of mine (NGI March, 1998). Production from the Crystal and Mystery orebodies commenced in October 1995.

In 1996 NGI carried out a helicopter-borne combined electromagnetic and magnetic survey, stream and soil surveys over the mine area and two nearby townships that resulted in the acquisition of additional land holdings considered prospective for skarn gold deposits. These claims are no longer part of the property owned by Fire River Gold Corp. Surface drilling, totalling 5,832 meters, was completed on the mine property in 1996, testing the area between the Mystery and Crystal areas and at the Whalen prospect.

In 1997 NGI completed 2,375 meters of additional surface drilling on the mine property around the Whelan Glory hole, High Grade and Southern Cross areas.

In 1998 exploration was confined to a small soil sampling program and trenching at the Warrior prospect.

The parent of NGI, Real Del Monte Mining Corporation, the successor to CNGC and its subsidiaries filed for Chapter 7 liquidation on June 25, 1999. The property, including the surface facilities and equipment, was formally abandoned by the Bankruptcy Trustee in March 2000 and returned to the original owners, Ted Almasy and Margaret Mespelt by formal document in November 2002.

6.3 Mineral Processing and Metallurgical Testing – 1984 to 2002

Flanders, Giroux & Rawsthorne (2010, pp.12) report:

Several recognized firms carried out metallurgical testing prior to the 1995 production decision. Denver Mineral Engineers (1995) summarized the bulk sample metallurgical test work on the Crystal oxide ores as follows:

- Gravity recovery of gold 19.6%
- With flotation, overall gold recovery of 81%
- Copper flotation of the oxide ore gave a recovery 15.4% copper with a concentrate grade of 15.5%
- Gravity recovery of gold 33.9% with flotation overall recovery of 91.3%
- Copper flotation gave a recovery of 97.9% copper with a concentrate grade of 28.3%
- Processing of the Crystal-Mystery ores were summarized as follows:

The mill operated from 1995 through June 1999. The Nixon Fork mill consists of gravity separation and flotation circuits employing conventional crushing, milling, gravity separation, flotation and concentrate- and tailings-dewatering circuits capable of handling 140 tonnes per day. Tailings are disposed of in a lined facility. Concentrates in the form of filter cake were loaded into polypropylene super sacks for shipment by air to Anchorage and then by ship to Dowa's smelter in Japan. Dore from the gravity circuit was taken to McGrath and mailed to Johnson Matthey in the United States.

Production records from the mine indicated that average mill recovery since start-up was 84.8%. Initially about half of the gold was recovered from the gravity circuit. As more sulfide ores were processed this amount decreased. Typically the gold recovery averaged 83.5% for the oxide ores while the sulfide ores averaged about 90% recovery. Flotation responses varied with the ore type, with higher copper recovery in the sulfide ore resulting in a higher quantity of lower grade concentrate. The average concentrate grade was 16.6% copper containing 300 to 600 gpt Au and an average of 277 gpt Ag per tonne. Concentrate penalty elements including arsenic and antimony (reported as combined arsenic plus antimony) averaged 0.58%, bismuth averaged 0.22% and selenium averaged 0.01%.

6.4 Mining Operations – 1995 to 1999

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

Five zones were originally considered for mining, the Crystal, Mystery, J5A, High Grade/Rec and the Southern Cross. Figure 6-1 shows the deposits and their relationship with the surface drilling. Ultimately, the Nixon Fork Mine was developed for trackless mining with two 4 meter by 3 meter, –15% declines on two separate zones, the Crystal (Crystal-Garnet) and the Mystery, about 500 meters apart. Initial production was from the Crystal oxide ores from October 1995 through to May 1996 when production began from the Mystery decline.

The Crystal ore bodies are accessed by a 1,600 meter decline and 3,305 meters of development. The decline bottom is at the 145 meter elevation with the portal at 400 m. The water table varies from 140 to 168 meters in elevation. The J5A and High Grade/Rec bodies were accessed from the Crystal decline.

Total production from the Crystal mine, which included the 3000 from the 390 meter level to the 145 meter level, portion of the C3300 between the 365 meter level and the 160 meter level, minor orebodies of the C3001, C3002, C3004 and J5A areas, amounted to 116,971 t at an average grade of 43.5 gpt Au.

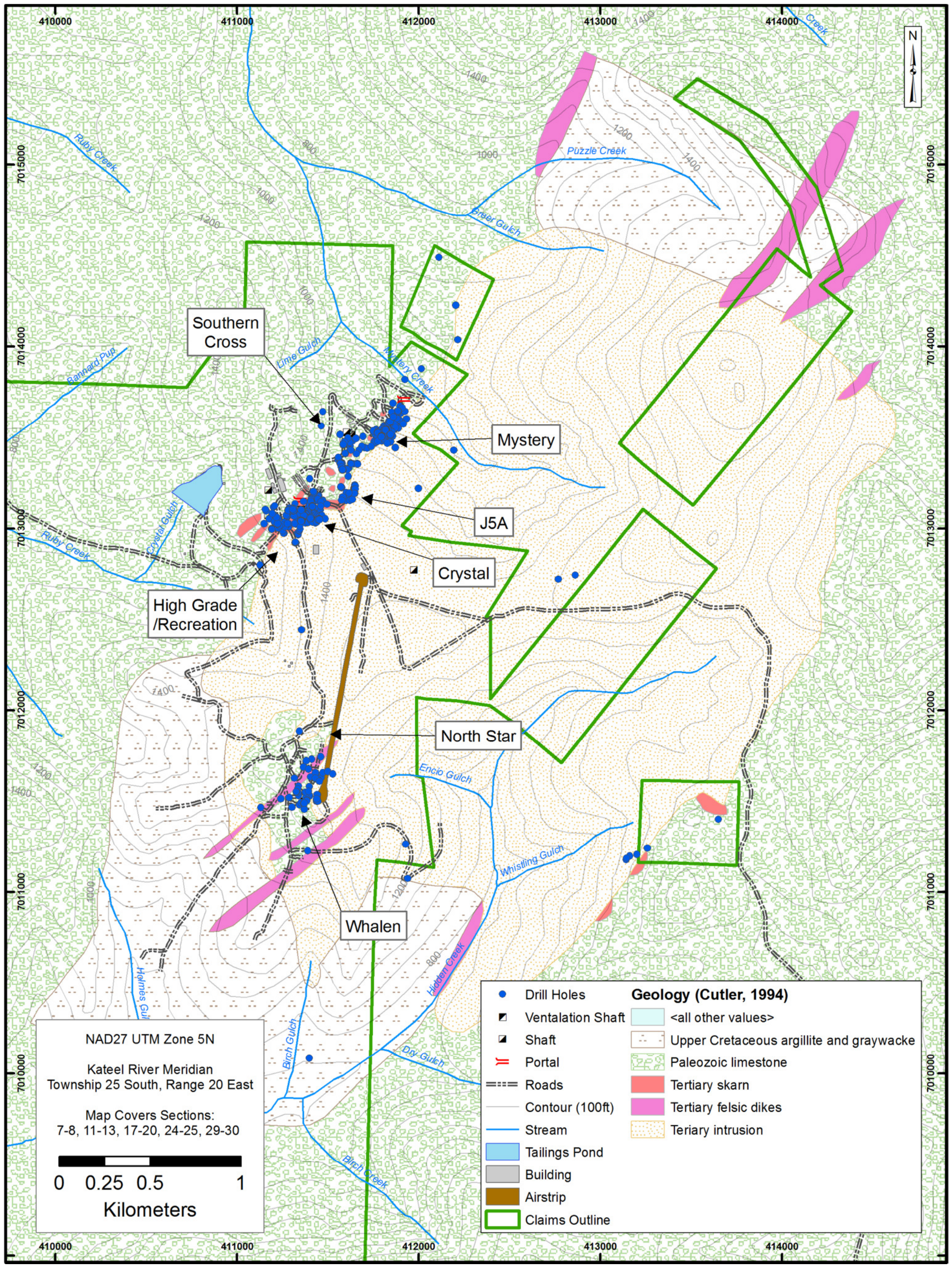
The Mystery decline, 642 meter in length, was developed in the sediments away from the quartz monzonite intrusive from the portal at 292 meter and the current face at 202 meters. Total development amounted to 783 meters. Total production from the Mystery was 5,410 t at 9.49 gpt Au. Several factors lead to shut down of mining in the Mystery, but gold grades were less than anticipated.

The stopes were developed with 15 meters from sill to sill. Three-meter crown pillars were left every 30 to 45 m. Stoping methods used initially were shrinkage and drift and fill where shallow dipping orebodies were encountered. However, fewer flatter dipping orebodies were encountered that originally thought. The typical stope was mined with jacklegs, overhand, shrinking vertically until day lighting through the sill above. After all the ore was broken it was mucked out and hauled to surface. Waste rock was dumped into mined out stopes when possible or hauled to surface. Waste rock mined was 122,381 t, of which approximately 35% was backfilled.

In total, the mine produced 137,749 ounces of gold and 2,100,000 pounds of copper from 1995 through 1998. Average production head grade for gold was 42 gpt and production costs averaged $266 per ounce recovered.

The profits from mining operations at Nixon Fork were utilized to fund the parent company Real del Monte's Mexican silver project development programs. The Mexican projects were not successful, forcing Real del Monte into bankruptcy and causing the closure of the Nixon Fork mine in 1999 (Freeman, 1999).

Figure 6-1 Nixon Fork Deposits



6.5 Exploration – 2003 to 2008

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

Unless otherwise noted, the following summary of historic exploration has been derived from Wallis and others (2003), Wallis and Rennie (2005) and Postle and others, (2006).

In early 2003, Geoinformatics Exploration Ltd. (Geoinformatics), was engaged to prepare a computerized three-dimensional geologic model of the Nixon Fork property and surrounding areas and analyze the model with the "Geoinformatics Process". All available geological data from the site was collected, placed in a digital format, and validated by Geoinformatics and Mystery's Exploration Manager. Geoinformatics prepared three-dimensional geologic models based upon the data collected with block modeling relying heavily on gold grades.

A limited program of geological mapping, trenching and sampling was carried out in the Whalen area where samples confirmed previous surface gold and copper mineralization extending for 75 meters northeast of the glory hole. Twenty trench samples returned from 0.86 to 44.8 gpt Au and 0.05% to 2.6% Cu over 0.3 to 2.8 m.

In late 2004 Aeroquest Ltd. conducted a detailed time domain EM and magnetic (AeroTEM) survey of the entire Mystery Creek pluton. Flight lines were spaced 50 meters apart, resulting in a higher resolution product than the surveys done in the past. A total of 735 line km were flown in two blocks, including approximately 500 line km of tie and traverse lines in the immediate vicinity of the mine and around 230 line km in a southern block (Bridge Capital, 2008). There were no significant EM anomalies located by this work.

In 2004 and early 2005 St. Andrew initiated a series of surface and underground drilling programs at Nixon Fork. This work included 121 NQ holes totalling 11,874.9 meters of underground drilling on the 3000 and 3300 bodies and an additional 32 NQ holes totalling 5,539 meters in the J5A area (Table 6-2). One surface drill hole (BQ core) totalling 63.9 meters was drilled in the Whalen area in 2004 to test for mineralization associated with NE trending dikes. From April 2005 to August 18, 2006, St. Andrew completed an additional 14,558 meters of core drilling in 121 holes in several areas of the project (Table 6-2).

Table 6-2 Exploration and Development Drilling - 2004 to 2008

Company	Year	Target Area	Type	# Holes	Meters
St. Andrew	2004-2005	3000/3300	Underground	121	11874.9
St. Andrew	2004-2005	J5A	Underground	32	5539
St. Andrew	2005-2005	Whalen	Surface	1	63.9
St. Andrew	2005-2006	J2100	Underground	17	2760.6
St. Andrew	2005-2006	3550	Underground	15	1998.5
St. Andrew	2005-2006	3300	Underground	10	1100.3
St. Andrew	2005-2006	Whalen	Surface	21	2850.1
St. Andrew	2005-2006	Mystery	Underground	35	3964.7
St. Andrew	2005-2006	Mystery	Surface	4	517.2
St. Andrew	2005-2006	Warrior	Surface	11	868.7
St. Andrew	2005-2006	3000 packer tests	Underground	8	497.9
St. Andrew	2007	3300	Underground	89	5454.9
St. Andrew	2007	Whalen	Surface	7	726.05
St. Andrew	2008	3300	Underground	33	2997.07
Total				**404**	**41,214**

During the period extending from late 2007 through 2008, St. Andrew completed 122 NQ diamond drill holes totalling 8,451.97 meters from underground in the 3300 zone and an additional 7 NQ diamond drill holes totalling 726.05 meters from the surface in the Whalen zone.

The exploration programs completed at the Nixon Fork project from 2003 to 2006 had a total cost of $2.3 million (Bridge Capital, 2008).

6.6 Mineral Processing and Metallurgical Testing – 2003 to 2008

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

Unless otherwise noted, the following summary of historic metallurgical testing has been derived from Wallis and others (2003), Wallis and Rennie (2005) and Postle and others, (2006).

A sampling program of the mine tailings pond was carried out in 2004 by H. Bogart, P. Eng. Samples were taken on 100 ft. line spacing over the dry part of the tailings, representing approximately one half of the pond. Some additional sampling was carried out on 50 ft. centers. The pond could not be completely sampled as part of it was under water. Prior to sampling, the depth of the material was determined by injecting water through a pipe and pushing the pipe through the tailings. A power auger was used to collect the samples to a maximum depth of 9.5 ft., the limit of the equipment available. A total of 13 holes were completed. A bulk metallurgical sample of material taken from the same sites assayed 0.279 oz/ton Au.

From December 2003 through October 2005, three phases of metallurgical testing were conducted on mined material and tailings from the Nixon Fork mine. In late 2003 MCRI took three 'bulk samples' of ore from the C-3000 ore chute considered to represent types of skarn ore to be milled from the Crystal Mine. Chlumsky, Armbrust & Meyer LLC (CAM) supervised testing of the C-3000 bulk sample by Phillips Enterprises, LLC of Golden, Colorado (Bridge Capital, 2008, Chumsky and others, 2005a, 2005b). The test work included a gravity circuit and a flotation circuit using fresh ore at Nixon Fork, and cyanide leaching of gold from flotation concentrates as well as from the previously processed tailings. CAM was also responsible for the development of a process flow diagram.

Phase 1 test work focused on cyanide leaching of whole ore to maximize the gold recovery into dore form. Due to the high levels of cyanide soluble copper, a proprietary process for copper and gold separation was tested. As the testing proceeded, it became apparent that while the process was technically viable, it was probably uneconomical due to the large amounts of cyanide consumed.

The Phase 2 test program focused on a more conventional processing scheme, calling for gravity recovery of coarse free gold followed by flotation of the copper minerals with additional gold recovered to a high grade copper concentrate. A separate copper circuit was tested to make a copper concentrate but more importantly to remove copper from the feed to the cyanide circuit to enhance the gold recovery in the cyanide circuit. After 25 flotation tests, it was clear that a 25% copper concentrate could be produced on a regular basis with an overall gold recovery of 75% and a copper recovery of approximately 80%. The recovery depends on the grade and the feed rate to the circuits.

Phase 3 testing focused on recovering gold from the 1995-1999 tailings pond. A gravity separation test was carried out resulting in a 6% recovery for gold and 4.5% recovery for gold. Because of the low recoveries, no further gravity work was considered. A flotation test carried out on the gravity tails did not recover an acceptable percentage of the precious metals (57% for Au) and flotation was not considered for further work.

6.7 Mining Operations – 1999 to 2008

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

Unless otherwise noted, the following summary of historic mining operations has been derived from Bridge Capital (2008).

Based on the results of exploration and metallurgical work conducted during the period 2003 through 2005, St. Andrew completed mining plans for the C3300 and lower portions of the C3000 ore bodies. These plans envision longhole mining where practical or shrink stope with backfill where needed. Based on the 2005 Nixon Fork economic analysis compiled under the supervision of Paul Jones, P.Eng, the Company's Executive Vice President, the Company proceeded with pans to commence mining operations at the Nixon Fork property.

In the first quarter of 2006, St Andrew began upgrading, rehabilitating and recommissioning the 200 tonne per day gold mill and related surface infrastructure. In addition, a cyanide leach circuit was planned to leach tailings produced by the operation in addition to extracting and retreating the tailings from the previous operation. Engineering work was completed in April 2006. Onsite construction commenced in the second quarter of 2006 and underground rehabilitation commenced in the third quarter of 2006. Initial underground ore extraction started late in the fourth quarter of 2006 from development ore obtained during the development of the stoping areas on the 3300 Shoot area. Concentrates were back-hauled by air on the fuel planes from the mine site to Fairbanks where they were placed in rail cars for transport by rail/barge to Xstrata Copper Canada's Horne Smelter in Rouyn-Noranda, Quebec. Prior to start-up, total costs for mine and mill rehabilitation, reclamation bonding and surface work was approximately C$10 million (St. Andrew, 2006).

In the first quarter of 2007, the Nixon Fork mine processed 8,198 tonnes of ore with an average grading of 21.8 grams per tonne Mill recovery rate for the quarter averaged approximately 64.40% producing 3,374 ounces of gold and 37,623 pounds of copper. During the quarter, the mine produced 172 tonnes of copper concentrate containing 2,789 ounces of gold. In the first quarter of 2007, mining operations at Nixon Fork were focused on advancing the development and establishment of stoping areas and resulted in lower volumes of stope ore being treated through the mill. The company anticipated the availability of additional stopes by the third quarter of 2007.

Initial pre-production efforts at Nixon Fork focused on the development of stoping areas in the 3300 Shoot. In May 2006 the Company announced the appointment of Procon Mining and Tunneling ("Procon") as the mining contractor for the Nixon Fork Gold Mine. During the underground development program problems were experienced in identifying the shapes of the ore bodies as outlined by three dimensional modeling. Although high grade (in excess of 25 grams per ton), the irregular shape of the ore shoots, resulted in excessive dilution in the ore delivered to the mill. Consequently gold production fell well short of production targets. An underground drilling program was commenced in June 2007 to better define the dimensions of the modeled ore shoots.

In the second quarter of 2007, the Nixon Fork Gold Mine processed 7,433 tonnes of ore with a head grade of 16.0 g/t Au . Mill recovery rate for the quarter averaged 65.2% producing 2,261 ounces of gold. During the quarter, the Company recognized gold sales of $2.7 million recovered from 258 tonnes of copper concentrate delivered to the smelter. The company reported that mine development scheduled for completion in the second quarter fell behind schedule due to equipment and mining personnel shortages and ore face availability issues encountered in the upper portion of the Crystal deposit. The company indicated that it planned to shut down the mill operations for about 6 weeks in August and September to allow for the planned installation of tailings filtration equipment and the integration of the dry stack tailings facility at the mine.

On October 10, 2007 St. Andrew announced that it had suspended operations at Nixon Fork, pending additional definition drilling and resource modeling (St. Andrew, 2007c). Mining at the time was under way on the upper portion of the 3300 ore body. Construction of a dry stack tailing facility and the installation of the mill cyanide circuit also were suspended. The company indicated that better definition of mineralization was required at both the 3300 and 3000 ore bodies before mining would resume. There have been no mining or milling activities at the Nixon Fork mine since that time. Total production during 2007 was 6,775 ounces of gold and 78,644 pounds of copper (Fire River, 2009).

6.8 Mineral Resources and Reserves – 2003 to 2008

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

Following its acquisition of the Nixon Fork project, St. Andrew Goldfield commissioned Roscoe Postle Associates Inc. to complete new mineral resources and reserves that were complaint with National Instrument 43-101 (Wallis and others, 2003). In 2005, Roscoe Postle provided and an updated resource estimate (Wallis and Rennie, 2005). On October 2, 2006, Scott Wilson Roscoe Postle Associates completed the most recent NI-43-101-compliant mineral resource and mineral reserve estimate for the Nixon Fork project (Postle and others, 2006)

6.9 Exploration and Development Work – 2009 to 2010

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

Following acquisition of the Nixon Fork project in early 2009, PFN commenced a CDN$1.25 million program with the objective of conducting a comprehensive re-evaluation of mine reserves and resources, metallurgy, tailing production scenarios, completion of updated NI 43-101 technical report, financial analysis update the mine plan and a recommended program for the exploration on the project. These studies will form the basis for a planned re-start of mining operations. These programs are on-going as this report was being written. Final conclusions from these work programs are not expected until the fourth quarter of 2010.

Work since late 2009 at the Nixon Fork project consists primarily of geologic logging of historic drill core, and selected re-sampling of important intervals (Fire River, 2010a-f). To date re-logging has been completed for all core drilled in 2007 and 2008. In addition, assay results have been received for all holes.

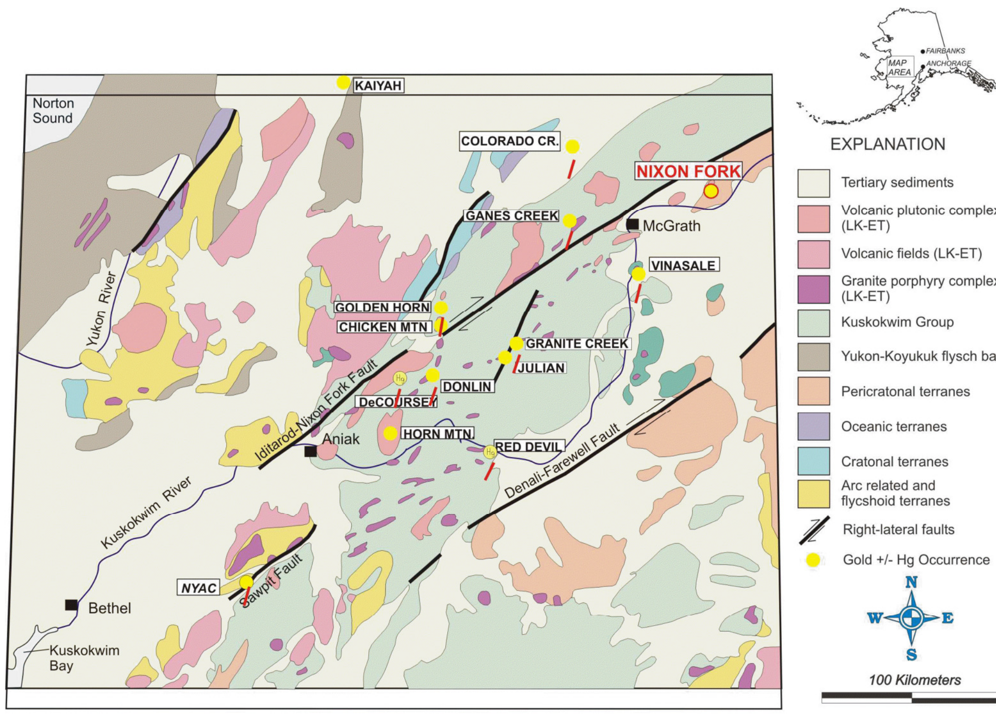
7 Geological setting

7.1 Regional geology

Flanders, Giroux & Rawsthorne (2010, pp. 21-23) report

> The Nixon Fork project is located on the northeastern edge of the Kuskokwim Minerals Belt (KMB) of southwestern Alaska (Figure 7-1, Bundtzen and Miller, 1996, 1997 modified by Avalon Development in 2009). The KMB roughly parallels the Kuskokwim Mountains which form a broad northeast-trending belt of accordant rounded ridges and broad sediment filled lowlands with locally rugged, glaciated igneous-cored massifs. The KMB covers an area approximately 550 km long by 350 km wide (192,500 km^2) that extends from Goodnews Bay on the extreme southwestern coast, to Von Frank Mountain, about 100 kilometers northeast of McGrath.
>
> Rocks in the KMB have been subdivided by age and tectonic history into two groups: Lower Cretaceous and older fault-bounded terranes, and middle Cretaceous and younger overlap and basin fill assemblages of sedimentary and volcanic rocks, which were subsequently intruded by mafic to felsic plutons (Bundtzen and Miller, 1997; Bundtzen and Gilbert, 1983; Decker et al., 1994; Miller and Bundtzen, 1994). Proterozoic to Lower Cretaceous rocks crop out in fault-bounded belts that generally parallel the northeasterly structural grain of the region. The Nixon Fork mine is situated between two regional northeast trending structures associated with the KMB, the Denali Farewell fault system to the south and the Iditarod-Nixon Fork fault to the north. Both of these structures have undergone Cretaceous-Tertiary offsets of less than 150 km (Decker and others, 1994). Numerous northeast and northwest trending subsidiary structures that are related to the Iditarod - Nixon Fork and Denali Farewell faults occur in the Nixon Fork project area and possibly influenced the emplacement of intrusive bodies in the area.
>
> Proterozoic through Lower Cretaceous basement rocks of the Nixon Fork area are considered part of the Nixon Fork terrane, variously interpreted as a discrete allochthonous terrane (Patton and other, 1994) or as various facies of the continental shelf and slope rocks of the Farewell terrane (Decker and others, 1994). Jones and others (1982) suggested that the Nixon Fork terrane was tectonically displaced several hundred kilometers from the northwestern part of the Canadian Cordillera before it was sutured to its current location.

Figure 7-1 General Geology of Southwestern Alaska (Bundtzen and Miller, 1997)



Amalgamation of the older allochthonous terranes of western Alaska was completed prior to middle Cretaceous time (Nokleberg and others, 1994; Decker et al., 1994; Patton et al., 1994). Subsequently, these older terranes were eroded and partly covered by terrigenous clastic rocks deposited into the Kuskokwim basins. These basin fill sequences are Middle to Late Cretaceous in age and display prograding turbidite facies interpreted to be of shallow marine and shoreline origin (Miller and Bundtzen, 1994; Patton et al., 1994). The regionally extensive Upper Cretaceous Kuskokwim Group was deposited primarily by turbidity currents into an elongate, probably strike-slip basin (Miller and Bundtzen, 1994). Local interbedded tuffs and volcaniclastic sandstone exist within the Kuskokwim Group however much of the Kuskokwim Group is derived from a mixture of sedimentary and metamorphic terranes (Decker et al., 1994). Small island of Kuskokwim Group sediments remain in the Nixon Fork area disconformably overlying the older basement rocks.

Late Cretaceous to Early Tertiary volcanic-plutonic complexes, plutons and subvolcanic dike and sill swarms intrude and overlie the older terranes and the Cretaceous Kuskokwim group (Bundtzen and Miller, 1996). These Late Cretaceous- Early Tertiary igneous rocks host a variety of mineral deposits that form the KMB and range in composition from gabbro to alkali granite with intrusives in the Nixon Fork mine area comprised primarily of quartz monzonite (Patton and other, 1980; Wilson and other, 1998).

The dominant deformation events affecting rocks of the KMB began in Late Cretaceous time, although earlier deformational events clearly affected the

Nixon Fork and adjacent allochthonous terranes prior to their amalgamation (Patton et al., 1994). The post-accretionary, overlap assemblages (Kuskokwim Group and Late Cretaceous to Early Tertiary volcanic and plutonic rocks) were deformed along continental-scale right lateral, strike slip faults with accompanying en-echelon folds and high-angle faults (Flanigan and others, 2000; Miller and Bundtzen, 1994). The oldest overlap assemblages (Middle Cretaceous) are the most highly deformed and were subjected to multiple fold episodes characterized by steep sub-isoclinal folds. Late Cretaceous and younger rocks are more broadly folded. The main fault zones affecting the KMB include the Poorman, Nixon Fork - Iditarod, Farewell-Denali and Susulatna faults, strike roughly 055° to 060°, with offsets that range in order from 16 to 160 kilometers. The dextral strike slip tectonic environment probably controlled the formation of the Kuskokwim basin and the emplacement of Late Cretaceous - Early Tertiary plutonic and volcanic rocks (Nokleberg and others, 1994; Miller and Bundtzen, 1994). The Nixon Fork terrane is bounded to the north by the Poorman fault and to the south by the Nixon Fork – Iditarod fault.

Unconsolidated fluvial, colluvial, and aeolian deposits that range in age from late Tertiary to Holocene cover at least 50% of the maturely eroded Kuskokwim Mountains.

7.2 Property geology

Flanders, Giroux & Rawsthorne (2010, pp. 24-26) report

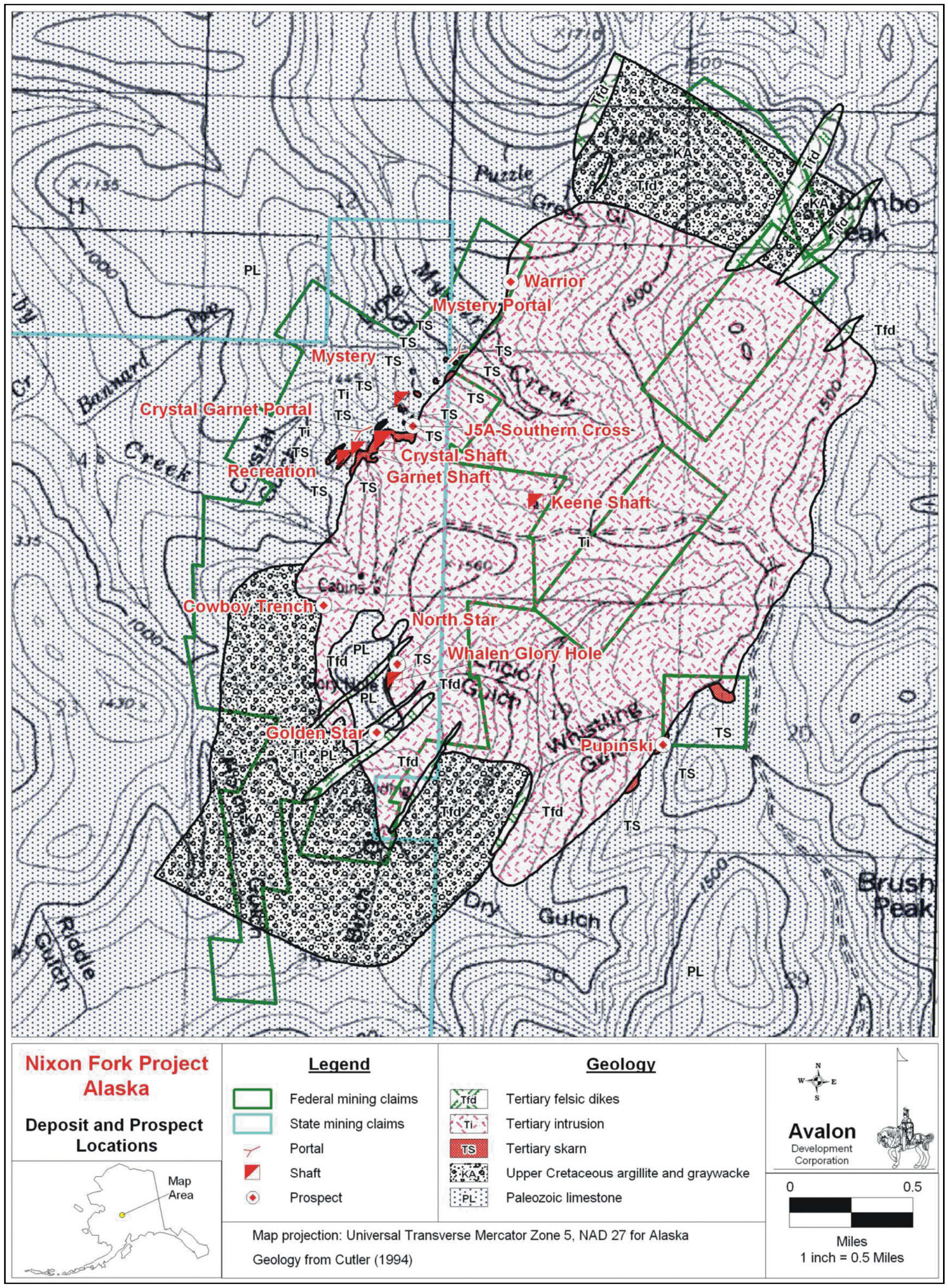
The majority of the basement rocks in the Nixon Fork project area are composed of Cambrian to Devonian-aged shallow water carbonate rocks which have been poorly described in the mine area. Cady and others (1955) assigned the limestone units in the Nixon Fork area to the early Paleozoic Holitna Formation. However, based on more detailed biostratigraphic evidence, subsequent investigators have assigned the shallow shelf carbonate units of the Nixon Fork mine area to the Ordovician Telsitna and Novi Mountain Formations (undivided), both shallow water limestone and dolomite units which crop out in the region (Wilson and others, 1998, Patton and others, 2009). These carbonate units are the primary host for both calc-silicate alteration and gold-copper mineralization at the Nixon Fork project.

Paleozoic formations of the Nixon Fork terrane were subsequently covered by terrigenous clastic formations of the Lower Cretaceous Kuskokwim Group (Patton and others, 2009, Wilson and others, 2009). Cutler (1994) reported that these rocks in the Nixon Fork area consisted of black, fine grained metagraywacke and slate which have been preserved from erosion along the keel of a northeast trending synform. This rock exhibits biotite hornfels alteration near contact zones with Cretaceous- to Tertiary-age stocks (Patton and others, 2009). These Cretaceous -age rocks host none of the gold-copper mineralization on the Nixon Fork project (Cutler, 1994).

In the Nixon Fork area, two 68-70 Ma (Cretaceous) quartz monzonite stocks have intruded the Paleozoic and Cretaceous sedimentary rock units. In the mine area the skarn mineralization is related to the polyphase Mystery Creek stock that outcrops over a 5 kilometer by 3 kilometer area. The stock contains both disseminated and vein style copper and gold mineralization and metasomatic skarn mineralization is formed in several areas along the margin of this stock (Cutler, 1994). Newberry and others (1997) and Cutler (1994) indicated that the plutonic rocks associated with the Nixon Fork mineralization are slightly to moderately alkalic, extremely reduced (I-type field) and contain little or no modal magnetite or ilmenite. The Eagle Creek stock, 11 km southwest of Nixon Fork contains similar mineralization within the stock and skarn mineralization in the adjacent sedimentary rocks (Thomas, 1948; Bundtzen, 1999). The Eagle Creek stock is outside of the Nixon Fork land block and will not be discussed further in this report.

Cutler (1994) identified three chemically distinct phases of the Mystery Creek stock. The oldest phase (Unit 1 of Cutler, 1994) occurs in the western and central portions of the pluton and consists of medium to coarse grained equigranular monzodiorite, quartz monzonite and monzonite. The volumetrically largest phase of the Mystery Creek pluton (Unit 2 of Cutler, 1994) consists of medium to coarse grained monzonite, quartz monzonite and quartz monzodiorite covering an estimated 75% of the surface area of the pluton. Unit 2 plutonic rocks appear to be more quartz rich and more evolved that those of Unit 1 (Cutler, 1994). The background gold content of all three of these plutonic rock units is generally less than 5 ppb and in some cases, less than 1 ppb (Cutler, 1994).

Figure 7-2 General Geology of Prospect Locations (Cutler, 1994)



The youngest plutonic rock in the Mystery Creek pluton forms (Unit 3 of Cutler, 1994) northeast-elongate altered dikes ranging in composition for granite to quartz monzonite. The dikes typically have an aplitic texture and post-date skarn alteration and gold-copper mineralization, although these dikes may have filled pre-existing structural corridors that were integral to prograde and/or retrograde skarn alteration and mineralization. These aplite dikes appear to follow northeast trending structures which parallel the regional structural fabric in this part of Alaska. Aplite or felsite dikes are most common in the northeastern and southwestern parts of the Mystery Creek pluton.

The structural geology of the Nixon Form area is extremely complex at the mine-scale but on the project scale consists of a broad northeast-trending synform, the core of which is occupied by the Mystery Creek pluton (Patton and others, 1980, Patton and others, 2009). The Nixon Fork synform (local name given by the author) is subparallel to a series of other fold axes mapped to the northeast of the Nixon Fork mine site and which are all bounded by two strands of the Nixon Fork - Iditarod strike slip fault. These major faults have been mapped to the northwest and southeast of the Nixon Fork property.

8 Deposit types

8.1 Nixon Fork geological model

Flanders, Giroux & Rawsthorne (2010, p. 27) report

> Bundtzen and Miller (1996, 1997) and Miller and others (2002) present a wide variety of gold deposit models that have been defined in time-equivalent rocks and similar structural settings in the KMB. Virtually all of the significant lode gold occurrences in the KMB are associated with late Cretaceous to early Tertiary volcanic and/or plutonic rocks which post-date the Kuskokwim Group post-accretionary basic fill sediments. At least five broad deposit types have been identified in the KMB with Nixon Fork considered to be part of the plutonic-hosted mesothermal gold deposit class.
>
> Plutonic-hosted mesothermal gold-copper-polymetallic deposits are hosted in plutonic rocks ranging from alkali gabbro to granite but gold-enriched skarns are more commonly associated with quartz monzodiorite, quartz monzonite, and monzogranite (Newberry and others, 1997). Exoskarn formation occurs in a wide variety of carbonate hosts with resultant calc-silicate mineralogy controlled to some degree by the Ca:Mg ratio of the carbonate host rocks. Associated intrusive rocks in gold-enriched skarns are generally reduced (I-type) plutonic bodies with alkalic chemical affinities. Evidence from Cutler (1994) indicates that the Nixon Fork intrusive body is an I-type granitic body based on a Na2O versus K2O alkalinity plot. Intrusive samples from Nixon Fork also plot primarily in the gold favorable field of alkalinity versus redox potential plots (Cutler, 1994). Skarns such as Nixon Fork commonly exhibit elevated Cu, Bi, As, Te and Sb and often contain anomalous Zn, Ag, Sn W, Co and Ni. Gold-enriched skarns also tend to be low in Mo content. Oxide skarn deposits in this class often contain copper oxides after bornite and chalcopyrite with coarse visible gold. Alteration assemblages are dominated by grossularitic garnet and hedenbergitic pyroxene, typical of calcic skarns. However with dolomitic (Mg rich) limestone and dolomite, skarns containing diopside, salite and olivine as well as abundant magnetite are common. Manganese is distinctly absent in garnet and pyroxene in gold-enriched skarns.

9 Mineralisation

Flanders, Giroux & Rawsthorne (2010, pp. 28-36) report

> Plutonic and sedimentary country rocks at Nixon Fork have been altered, and in some cases mineralized, following emplacement of the Mystery Creek intrusion(s). Since post-intrusion alteration is genetically linked to skarn-hosted gold-copper mineralization at Nixon Fork, the various styles of alteration affecting the Nixon Fork area are discussed in this section. The alteration/mineralization of rocks at Nixon Fork can be divided into five distinct, but often overlapping types or assemblages: 1) contact metamorphism, 2) prograde skarn, 3) retrograde skarn, 4) local overprinting of skarn by quartz-sericite-carbonate alteration and 5) supergene oxidation and metal enrichment or depletion (Cutler, 1994).
>
> A variety of skarn types are present at Nixon Fork. These include calcic-skarns, after limestone host rocks, and magnesium skarn after dolomites. The percentage of dolomitic host rocks are likely equal to that of limestone host rocks. In addition, skarn types can be further divided into units useful in modeling that are based on their mineralogical compositions: 1) garnet > pyroxene skarn, 2) pyroxene > garnet skarn, 3) wollastonite skarn, 4) magnesian skarn (serpentine-phlogopite-talc-tremolite) and 5) retrograde sulfide-rich skarn.

9.1 Contact Metamorphic Alteration

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

> Paleozoic and Mesozoic country rocks which were intruded by the Nixon Fork intrusive complex exhibit variable contact metamorphic alteration depending primarily on original host rock composition. Massive limestones are commonly altered to dense gray marbles while impure limestones and dolomites exhibit marbles with fine grained bands of calc-silicate minerals, primarily clinopyroxene, idocrase and garnet (Cutler, 1994). Pyroxene compositions of contact metamorphic origin are magnesium rich and are chemically distinctive from prograde skarn pyroxenes. Where observed together in drill core or in surface or underground exposures, prograde metasomatic skarn alteration post-dates contact metamorphic alteration and contact metamorphic affects extend laterally beyond the limits of prograde metasomatic skarn alteration. Einaudi (1982) suggested that the brittle nature of contact metamorphic rocks makes them more susceptible to later prograde metasomatic skarn alteration and that the conversion of contact metamorphic hornfels to skarn causes volumetric reductions, enhancing the porosity of these rocks. Both the chemical and physical changes in contact metamorphic rocks make them more favorable hosts for later gold-copper mineralization.

9.2 Prograde Skarn

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

> Prograde skarns at Nixon Fork occur primarily as exoskarn hosted in former impure limestones and dolomites. Endoskarn, formed within the causative plutonic body, does occur locally at Nixon Fork but is less significant volumetrically relative to exoskarn. Spatially, the most important skarn alteration and mineralization occurs on the western and southwestern edges of the Nixon Fork plutonic suite in areas where previous mining has occurred: the Mystery area, the Crystal area and the Whale area. Several smaller occurrences also have been identified around the Nixon Fork plutonic suite although none have been mined or prospected extensively.
>
> Prograde skarn minerals at Nixon Fork consist primarily of variable amounts of garnet, ranging from Fe-rich andradite to Ca-rich grossularite along with pyroxene, ranging from Fe-rich hedenbergite to Mg-rich diopside (Cutler, 1994). Manganese-rich garnet and manganese rich pyroxene are rare or absent in prograde skarns at Nixon Fork. Textural evidence suggests that pyroxenes formed first during prograde metasomatism while garnets formed later by replacement of pyroxene (Cutler, 1994).
>
> Pyroxene > garnet skarn, with pyroxene forming 20-60% of the rock, is volumetrically the most abundant prograde skarn assemblage at Nixon Fork (Cutler, 1994). Pyroxene > garnet skarn is less abundant closer to the intrusive contact and shows partial replacement of pyroxene by garnet. Wollastonite occurs in minor amounts in pyroxene > garnet skarns. Chalcopyrite, pyrite, marcasite and arsenopyrite occur as disseminations and late veins within pyroxene > garnet skarn and can make up 2-40% of the total rock mass. Retrograde alteration (see below) is best developed in pyroxene > garnet skarn and most of the gold is hosted in pyroxene > garnet skarn.
>
> Garnet > pyroxene skarn with garnet forming 50-70% of the rock, is volumetrically the most abundant prograde skarn assemblage at Nixon Fork (Cutler, 1994). Garnet > pyroxene skarn is more abundant closer to the intrusive contact and shows partial to complete replacement of pyroxene by garnet. Wollastonite and idocrase occur in minor amounts in garnet > pyroxene skarns. Chalcopyrite, pyrite, and pyrrhotite occur as disseminations and late veins within pyroxene > garnet skarn and can make up less than 1% of the total rock mass.
>
> Wollastonite skarns at Nixon Fork post-date and cross-cut previously formed pyroxene > garnet and garnet > pyroxene skarn (Cutler, 1994). Cross-cutting features are sharp and suggest wollastonite skarn formed as a late-stage vein or vug-filling event. Wollastonite skarn is most common near the intrusive contacts and no retrograde alteration is evident at the interfaces between wollastonite skarn and other prograde skarn minerals, suggesting Wollastonite alteration may post-date retrograde alteration. Accessory minerals in wollastonite skarn include garnet, pyroxene and idocrase. Sulfide minerals comprise less than 1% of wollastonite skarn assemblages and include pyrite

and chalcopyrite. Gold values in wollastonite skarn are generally less than 5 ppb.

Magnesian skarns have been identified in drill holes but have not been seen at the surface at Nixon Fork. Magnesian skarns consists of fine grained serpentine, phlogopite, diopside and idocrase with minor olivine, talc and tremolite. Magnetite and hematite also occur with magnesian skarns. Magnesian skarns are commonly anomalous in gold at Nixon Fork and locally are ore grade.

Based primarily on the work of Cutler (1994), prograde skarn alteration at Nixon Fork appears to have occurred at shallow depths of emplacement. This conclusion is supported by the presence of post-skarn porphyry dikes, widespread retrograde alteration, numerous open-space fillings and veins, abundant evidence of brittle fracture and a relatively small contact metamorphic aureole surrounding the Nixon Fork plutonic complex. Based on petrological and mineralogical data, Cutler estimates that the Nixon Fork prograde skarn formed at shallow depths (0.5 kb) from a slightly reduced magma. Prograde assemblages formed somewhere in the 400 to 500oC degree temperature range followed by retrograde alteration below 410^{o}C and consisting of amphibole, epidote, quartz, calcite and sulfides. Wollastonite alteration appears to have occurred during or after retrograde alteration.

9.3 Retrograde Skarn

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

Previous workers have suggested that all of the commercially important metals in the Nixon Fork deposit are hosted in retrograde altered prograde skarn and that gold grades and associated sulfide content are directly proportional to the intensity of the retrograde alteration (Cutler, 1994). Qualitative work conducted by PFN and Fire River in 2009 suggest that much of the previously mined Mystery ore was high grade pyroxene chalcopyrite skarn with very minimal retrograde alteration. It is possible that some mineralization previously classified as being hosted in retrograde skarn alteration, was in fact hosted in rocks affected by recently identified ferroan dolomite alteration and/or subjected to supergene oxidation, both of which appear similar to true retrograde altered skarn. Additional petrographic and geochemical research will be required to determine the extent and significance of ferroan dolomite alteration and supergene oxidation.

Cutler (1994) identified 4 types of metallic mineralization associated with retrograde alteration: 1) disseminated and vein chalcopyrite + pyrrhotite + pyrite with lesser marcasite and bornite in pyroxene-dominant skarn, 2) massive arsenopyrite cut by veins containing chalcopyrite + pyrrhotite + pyrite with lesser marcasite and bornite, 3) oxidized and possibly supergene-enriched zones containing free gold with chalcopyrite, pyrite, malachite and azurite in a gangue of chlorite, clay, limonite, calcite and quartz in formed from garnet pyroxene skarn, and 4) recrystallized limestone and dolomite that contain free gold in a mixture of limonite, hematite, chlorite, calcite, quartz and serpentine.

Pyroxene > garnet prograde skarn is the dominant host for economically significant gold-bearing retrograde skarn. Primary retrograde alteration products include amphibole, quartz and calcite. Primary metallic minerals include native gold, chalcopyrite, pyrrhotite, pyrite with lesser amounts of bornite, marcasite, bismuthinite and several gold-silver telluride minerals. Chalcopyrite is volumetrically the most abundant sulfide at Nixon Fork and occurs as both fine grained disseminated grains in prograde skarn and as later, coarse grained sulfide veins in retrograde skarn. Higher gold values also accompany copper in retrograde skarn. Approximately 30-40% of the chalcopyrite observed at Nixon Fork displays exsolution of bornite from bornite-chalcopyrite solid solutions. Pyrite is the second-most abundant sulfide at Nixon Fork and is present in disseminated form in all skarn types as well as in recrystallized marbles and in intrusive rocks (Cutler, 1994). Pyrrhotite is present in retrograde altered pyroxene > garnet and garnet > pyroxene skarns and shows replacement by marcasite in retrograde skarn. In retrograde zones, pyrrhotite is spatially associated with chalcopyrite and pyrite. Pyrrhotite also has been seen as disseminations in recrystallized marbles and in intrusive rocks but has not been identified in prograde skarn that has not been retrograde altered.

Arsenopyrite occurs primarily as monomineralic masses to 5-6 m^3 which cut pyroxene > garnet prograde skarn near the marble-out front (Cutler, 1994). These arsenopyrite masses do not contain gold and are cut by gold-bearing chalcopyrite-pyrrhotite-pyrite veinlets formed during retrograde alteration. Microprobe analyses of arsenopyrite from Nixon Fork has indicated that it formed at temperatures of 360-390°C (Cutler, 1994), a temperature range which fits well with retrograde skarn development temperatures derived from other skarn deposits.

Limited multi-element data analysis conducted by Cutler (1994) indicates that gold has an extremely strong, positive linear correlation with bismuth (bivariate correlation coefficient of 0.94) while gold and cobalt share a strong positive linear correlation (bivariate correlation coefficient of 0.71). Silver, copper and arsenic show weaker but still positive correlations with gold (bivariate correlation coefficients of 0.44, 0.40 and 0.37, respectively). Evidence from other intrusive-related gold and gold-copper systems in Alaska also shows this strong gold - bismuth correlation (McCoy and others, 1997, Gage, 2002, Ebert and others, 2003).

9.4 Metallic and Silicate Zoning

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

The Mystery and Crystal areas of the Nixon Fork project exhibit two distinctly different styles of skarn development and later retrograde alteration associated with gold-copper mineralization (Cutler, 1994). Skarn zonation in the Mystery Creek area (Mystery Decline) consists of regular zonation outward from the intrusive stock on the east through a 2-8 meter thick garnet > pyroxene zone through a 3-22 meter thick pyroxene > garnet zone, with or without the presence of a 1-6 meter thick wollastonite zone which overprints the garnet > pyroxene zone. Outward of the pyroxene > garnet zone, alteration grades is a highly variable thickness of calc-silicate-bearing hornfels composed of argillite, dirty limestone and dolomite. Magnesian skarn is sporadically developed in the dolomite bed which occurs about 50 meters west of the intrusive contact.

Gold and copper mineralization in the Mystery Creek area occurs in retrograde skarn with forms primarily in the pyroxene > garnet skarn (85% of Cu-Au mineralization) with the remaining 15% of the copper and gold mineralization hosted by retrograde altered garnet > pyroxene skarn. Retrograde gold-copper skarn bodies average 12 to 20 m3 in volume. Massive arsenopyrite bodies replace and cut across pyroxene > garnet prograde skarn near the marble front. These arsenopyrite bodies are in turn cut by retrograde alteration veins.

In contrast, skarn zonation in the Crystal-Garnet area does not exhibit the regular zonation that has been mapped at the Mystery Creek area and retrograde alteration (and associated gold-copper mineralization) appears to be controlled more by late felsic dikes and fault structures (Cutler, 1994). The intrusive-skarn contact in the Crystal area is generally faulted and retrograde alteration is best developed at or close to this faulted contact, although both prograde and retrograde skarn develop along structures up to 60 meters west of the intrusive contact. Retrograde skarn bodies are controlled by northeast trending felsic dikes and structures and northwest-trending structures. Unlike copper - gold mineralization in the Mystery Creek area, some of the copper - gold mineralization in the Crystal area is highly oxidized and variably supergene enriched, both of which may be a function of the higher fracture density in the Crystal area. In addition, previously identified supergene oxidation effects may be related in part to recently identified ferroan dolomite alteration (G. Myers, oral comm.., 2009). Oxidation at Nixon Fork is variable in general but extends to at least 140 meters below surface at the bottom of the Garnet shaft as evidenced by the presence of solution collapse breccias and copper oxides (Jasper, 1961).

Limited modern exploration has been conducted on the Whalen prospect however evidence from drilling and past underground mining suggest that prograde and retrograde skarn bodies are formed in primarily dolomitic limestone. Skarn bodies occur along significant northeast-trending fault zones as well as prominent northeast-trending dikes. Late quartz-sericite carbonate alteration post-dates retrograde alteration and may have upgraded gold values in the skarn bodies. Historic mining records indicate that above the 100 foot level of the Whalen workings, gold and copper were hosted in highly oxidized rocks, possibly upgraded by supergene enrichment processes (Herreid, 1966, Brown, 1926). Sulfide mineralization becomes dominant below the 100 foot level.

Limited information is available on the origin of the gold and other metals in the Nixon Fork skarn. Cutler (1994) examined metal depletion ratios relative to the Nixon Fork intrusive complex and determined that the gold in the Nixon Fork deposit did not originate in the surrounding carbonate country rocks, leaving the Nixon Fork intrusive complex or an undiscovered (buried) source as the only other likely candidates for the source of gold. Sulfur isotopes from chalcopyrite, bornite and arsenopyrite from Nixon Fork range from 3.6 to 4.2 and suggest magmatic sulfur derivation (Cutler, 1994).

9.5 Controls of Mineralization

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

Many different types of control have been hypothesized for the gold copper ore bodies at Nixon Fork. Wallis and others (2003) suggested that skarn development occurs in close proximity to the margins of the quartz monzonite stock. The more significant zones of prograde skarn appear to be associated with favorable lithologies such as impure dolomite or thinly bedded micritic limestone. This type of control is typical in many skarn deposits. Postle and others (2006) suggest the presence of a single permissive lithologic unit that may be a host gold mineralization in some of the skarn zones. This hypothesis has not been confirmed by the recent re-logging campaign of Fire River Gold. This favorable bed is a fine grained calcareous siltstone that appears to be a simple facies equivalent within more common recrystallized carbonate host rocks. This favorable horizon averages 3 meters in thickness, strikes northeast, and dips 70^{o} to the southeast.

Pre-mineral northeast and northwest trending fault and fracture zones clearly acted as conduits for mineralizing fluids. The zones are evident in the Mystery Creek intrusion and can modeled as linear alteration and metal bearing zones that typically trend north-south to northwest. These zones include tabular bodies of endoskarn as well. Postle and others (2006) reported that drilling in 2004 and 2005 in the J5A zone that intercepted quartz-arsenopyrite-pyrite veins with sericite-carbonate selvages within the Nixon Fork plutonic complex. These veins are also expressions of pathways for later hydrothermal fluids that resulted in gold and copper bearing ore bodies.

At the local scale folds are prominently displayed in the underground mapping as well as in drill core. This folding may well influenced skarn development as well as metal concentrations (Wallis and others, 2003, Flanders, 1994). The work of the author found that dilatant cm-scale fold noses in the Crystal decline with an average axis attitude of 167°/37° were often the focus of gold-bearing sulfide mineralization and a useful guide for tracing gold mineralization that “disappeared” during failed attempts to follow it with drifting. This trend is very similar to the trends of the two principal ore zones in the Crystal, 3000 and 3300. Both are pipe-like in shape, the 3000 trends 196 and plunges -61, while the 3300 trends 180 and plunges -57.

On a district, the Mystery Creek intrusive complex and the adjacent Paleozoic and Mesozoic country rocks are located in the keel of a northeast trending synform that extends for at least 24 kilometers from Hidden Creek to the south to beyond Boulder Creek on the north (Patton and others, 1980, Patton and others, 2009). Host rocks for skarn on the northwest side of the Nixon Fork plutonic complex dip steeply to the southeast and into the intrusion. This configuration allows bedding and other bedding related structures to act more easily as pathways.

Other factors include the brittle nature of calc-silicate hornfels developed early in the alteration/mineralization sequence. A well-fractured hornfels would localize later fluids associated with prograde or regrade skarn alteration. An finally the extremely irregular contact of the Mystery Creek pluton, results in embayments and apophyses that likely help channel fluids preferentially. May gold-copper zones are developed within embayments and sometimes with overhanging or under hanging intrusions.

Recent petrographic and geochemical research of highly altered rocks in drill core, indicate a spatial association of gold-copper mineralization with highly altered felsite dikes. More work is in progress to model and define these potentially important intrusive phases.

9.6 Mining Areas

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

Unless otherwise noted, the following descriptions of the geometry and grade of the underground skarn bodies mined between 1994 and 2007 are derived from Wallis and others (2003) and Wallis and Rennie (2005).

The majority of the recent underground gold-copper production from the Nixon Fork mine area occurred in the Crystal decline. The Crystal area includes five main mineralized shoots that were accessed by a decline and mined during the period 1995 to 1998 and 2006-2007. The majority of the production came from the C3000 (3000) ore shoot in 1994-98. The mineralized skarn bodies in this part of the mine range up to 6 meters in width with an average of 3 meters. The strike length of the individual shoots varies from 10 to 30 meters but these pipe-like bodies extend down dip for several hundred meters. Average grade of the mineralized bodies varies from 9 gpt up to 45 gpt Au. In the Crystal zone, silver and copper values average 27 gpt and 1.2 %, respectively.

Mineralization in the 3000 shoot was hosted in a folded calcareous siltstone unit in contact with a dike or apophysis of the Nixon Fork pluton. North-south faults have localized the monzodiorite dikes and the quartz-chalcopyrite veins. Both oxidized and un-oxidized pyroxene-garnet skarn is mineralized. Primary sulfide mineralization includes chalcopyrite, pyrrhotite, arsenopyrite, minor sphalerite and molybdenite. Native gold, bismuth, silver, and copper oxides are also present. The C3000 shoot produced 101,000 tonnes at an average reconciled grade of 43.3 gpt gold. The shoot extends from surface (approx. 400 meters above sea level) to at least 70 meters above sea level where it remains open to depth.

Limited mining occurred in the C3300 shoot in 2006-2007. The C3300 shoot rakes south along the faulted monzonite contact. A series of quartz-chrysocolla-chalcopyrite gold veins occur within the quartz monzonite with the adjacent limestone dipping into the contact. The quartz monzonite is sericitized and argillized. The mineralization is oxidized from the surface to a depth of 160 m with the calcic retrograde skarn containing quartz, chalcopyrite, copper and iron oxides and native gold. Below 160 m the mineralization consists of pyroxene, garnet, pyrrhotite, chalcopyrite, pyrite and quartz. Past production by Nevada Goldfields in the period 1994-1998 was 5,440 tonnes at an average grade of 23.6 grams of gold per tonne. The shoot is open to depth. Limited mining was conducted in the C3300 shoot by St. Andrew Goldfields in 2006 and 2007 (St. Andrew, 2007a, 2007b). In the first quarter of 2007, the Nixon Fork mine processed 8,198 tonnes of ore with an average grading of 21.8 gpt Au (St. Andrew, 2007a). Mill recovery rate for the quarter averaged approximately 76.0% producing 3,374 ounces of gold and 37,623 pounds of copper. In the second quarter of 2007, the Nixon Fork Gold Mine processed 7,433 tonnes of ore with a head grade of 16.0 gpt Au (St. Andrew, 2007b). Mill recovery rate for the quarter averaged 70.5% producing 2,661 ounces of gold. Mining was halted in late 2007 after production goals fell behind schedule and underground development activities encountered significantly higher volumes of open voids related to solution collapse breccias that were not detected by previous definition drilling (S. Teller, oral comm., 2009).

The C3001 and C3002 bodies are smaller shoots associated with the noses of monzodiorite dikes. It is believed that retrograde alteration and mineralization occurred where folding created open spaces. The C3001 body produced 6,735 tonnes at 39.6 gpt Au and the C3002 produced 2,150 tonnes at 23.4 gpt Au. East of the C3000 shoot is the C3004 shoot. This is an iron-gold-copper calcic skarn with a high iron content and low copper. The mineral assemblage includes garnet, pyroxene, chlorite, massive magnetite, pyrrhotite, chalcopyrite, native copper and gold. It is reported to have produced 1,531 tonnes at 10.8 gpt Au.

The Mystery mine was accessed by a separate decline and consists of six mineralized zones that have produced a total of 5,159 tonnes at an average grade of 12.9 gpt Au to a depth of 90 meters. Mining was halted in the Mystery due to low gold prices and lower than expected grades. The current resources in the Mystery decline are described in a subsequent section. Mineralized zones in the Mystery area have not been explored significantly at depth. The zones are associated with dikes as well as the contact zone to the main stock. In addition, 2 northwest-trending zones within the intrusion are thought to be feeder zones. Contact skarns contain mostly garnet-pyroxene with minor retrograde alteration. Metallic minerals include chalcopyrite, pyrrhotite, magnetite, arsenopyrite, bismuthinite and gold.

North of the Crystal decline three mineralized zones are present along the contact and include the 3100, J5 and the Southern Cross zones. These zones have not been adequately tested at depth and in some cases along strike. North-trending faulting appears to control at least one of the zones but in all cases theses higher grades are along the contact with quartz monzonite and localized in altered carbonate units.

The upper part of the 3550 body was mined historically as the High Grade – Recreation zone. A drill hole in 1997 intercepted 4 gpt gold over 2.7 meters which is thought to be the down-dip extension of the mineralization. And 3550 zone is 1ocated about 100 meters west southwest of the 3300. This area received limited drill testing in 2005.

Between the C3000 and J5A shoots is the C3004 shoot. This is an iron-gold-copper skarn with a high iron content and low copper. The mineral assemblage includes garnet, pyroxene, chlorite, massive magnetite, pyrrhotite, chalcopyrite, native copper and gold. It is reported to have produced 1,531 tonnes at 10.8 gpt gold.

St. Andrew conducted additional underground drilling (122 holes, 8,451.97 m) in the 3300 zone in 2007 and 2008. Significant geochemical results from this work are presented in Appendix 2. Although detailed three-dimensional comparison of this drilling with previous drilling in the 3300 zone is being conducted by Fire River as of the date of this report, results from the 2007-2008 drilling are not obviously different from previous gold, silver and copper grades, interval thicknesses and metal ratios seen in past drilling from the zone. As is typical for skarns in general and for the Nixon Fork skarn in particular, gold grades from the 2007-2008 drilling in the 3300 zone are highly variable, ranging from a low of 0.12 gpt Au to a high of 2,000 gpt (+58 opt). Abrupt changes in gold, silver and copper grades, sometimes in excess of three orders of magnitude, regularly occur over sub-meter interval thicknesses.

9.7 Other Exploration Targets

Flanders, Giroux & Rawsthorne (2010, pp.8-20) report

Other showings of importance on the Nixon Fork project include the Warrior prospect, Whelan and the Northstar zones. The Pupinsky prospect is the only known significant mineral prospect on the southeast side of the intrusion. It consists of a magnetite-sulfide skarn that differs from the Nixon Fork mine area in that gold is absent at Pupinsky while copper, silver, tin and minor tungsten values are elevated. Cassiterite is locally abundant in late stage veins and in association with tourmaline (Bundtzen, 1999).

10 Exploration

Flanders, Giroux & Rawsthorne (2010, pp. 37-38) report

> Samples collected in the historic core are essentially field duplicates. Intervals are marked, and tags placed in the core boxes. All remaining core is placed into sample bags for analysis. Because the 2007 and 2008 results were not formerly reported, a careful review of all quality control and quality assurance procedures used by the former operator was initiated. Assay certificates were verified with the digital database and the results from blank and certified standards were compiled. This review shows more than an adequate number of certified standards had been used along with numerous blanks and duplicates.
>
> Five different gold standards were used in 2008 that ranged from 1.02 to 30.04 grams per tonne. These are commercial standards produced by Rock Labs. A total of 49 standards were submitted for this program. Acceptable values for standards form a range that lies between two standard deviations below and above the mean.
>
> During 2008, 22 of the submitted standards fell outside the acceptable limits. Of those 22 standards 17 were low to marginally low and generally within 3 standard deviations of the mean. Five of those standards reported values that were higher than the accepted limits. All blank samples submitted were in acceptable ranges and the duplicate samples showed typical variability for this type of gold system.
>
> Historic results for 2007 and 2008 drill programs are shown Appendix 2. The intercepts were calculated in Gemcom using a cutoff grade of 3gpt Au, a minimum width of 0.5 meters, and with up to 1m of core below the minimum cutoff grade included in the sample. Only those intercepts greater than 5 gpt Au are shown. Although all high grade intercepts typical have significant copper and silver credits, these elements were not included in the cutoff.
>
> A significant number of historic intercepts were re-assayed as described above. Although every attempt was made to match the historical interval, in some cases this was not possible. Table 10-1 shows a comparison of the 2010 assays to the historical assays for the 2008 drill holes. In general, given the coarse gold present at Nixon Fork, there is good replication of intervals and grades. The two most significant discrepancies are in two intervals in drill holes N08U011(+1500% change) and N08U023(-39% change). The average difference in gold values using all the re-assayed intervals is +2.85 gpt; if the two most anomalous results from holes 11 and 23 are removed, the difference changes to -2.38 gpt, indicating a significant nugget effect.
>
> As per section 12 of NI43-101-F1, all exploration work conducted by parties other than PFN or Fire River Gold is discussed under "History", "Geologic Setting" or "Mineralization".

Table 10-1 Comparison of 2010 Re-Assays of Select 2008 Drill Hole Intercepts

Hole	Historic Intercepts - 2008				2010 Reassays - Field Duplicate			
Number	From	To	Length(m)	Au(gpt)	From	To	Length(m)	Au(gpt)
N08U001	231.6	239.6	8.0	1.6	231.6	239.2	7.6	0.8
N08U003	235.3	238.7	3.4	11.2	234.7	239.3	4.6	6.3
N08U011	71.6	76.2	4.6	8.7	71.6	76.2	4.6	140.0
N08U012	78.3	83.8	5.5	21.9	78.3	83.8	5.5	15.8
N08U017	35.2	37.5	2.3	24.9	34.6	37.5	2.9	18.7
N08U017	21.5	29.1	7.6	26.8	21.5	28.9	7.4	35.0
N08U021	26.0	29.0	3.0	18.3	26.8	29.9	3.1	15.6
N08U023	33.7	37.9	4.2	9.0	33.8	37.8	4.0	10.8
N08U023	12.5	16.9	4.4	121.6	12.4	17.3	4.9	74.5
N08U023	33.7	37.0	3.3	12.9	33.8	37.8	4.0	10.8
N08U024	36.0	38.2	2.2	14.6	36.0	38.1	2.1	8.2
N08U025	14.0	20.4	6.4	13.1	13.6	20.1	6.5	15.8
N08U025	14.0	26.6	12.6	8.9	14.0	26.8	12.8	8.6
N08U027	14.0	17.8	3.8	11.2	14.0	17.7	3.7	8.1
N08U027	14.0	17.8	3.8	11.2	14.0	17.7	3.7	8.1
N08U030	21.4	24.6	3.1	45.1	22.6	22.8	2.2	44.9
N08U031	29.4	32.0	2.6	65.4	29.4	32.1	2.7	52.9

11 Drilling

11.1 Historic drilling

As per section 12 of NI43-101-F1, all exploration work conducted by parties other than PFN or Fire River Gold is discussed under "History", "Geologic Setting" or "Mineralization".

11.2 Current drilling

Flanders, Giroux & Rawsthorne (2010, pp. 38-39) report

> As of the date of this report, Fire River has begun surface drilling in the Whalen and Northstar areas with a total of 11 holes completed. In addition, drilling has begun underground with 6 holes completed in the 3100 zone and 5 holes completed in the 3300. Assay results are pending for all holes at the time of this report.
>
> Modern exploration and development drilling on the Nixon Fork project began in 1985 and has been conducted intermittently to as recently as 2008 (Table 11-1). Wallis and others (2003) reported that surface core drilling conducted before 2003 was HQ (63.5 mm in diameter) while the underground core drilling was BQ in size (36.4 mm in diameter). Collars for all the surface exploration and underground holes were surveyed in addition to down hole surveys at varying intervals of 20 to 60 m, depending on the length of the hole.

Table 11-1 Summary of Exploration Drilling - 1985 through 2008*

Company	Year	Type	# Holes	Meters	Target Area
Battle Mt. Gold	1985-1988	Surface RC	85	7,342	Various
Nixon Fork JV	1989	Surface HQ Core	18	1,463	Various
Nixon Fork JV	1990	Surface HQ Core	70	8,874	Various
Nixon Fork JV	1993	Surface HQ Core	23	3,638	Various
Nevada Goldfields	1994	Surface HQ Core	71	5,985	Various
Nevada Goldfields	1994	Underground BQ Core	43	1,764	Various
Nevada Goldfields	1996	Surface HQ Core	69	6,465	Various
Nevada Goldfields	1996	Underground BQ Core	117	7,110	Various
Nevada Goldfields	1997	Surface HQ Core	30	3,012	Various
Nevada Goldfields	1997	Underground BQ Core	163	13,411	Various
Nevada Goldfields	1998	Underground BQ Core	30	4,030	Various
St. Andrew	2004-2005	Underground NQ Core	121	11,875	3000/3300
St. Andrew	2004-2005	Underground NQ Core	32	5,539	J5A
St. Andrew	2004	Surface BQ Core	1	64	Whalen
St. Andrew	2005-2006	Underground NQ Core	17	2,761	J2100
St. Andrew	2005-2006	Underground NQ Core	15	1,999	3550
St. Andrew	2005-2006	Underground NQ Core	10	1,100	3300
St. Andrew	2005-2006	Surface NQ Core	21	2,850	Whalen
St. Andrew	2005-2006	Underground NQ Core	35	3,965	Mystery
St. Andrew	2005-2006	Surface NQ Core	4	517	Mystery
St. Andrew	2005-2006	Surface NQ Core	11	869	Warrior
St. Andrew	2005-2006	Underground NQ Core	8	498	3000 packer
St. Andrew	2007	Underground NQ Core	89	5,455	3300
St. Andrew	2007	Surface NQ Core	7	726	Whalen
St. Andrew	2008	Underground NQ Core	33	2,997	3300
Total			**1,123**	**104,307**	

*Data compiled from Wallis and others (2003), Postle and others (2006) and St. Andrew Goldfields, 2009, oral comm.

With the exception of one BQ (36.4 mm in diameter) surface drill hole that was drilled in the Whalen area in 200, Wallis and Rennie (2005) reported that, after St. Andrew acquired the Nixon Fork project and commenced drilling in 2004, all of the surface and underground drilling was conducted with NQ (47.6 mm) size core.

12 Sampling method and approach

Flanders, Giroux & Rawsthorne (2010, p. 40) report

> Wallis and others (2003) reported that prior to 2003; drill core was logged on site, samples marked and the remaining half core stored on site. Special attention was paid to lithology, alteration oxidation, sulfide and gold content, bedding, foliation, and intrusive contacts. Most of the previous core is stored on site and is in reasonable shape. Some of the underground drill core has been retained however, much has been discarded.
>
> Postle and others (2006) reported that after 2004, core was either sawn in half or split with a hydraulic splitter. Samples, selected by geological contacts and the sulfide content, were generally taken in 0.5 to 1.5 m intervals and bagged for shipment. Larger samples were confined to visually low sulfide-bearing rocks.
>
> For diamond drill core, each sample of half core was put into a sample bag, securely fastened and included in a large shipping bag which is then flown by charter aircraft to the laboratory in Fairbanks.
>
> During the re-logging campaign of 2010 for drill core of 2007 and 2008, samples collected were essentially field duplicates. That is the other half of the core was placed in a sample bag, there was no remaining sample left in the core box. This method provided the most statistically meaningful method of comparing historic results to the results of 2010. Once placed in a 6 mil plastic bag the samples were securely sealed and placed in large bags for shipping. These bags were flow out by charter aircraft to the prep laboratory in Fairbanks (ALS Chemex).

13 Sample preparation, analyses, and security

Flanders, Giroux & Rawsthorne (2010, pp. 41-42) report

> For the programs carried out prior to 1999, the sample preparation and assaying was carried out by Chemex Laboratories and other recognized assay labs using standard industry methods (Postle and others, 2006). Although all the samples were fire assayed for gold, the "finish" method varied and included AA, gravimetric, and metallic screens. During the period 1989 through 1994, all samples were assayed for gold, silver, copper, and arsenic. After the commencement of production by Nevada Goldfields, surface and underground samples were normally prepared and fire assayed at the mine assay lab using standard industry methods.
>
> After the commencement of production by Nevada Goldfields, surface and underground samples were normally prepared and fire assayed at the mine assay lab using standard industry methods (Wallis and others, 2003). Mine samples were dried and crushed to 3/8 inches in a jaw crusher, then split to 200 g to 300 g in a small Jones splitter. The sample was then pulverized in a ring and puck pulverizer, rolled, and a 30 g cut (one-assay ton) taken for fire assay. Samples were mixed with fluxes and fused. After the lead was removed, the beads were weighed and placed into parting cups. Silver was inquarted at three times the bead weight, parted with nitric acid and the remaining gold dried and weighed. Wallis and others (2003) reported that during production from 1994 to 1997, 10% of the samples were duplicated and 5% of the samples were sent to Chemex for checks but no checks were completed after August 1997. The mine lab did not run standards or blanks on a routine basis. KPGM Peat Marwick (1996) audited the mine assay facilities at the Nixon Fork mine and reported some concern regarding the accuracy of sampling and assaying procedures. Copper and silver assays were not routinely run on drill core and underground samples. Wallis and others (2003) reported that the mine lab was dusty, the jaw crusher had not been properly cleaned and large drying oven was located in the mill where it was subject to dust contamination.
>
> Following acquisition of the Nixon Fork project by St. Andrew, all samples were sent to SGS laboratories in Fairbanks for sample preparation and the pulps were sent to SGS Lakefield in Toronto for assay (Wallis and Rennie, 2005). Gold content was determined using a standard one assay ton fire assay with gravimetric finish, with copper and silver determined by aqua regia digestion and AA finish. When SGS closed its sample prep facility in June 2004, the samples were then sent to ALS Chemex in Fairbanks for sample preparation and then to Vancouver for standard fire assay and aqua regia digestion as described above. During the surface exploration program carried out in 2005 and 2006, field standards and blanks were not submitted with the samples but duplicates were run every 20 samples and those that ran greater than 20 gpt were rerun on a regular basis by the laboratory. Commencing in June 2005, the samples were sent to Alaska Assay Labs Inc. in Fairbanks for preparation and then shipped to Inspectorate in Sparks, Nevada for assaying. Inspectorate used the same assay methods as those performed by ALS Chemex.

.

Details relating to sample preparation, analysis and security for the 2007-2008 drilling program at Nixon Fork are not available to the author.

During the 2010 program all assay work was completed by ALS Chemex, with sample preparation in their Fairbanks Laboratory and analysis in the Vancouver Lab. The specific methods include:

Code	Description
LOG-22 LOG-24 LOG-22D	Log in security procedures
WEI-21	Weigh every sample
CRU-31 CRU-QC SPL-21d PUL-31d PUL-31 PUL-QC	Crushing and pulverizing procedures
SPL-21	Premium splitting procedures
WSH-22	Cleaning of pulverizer after every sample with barren material
FA-FUSGV1	Fire Assay Fusion 30gm sample
Au-Grav21	Gravimeteric finish of 30gm
ME-ICP61a	Four acid digestion for ICP-AES method.

Procedures used by ALS Chemex during the 2010 program are described in detail in *ALS Laboratory Group – Minerals, Schedule of Service and Fees, 2010.*

14 Data verification

Flanders, Giroux & Rawsthorne (2010, pp. 43-45) report

> Limited information is available to the author on data verification procedures at the Nixon Fork mine property prior to 2003. The majority of the pre-2003 data verification information that is known to the author and summarized below is referenced in Wallis and others (2003), Wallis and Rennie (2005) and Postle and others (2006).
>
> Wallis and others (2003) reported that Pincock Allen & Holt(1996) audited the Nixon Fork resources and reserves in June 1996. The database included records for 725 holes of which some 366 are surface holes and the remaining 359 were underground holes. The Nixon Fork mine staff audited the database in April 1997 and again in late 1998. It is reported that a few intervals were missing assays but no other errors were found. The missing intervals were found and entered. During a database review conducted by Wallis and others (2003) an additional 133 checks were made on the 1997 underground and surface drill programs. Two intervals in hole 97U-76 were found to be missing mine assays and 18 of the assays in the database from three underground holes were identified as duplicate assays from the lab sheets rather than the originals. Because of the small number of samples involved RPA believes that the differences in the values would not make a significant difference in the previously completed resource estimate. To insure database integrity, Wallis and others (2003) recommended that that the resource database be carefully audited and that the resource model be updated with the 1998 database and that the 1998 database be used in all future estimations of the resources.
>
> Wallis and others (2003) also reported that they reviewed the 1999 NGI Surpac model for the Nixon Form mine and identified a few minor errors in the database that they believed would have no noticeable effect on the estimation of the resources.
>
> Wallis and others (2003) reported that during their work in 2003 they located the results of 68 check assays completed by ALS Chemex in 1997 on the underground and surface drilling. The samples were analyzed by fire assay methods but the finish type was unknown. They also recovered 34 pulps at random from the 1996 and 1997 drilling. The pulps were analyzed by fire assay with gravimetric finish by Assayers Canada in Vancouver. Compared to the mine assays, The average difference between the mine assay results and the ALS Chemex check assay results is -2.62 gpt gold and -15.2%. The average difference between the mine assay results and the Assayers Canada check assays is –13.4 gpt gold and –17.0 %. Wallis and others (2003) noted that five pairs of assays exhibited extreme grade variances, the largest of which was a 0.03 gpt Au mine sample that returned 59.97 gpt Au when check assayed by ALS Chemex. Given the coarse gold commonly seen in Nixon Fork ore, such extreme variances may be caused by nugget effect. Wallis and others (2003) recommended that an independent laboratory carry out additional check assays on all available pulps used for resource estimation purposes.

As a follow up to recommendations made by Wallis and others (2003, Wallis and Rennie presented a comprehensive quality assurance/quality control procedure for all sampling conducted at Nixon Fork after 2003. This program consisted of insertion of a series of duplicates, blanks and standards at various points in the sample preparation and analysis sequence. This QA/QC procedure was followed by St. Andrew beginning with the 2004 exploration program.

Of the 65 standards that were run during the 2004 drill program, Wallis and Rennie (2005) reported that 13 returned values above three standard deviations from the accepted value (20% failure rate). One assay (#2680) appears to be a mislabeling of the standard. Standard GS5 assayed at SGS was the worst performer. In general the labs appear to show a low bias, lending a possible conservative element to resource estimates completed with these assays. Wallis and Rennie (2005) recommended rerunning samples with significant variance to try to determine a more accurate value for these samples.

Wallis and Rennie (2005) reported that "barren-looking limestone" was used as a blank for the 2004 drilling program. The source of this limestone was not specified but is assumed to be from the Nixon Fork area. These blank samples generally assayed less than 0.05 gpt gold. Three samples returned values in the 0.05 to 0.07 gpt gold range and five samples that returned values >0.07 gpt gold. Four of these high samples were run immediately after a high grade sample, suggesting the anomalously high blank value was the result of contamination during sample preparation. Postle and others (2006) noted that these gold values in what appears to be barren limestone may be the result of low level gold values in the limestone even though it was selected on the basis of no visible mineralization or alteration. Wallis and Rennie (2005) recommended that the five samples be re-run (samples 2460, 2677, 2772, 2789, 2934) and also recommended that a large sample of barren material be sent for crushing and multiple analyses to at least two laboratories and that this be used as a standard blank in the future.

assay values. The certified standards used ranged from 1 gpt to 30 gpt and were inserted into the sample stream approximately every 25 samples. The standards showed generally good reproducibility except for the lower grade (1 gpt) standard which was believed to be defective.

Duplicates and replicates showed a high degree of variability, which is to be expect in a coarse gold system such as Nixon Fork. This variability is discussed in total in section 17, Mineral Resource Estimates.

15 Adjacent properties

Snowden is not aware of any information relating to properties adjacent to Nixon Fork relevant for disclosure in this report.

Flanders, Giroux & Rawsthorne (2010, pp. 43-45) report

Digital mining claim records published by the State of Alaska indicate that there are no third-party mining claims in the Nixon Fork area. Fairbanks-based Doyon Ltd., one of 13 Alaska native corporations, owns surface and mineral rights to extensive tracts of land to the east and south of the Nixon Fork mine site. Little public geological or geochemical data are available from these lands so their relationship to mineralization on the Nixon Fork project is uncertain.

16 Mineral processing and metallurgical testing

Flanders, Giroux & Rawsthorne (2010, pp. 45-48) report

Unless otherwise noted, the following summary of historic metallurgical testing has been derived from Wallis and others (2003), Wallis and Rennie (2005) and Postle and others, (2006). As of the date of this report, PFN or Fire River Gold has not conducted or contracted others to complete mineral processing or metallurgical testing on the Nixon Fork project.

Several recognized firms carried out metallurgical testing prior to commencement of production in 1995. Denver Mineral Engineers (1995) summarized the bulk sample metallurgical test work on the Crystal oxide ores as follows:

- Gravity recovery of gold 19.6%
- With flotation, overall gold recovery of 81%
- Copper flotation of the oxide ore gave a recovery 15.4% copper with a concentrate grade of 15.5%
- Gravity recovery of gold 33.9% with flotation overall recovery of 91.3%
- Copper flotation gave a recovery of 97.9% copper with a concentrate grade of 28.3%.

The mill operated from 1995 through June 1999. The Nixon Fork mill consists of gravity separation and flotation circuits employing conventional crushing, milling, gravity separation, flotation and concentrate- and tailings-dewatering circuits capable of handling 140 tonnes per day. Tailings are disposed of in a lined facility. Concentrates in the form of filter cake were loaded into polypropylene super sacks for shipment by air to Anchorage and then by ship to Dowa's smelter in Japan. Dore from the gravity circuit was taken to McGrath and mailed to Johnson Matthey in the United States.

Production records from the mine indicated that average mill recovery since start-up was 84.8%. Initially about half of the gold was recovered from the gravity circuit. As more sulfide ores were processed this amount decreased. Typically the gold recovery averaged 83.5% for the oxide ores while the sulfide ores averaged about 90% recovery. Flotation responses varied with the ore type, with higher copper recovery in the sulfide ore resulting in a higher quantity of lower grade concentrate. The average concentrate grade was 16.6% copper containing 300 to 600 gpt Au and an average of 277 gpt Ag. Concentrate penalty elements including arsenic and antimony (reported as combined arsenic plus antimony) averaged 0.58 %, bismuth averaged 0.22% and selenium averaged 0.01 %.

From December 2003 through October 2005, three phases of metallurgical testing were conducted on mined material and tailings from the Nixon Fork mine. In late 2003 MCRI took three 'bulk samples' of ore from the C3000 ore chute (3000) considered to represent types of skarn ore to be milled from the Crystal Mine. Chlumsky, Armbrust & Meyer LLC (CAM) supervised testing of the C3000 bulk sample by Phillips Enterprises, LLC of Golden, Colorado (Bridge Capital, 2008, Chlumsky and others, 2005a, 2005b). The test work included a gravity circuit and a flotation circuit using fresh ore at Nixon Fork, and cyanide leaching of gold from flotation concentrates as well as from the previously processed tailings. CAM was also responsible for the development of a process flow diagram.

Phase 1 test work focused on cyanide leaching of whole ore to maximize the gold recovery into dore form. As the testing proceeded, it became apparent that while the process was technically viable; however it was not shown at this time to be economically viable due to the consumption of cyanide by copper.

The Phase 2 test program focused on a more conventional processing scheme, calling for gravity recovery of coarse free gold followed by flotation of the copper minerals with additional gold recovered to a high-grade copper concentrate. After 25 flotation tests, it was clear that a 25% copper concentrate could be produced on a regular basis with an overall gold recovery of 75% and a copper recovery of approximately 80% (depending on the copper minerals). The recovery depends on the grade, feed rate as well as the oxidation of the ore.

Historically, gold recovery from 1993 to 1998 was approximately 83%; 20% by gravitational separation and 60% by flotation in a copper concentrate, with 17% sent to tailings pond.

The CIL circuit could potentially recover 80% of the gold that was historically lost to the tailings, increasing overall recovery to 97%. In the 2007 mining program, a lesser recovery of 68% was realized. Leaching could potentially increase total recovery in this program to 94%. The positive impact leaching could have had on both prior mining campaigns is demonstrated in Table 16-1.

Table 16-1 Potent recovery for prior mining campaigns

Operator	Period	Tons Mined	Grade Au (opt)	Projected Recovery (%) Gravity + Flotation	Leaching	Total
Nevada	1995 to 1999	134,902	1.2	83%	13%	96%
St. Andrews	2007 (5 mos)	19,957	0.5	68%	25%	93%

17 Mineral Resource and Mineral Reserve estimates

17.1 Mineral resources

Flanders, Giroux & Rawsthorne (2010, pp. 45-48) report

Fire River contracted Giroux Consultants Ltd. to produce a resource estimate for the Nixon Fork Mine and the historic tailings on the Nixon Fork Gold Project in Alaska. The estimate is summarized in Table 17-1.

G.H. Giroux is the qualified person responsible for the resource estimate and visited the site on May 3 to May 5, 2010. Mr. Giroux is a qualified person by virtue of education, experience and membership in a professional association. He is independent of both the issuer and the vendor applying all of the tests in section 1.4 of National Instrument 43-101.

Table 17-1 Resource estimate at a 10 g/t cut-off grade (Giroux 2010)

	Indicated					Inferred				
		Grade		Contained Gold			Grade		Contained Gold	
Zone	Tonnes	g/t	opt	g	oz	Tonnes	g/t	opt	g	oz
Lode Mining:										
3000	15,500	37.3	1.09	577,840	18,578	37,600	29.7	0.87	1,116,344	35,892
3300	68,900	27.5	0.80	1,891,305	60,809	20,900	29.5	0.86	617,177	19,840
3500	1,200	11.7	0.34	14,052	452	0	0.0	0.00	0	0
Whalen	630	11.2	0.33	7,056	227	10	10.2	0.30	102	3
J5	7,500	16.7	0.49	125,025	4,020	660	13.6	0.40	8,963	288
3100	560	11.3	0.33	6,350	204	410	12.4	0.36	5,076	163
Mystery	27,400	23.7	0.69	649,380	20,878	100	18.9	0.55	1,885	61
Southern Cross						11,100	19.6	0.57	218,004	7,009
Subtotal - Lode	***121,690***	***26.9***	***0.78***	***3,271,008***	***105,168***	***70,780***	***27.8***	***0.81***	***1,967,551***	***63,256***
Existing Tailings*	92,000	7.9	0.23	724,040	23,287	48,000	7.4	0.21	353,760	11,377
Total	**213,690**	**18.7**	**0.55**	**3,995,048**	**128,455**	**118,780**	**19.5**	**0.57**	**2,321,311**	**74,633**

*Tailings estimated at 5 g/t cut-off grade for 100% inclusion

17.1.1 Assays

Flanders, Giroux & Rawsthorne (2010, pp. 45-48) report

For the resource estimate at Nixon Fork a total of 1,233 diamond drill holes from both surface and underground sites were available for analysis. A total of 17,693 samples were assayed for gold. Of these 4,762 reported as 0.00 were set to 0.001 g/t Au, 778 reported as <0.03 were set to 0.015 g/t Au, 613 reported as <0.05 were set to 0.025 g/t Au, 197 reported as <0.01 were set to 0.005 g/t Au and 398 reported as blank were set to 0.001 g/t Au. In addition 4,318 gaps between assays were filled with 0.001 g/t Au. This produced a final data base of 22,011 gold assay values.

During the 2009 field season a total of 660 samples were re-assayed from underground drill holes N07U016 to N08U032 by taking ½ of the remaining ½ drill core. These samples were assayed for Au and a suite of other elements.

In most cases the re-assayed core covered sections of drill core previously assayed but in some cases the 2010 samples covered areas not previously sampled. A total of 137 gaps in the historic data base were filled with 2010 assays.

To validate the previous assay values the original assay intervals were compared to the re-assayed results. In cases where the new samples were not exact matches to the original intervals composites were produced to the original from-to intervals. This resulted in 475 samples with an original gold assay and a duplicate check assay.

Figure 17-1 shows a scatter plot for all 475 pairs of gold assays with no bias indicated. Original assays are shown on the X axis and duplicates on the Y axis (both axis values multiplied by 10^3).

The best fit regression line is pulled slightly below the equal value line due to a single outlier. The coefficient of correlation between the two data sets is 0.9224. When four outliers with one or both assays greater than 400 g/t Au were removed the resulting scatter plot is shown as Figure 17-2. Again no sampling bias is indicated with the best fit regression line slightly below the equal value line and a coefficient of correlation equal to 0.7174. The mean of 471 original assays was 20.41 g/t compared to the mean of 471 check assays of 20.75 g/t. A large amount of scatter is indicated which is to be expected with samples exhibiting these level of grades.

Figure 17-1 Scatter plot of 475 Au assay duplicates

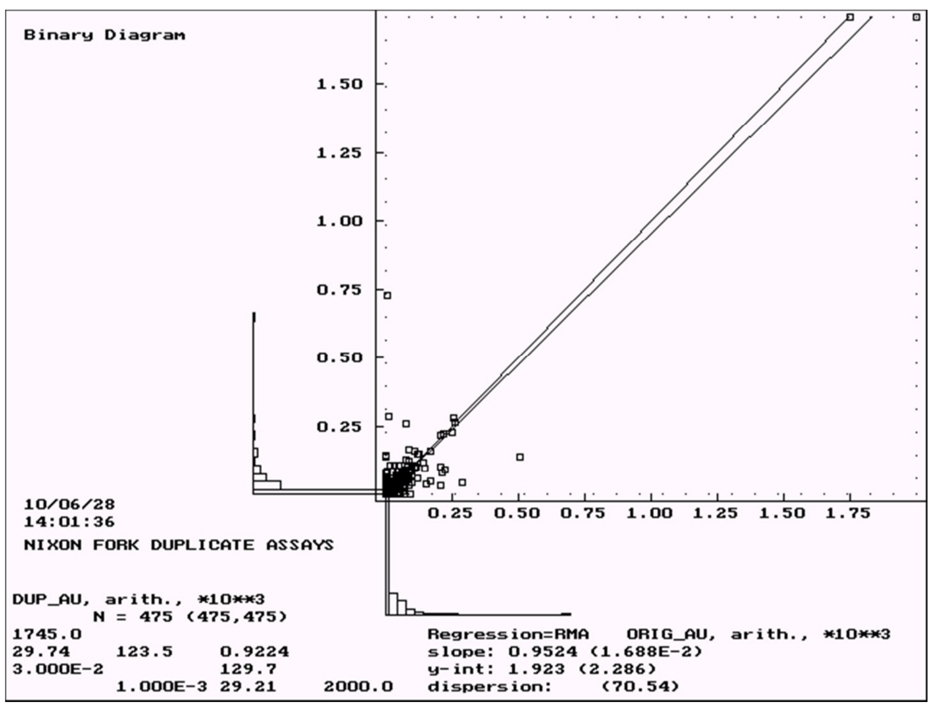
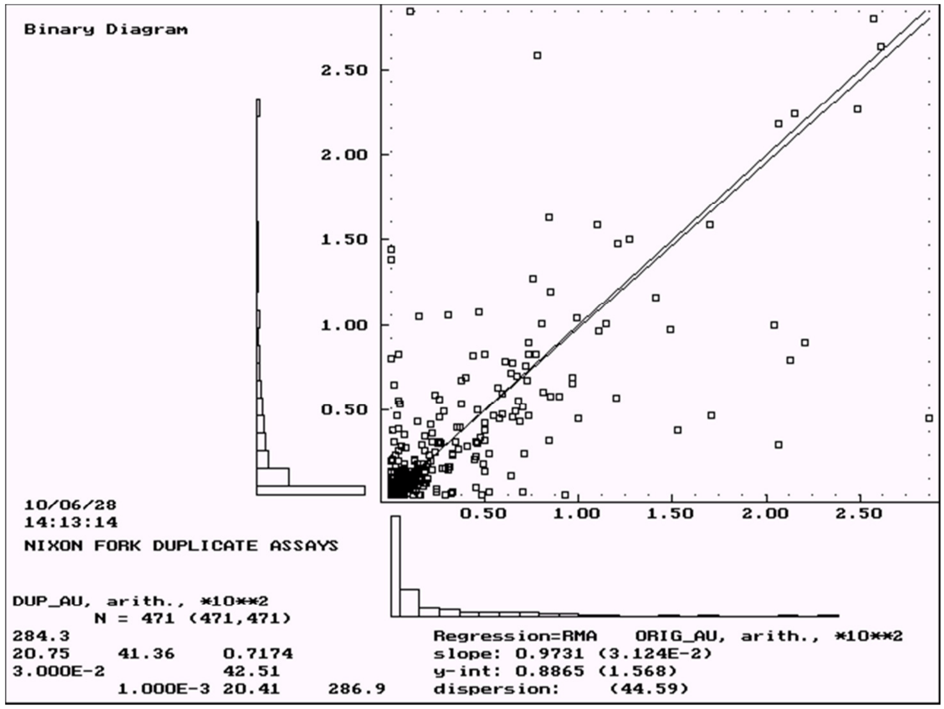


Figure 17-2 Scatter plot: 471 Au assay duplicates, 4 outlier samples removed



17.1.2 Solids

Flanders, Giroux & Rawsthorne (2010, pp. 51-54) report

> A geologic three dimensional model was made in Gemcom by Larry Hillesland using underground and surface drill holes. A total of 17 mineralized solids were produced from cross sections and level plan interpretation, as listed in Table 17-2. Relevant statistics for the solids are shown in Table 17-3.
>
> The mineralized solids for the mines and satellite deposits are shown in 3D projections on through Figure 17-3 to Figure 17-6.

Table 17-2 List of Mineralized Solids Modelled

Name of Solid	Area	Uppermost Plan	Lowermost Plan	Volume m3	Comments
Whalen	Whalen	470	365	18,640	2 zones, looks strat controlled, possibly folded, w/ structural influence
North Star	North Star	470	405	44,240	Complex control, large zone along contact, probable structure influence, Mg Skarn
MystS	Mystery	280	260	1,362	NW frac zone, few DDH's small
MystS2	Mystery	270	250	355	NW frac zone, few DDH's small
M1	Mystery	280	230	5,040	
M2	Mystery			5,211	
M3	Mystery		330	7,609	
M4	Mystery	290	250	5,500	strat contol, and along contact of dike
SC	Southern Cross	390	370	4,242	skies out, open at depth, at NFQM contact, strat controlled?
J5	J5A	360	245	21,134	strat, contact and structural controls, NMHG
3300 zones					
3300-3E		220	195	4,445	NNE frac zone in NFQM
3300-3C		225	215	2,870	pipe like
3300-300		300	220	27,443	offset from 3300-383, clearly some NW-trending frac/flt control, w/ minz in NFQM
3300-3S		215		23,781	offset
3300-383	uppermost	400	300	6,063	upper parts clearly along contact (nose) with strat control, lower parts more likely structurally controlled?
3510	South of 3300	375	345	2,495	strat control, folded
3500	SE of 3300	380	370	544	needs work, exploration, NW frac/flt control

Figure 17-3 Mineralized Solids: Whalen (Purple) and North Star (Green)

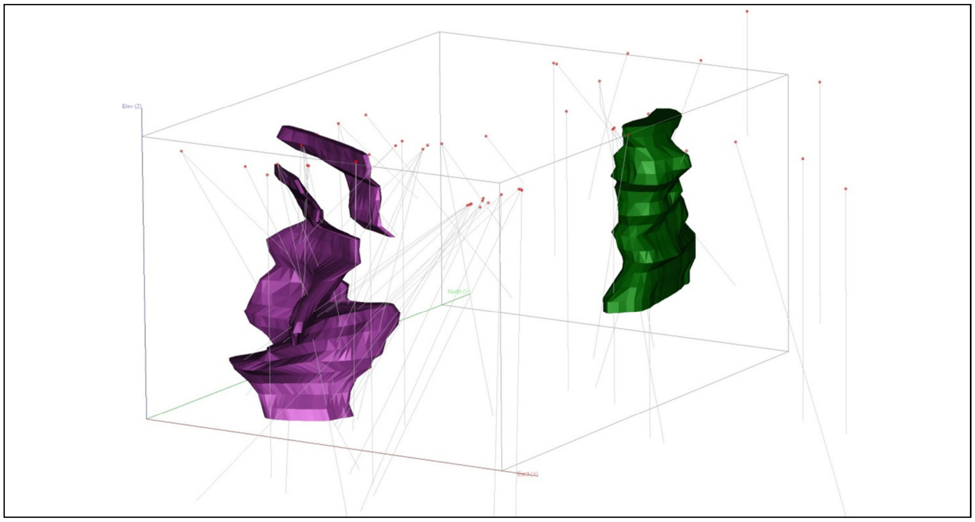


Figure 17-4 Mineralized Solids: 3500, 3300 and 3000 (L to R)

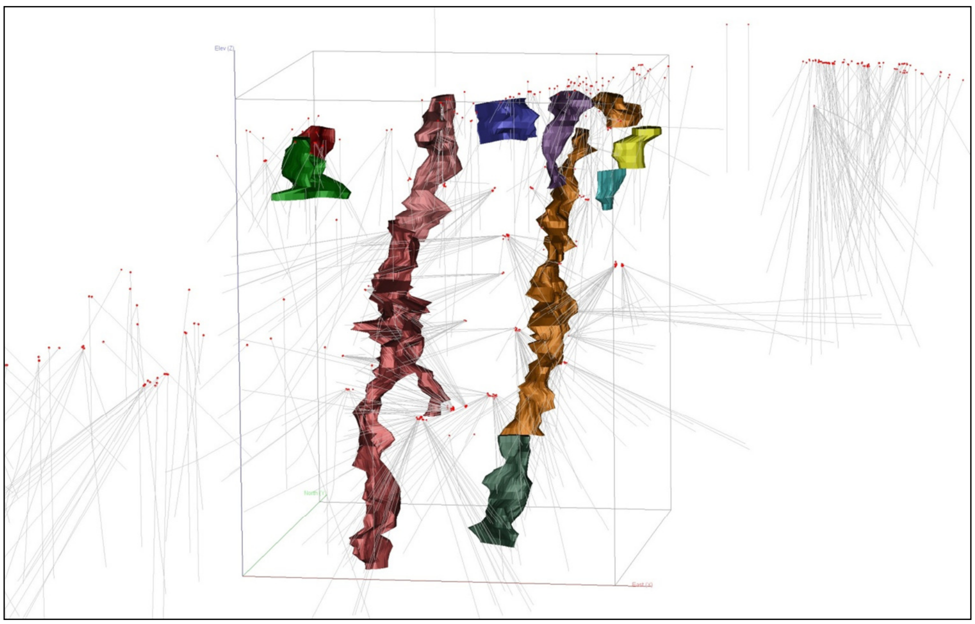


Figure 17-5 Mineralized Solids: 3100, J5 and Southern Cross (L to R)

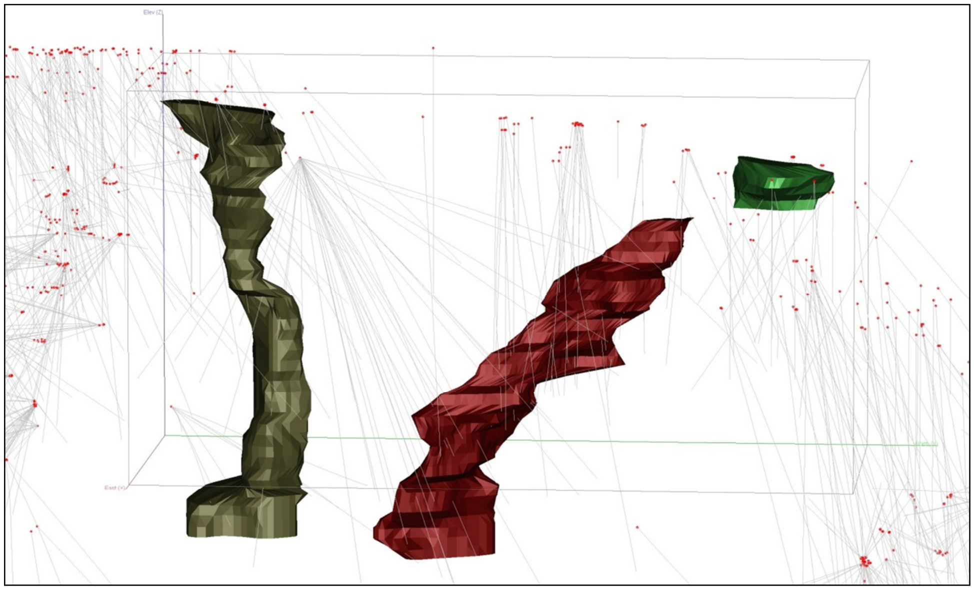


Figure 17-6 Mineralized Solids for the Mystery Zones

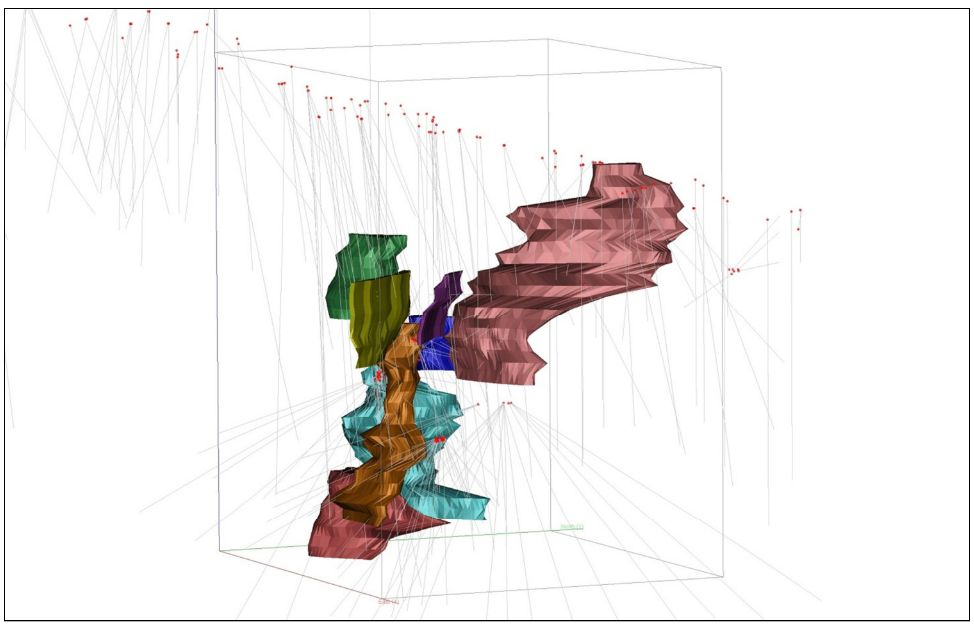


Table 17-3 Statistics for Gold from Mineralized Solids

Zone	Number	Mean Au (g/t)	Standard Deviation	Minimum Value	Maximum Value	Coefficient of Variation
3000D	159	27.38	61.43	0.001	410.50	2.24
3000M	956	35.71	77.54	0.001	871.63	2.17
3000X	49	5.55	17.26	0.001	115.52	3.11
3000Z	54	10.95	47.28	0.001	346.17	4.32
3077	312	30.7	81.11	0.001	543.54	2.64
3100	269	3.15	11.87	0.001	139.22	3.77
3200	51	4.98	9.67	0.001	38.05	1.94
3300-300	1950	13.32	72.03	0.001	2,000.00	5.41
3300-383	117	15.32	41.35	0.001	308.59	2.7
3500N	20	13.7	24.87	0.001	101.77	1.81
3510	104	12.62	70.55	0.001	709.62	5.59
J5	286	4.66	17.09	0.001	182.43	3.67
M1	349	26.34	194.95	0.001	3,209.74	7.4
M3	198	2.91	14.13	0.001	166.24	4.85
M4	38	8.54	25.07	0.001	150.11	2.93
M5	46	7.96	13.48	0.001	63.28	1.69
M6	290	15.1	61.75	0.001	707.40	4.09
M7	106	5.69	21.61	0.001	165.73	3.79
MystS	21	5.85	16.56	0.001	76.44	2.83
MystS2	55	4.38	17.24	0.001	119.26	3.94
NS	236	1.2	3.9	0.001	39.40	3.25
SC	45	9.36	26.55	0.001	128.05	2.84
Whalen	318	2.84	9.69	0.001	119.97	3.41
WASTE	7714	0.63	8.89	0.001	386.06	14.08

17.1.3 Capping

Flanders, Giroux & Rawsthorne (2010, pp. 54-55) report

> For statistical analysis and determination of capping levels the solids were grouped based on location into the following groups:
>
> 3000 Series- 3000D, 3000M, 3000X, 3000Z, 3077, 3200
>
> 3100 Series - 3100
>
> 3300 Series – 3300-300, 3300-383
>
> 3500 Series – 3500N, 3510
>
> J5 Series- J5, SC
>
> Mystery Series- M1, M3, M4, M5, M6, M7, MystS, MystS2
>
> North Star Series- NS
>
> Whalen Series- Whalen
>
> Waste Series- WASTE
>
> For each of the above groups the gold grade distribution was examined using lognormal cumulative frequency plots to determine if capping was necessary

and if so at what level. In all cases gold showed multiple overlapping lognormal populations. In each case the individual populations were partitioned out and a capping level established to reduce the effect of high grade outliers. The cap levels and number of samples capped are tabulated in Table 17-4 below.

Table 17-4 Capping Strategy for Mineralized Groups

Group	Capping Strategy	Cap Level	Number Capped
3000	2SDAMP2	440 g/t	10
3100	2SDAMP2	52 g/t	2
3300	2SDAMP2	326 g/t	7
3500	2SDAMP2	34 g/t	7
J5	2SDAMP3	76 g/t	7
Mystery	2SDAMP4	226 g/t	14
NS	2SDAMP1	92 g/t	0
Whalen	2SDAMP2	48 g/t	2
Waste	2SDAMP4	13 g/t	41

The statistics for gold after capping are tabulated below in Table 17-5 for the various mineralized groups.

Table 17-5 Statistics for Capped Gold from Mineralized Solids

Group	Number	Mean Au Au (g/t)	Standard Deviation	Minimum Value	Maximum Value	Coefficient of Variation
3000	1581	30.4	68.65	0.001	440	2.26
3100	269	2.73	8.28	0.001	52	3.04
3300	2067	11.48	36.51	0.001	326	3.18
3500	1214	5.6	10.08	0.001	34	1.8
J5	331	4.47	13.34	0.001	76	2.98
Mystery	1103	8.86	32.19	0.001	226	3.63
NS	236	1.2	3.9	0.001	39.4	3.25
Whalen	318	2.56	7.15	0.001	48	2.79
WASTE	7714	0.27	1.24	0.001	13	4.51

17.1.4 Composites

Flanders, Giroux & Rawsthorne (2010, pp. 55-56) report

The drill holes were "passed through" the mineralized solids with the point each hole entered and left each solid recorded. Uniform down hole 2 m composites were formed that honoured the solid boundaries. Composites less than 1.0 m at the solid boundaries were combined with adjoining samples to produce composites of uniform support of 2 ± 1 m in length. After forming the composites a total of 100 isolated single composites less than 0.5 m in length were dropped from the data base. The 2 m composite file statistics are tabulated below in Table 17-6.

Table 17-6 Statistics for 2 m Gold Composites

Group	Number	Mean Au (g/t)	Standard Deviation	Minimum Value	Maximum Value	Coefficient of Variation
3000	1,314	22.77	46.75	0.001	389.6	2.05
3100	246	1.65	4.37	0.001	28.35	2.65
3300	1,532	9.02	26.52	0.001	293.79	2.94
3500	214	2.1	5.26	0.001	29.29	2.5
J5	358	2.82	9.05	0.001	76	3.21
Mystery	938	5.11	17.96	0.001	188.27	3.52
NS	156	1.04	2.38	0.001	16.22	2.28
Whalen	335	1.68	4.42	0.001	30.92	2.62
WASTE	21,077	0.06	0.4	0.001	9.99	7.13

17.1.5 Variography

Flanders, Giroux & Rawsthorne (2010, pp. 56-57) report

> Pairwise relative semivariograms were used to model gold in the eight different zones, as shown in Table 17-7. In all cases, but Whalen and North Star, a geometric anisotropy was demonstrated. In the case of Whalen a simple isotropic model was fit to the data. The North Star had too few points to model so the nearby Whalen model was used. Likewise the 3500 zone had insufficient data to establish a model so the nearby 3300 zone model was used.
>
> In all cases a nested spherical model was used. The nugget to sill ratio varied from a low of 28% in the Mystery Zone to a high of 58% in the 3300 zone. The nugget to sill ratio is an indication of sampling variability with the higher the percentage the more variability present. The parameters for each model are shown in Table 17-7.

Table 17-7 Summary of Semivariogram Parameters

Zone	Azimuth/ Dip	C_0	C_1	C_2	Short Range (m)	Long Range (m)
3000	015 / 0	0.4	0.7	0.25	7	15
	285 / -30	0.4	0.7	0.25	6	10
	105 / -60	0.4	0.7	0.25	4	6
3300	010 / 0	0.8	0.35	0.24	7	18
	280 / -40	0.8	0.35	0.24	3	16
	100 / -50	0.8	0.35	0.24	6	36
Mystery	030 / 0	0.4	0.8	0.22	10	40
	300 / -45	0.4	0.8	0.22	10	30
	120 / -45	0.4	0.8	0.22	20	40
J5	180 / -40	0.72	0.34	0.34	10	20
	090 / 0	0.72	0.34	0.34	18	25
	270 / 0	0.72	0.34	0.34	18	25
3100	045 / 0	0.8	0.4	0.4	12	30
	315 / 0	0.8	0.4	0.4	5	10
	000 / -90	0.8	0.4	0.4	15	30
Whalen	Omni Directional	0.6	0.54	0.16	6	40
Waste	Omni Directional	0.15	0.25	0.15	18	30

17.1.6 Bulk Density

Flanders, Giroux & Rawsthorne (2010, pp. 57-58) report

The following is taken from Scott Wilson Roscoe Postle, 2006:

St Andrew carried out 113 bulk density measurements of diamond drill core. Prior to 2003, the average bulk density was assigned by ore-type according to the following list:

- *Oxide: 2.52 t/m3*
- *Mixed: 2.72 t/m3*
- *Sulphide: 2.93 t/m3*

As no significant bulk density studies are available since this work these "broad brush" averages are applied as shown in Table 17-8 .

Table 17-8 Bulk Densities by Zone

Name of Solid	Area	Strike or Trend	Dip or Plunge	Ore Type	Specific Gravity
Whalen - North Star Zones					
Whalen	Whalen	96	-68	Ox	2.52
North Star	North Star	328	-67	Ox	2.52
Near 3300 - (west and south)					
3510	South of 3300	22	-80	Ox	2.52
3500N	SE of 3300	308	-46	Ox	2.52
3300 zones					
3300-300		175	-62	Ox	2.52
3300-383	uppermost	221	70	Ox	2.52
3200		34	-85	Ox	2.52
3000 Zones					
3000Z	east of 3000	271	-82	Ox	2.52
3077	apophyse of 3000	203	-71	Ox	2.52
3000M	Mined Out	119	-74	Ox	2.52
3000D	deep extension to 3000	203	-73	Ox	2.52
3000X		239	-57	Ox	2.52
3100		22	-80	Mixed	2.72
J5	J5A	186	-50	Mixed	2.72
SC	Southern Cross	286	-43	Mixed	2.72
Mystery Zones					
MystS	Mystery	127	-74	Mixed	2.72
MystS2	Mystery	129	-83	Mixed	2.72
M1	Mystery	200	-59	Mixed	2.72
M2	Mystery	152	-74	Mixed	2.72
M4	Mystery	229	-72	Mixed	2.72
M5	Mystery	44	-55	Mixed	2.72
M6	Mystery	101	-71	Mixed	2.72
M7	Mystery	208	-80	Mixed	2.72

It is recommended that during the next drill campaign a systematic check of bulk density is completed on the new drill core. The potential exists for heavier rock in skarn zones but the potential for voids also exists particularly in the breccia zones. A better understanding of bulk density is necessary going forward.

17.1.7 Block Models

Flanders, Giroux & Rawsthorne (2010, pp. 58-61) report

Due to the small block size, a total of 4 block models were created to cover the various mineralized zones to be estimated. The models all used 2 x 2 x 2 m blocks and the origin of each model was set so that all models could be combined into a single large model if required.

The model origins were as follows:

Model 1 – to cover Whalen and North Star Deposits

Lower Left Corner		
Easting – 411300 E	Column Size = 2 m	85 Columns
Northing – 7011470 N	Row Size = 2 m	115 Rows
Top of Model		
Elevation – 480	Level Size = 2 m	60 Levels
No Rotation		

Model 2 – to cover 3000, 3300 and 3500 Deposits

Lower Left Corner		
Easting – 411190 E	Column Size = 2 m	139 Columns
Northing – 7012896 N	Row Size = 2 m	147 Rows
Top of Model		
Elevation – 428	Level Size = 2 m	178 Levels
No Rotation		

Model 3 – to cover 3100, SC and J5 Deposits

Lower Left Corner		
Easting –411528 E	Column Size = 2 m	74 Columns
Northing –7013154 N	Row Size = 2 m	170 Rows
Top of Model		
Elevation – 430	Level Size = 2 m	94 Levels
No Rotation		

Model 4 – to cover Mystery Deposits

Lower Left Corner		
Easting –411708 E	Column Size = 2 m	96 Columns
Northing – 7013460 N	Row Size = 2 m	85 Rows
Top of Model		
Elevation – 376	Level Size = 2 m	117 Levels
No Rotation		

The models are shown in 3D projection on Figure 17-7 through Figure 17-9 .

Figure 17-7 Block Models in Plan View showing Models 1 through 4 (S to N)

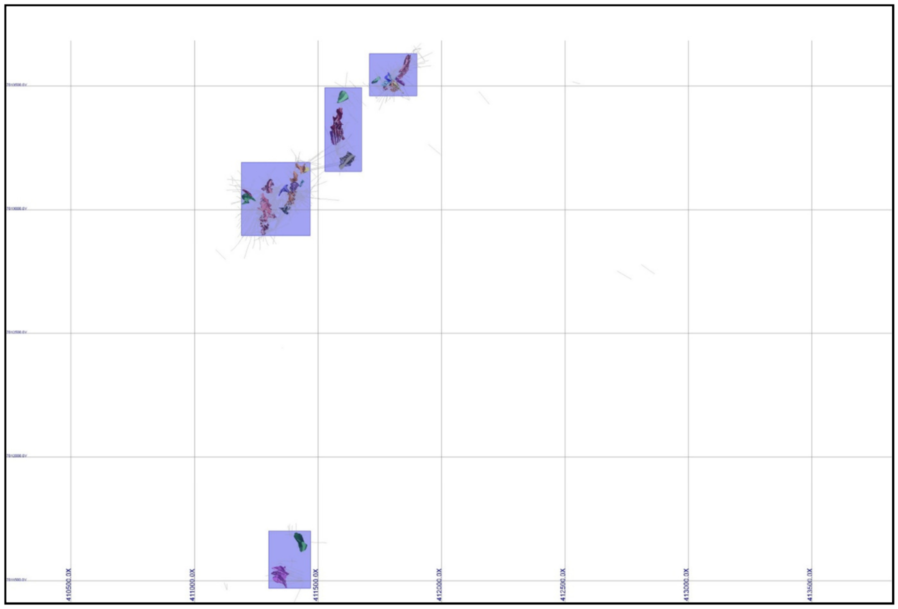


Figure 17-8 Block Models Lokking North, Models 1 through 4 (W to E)

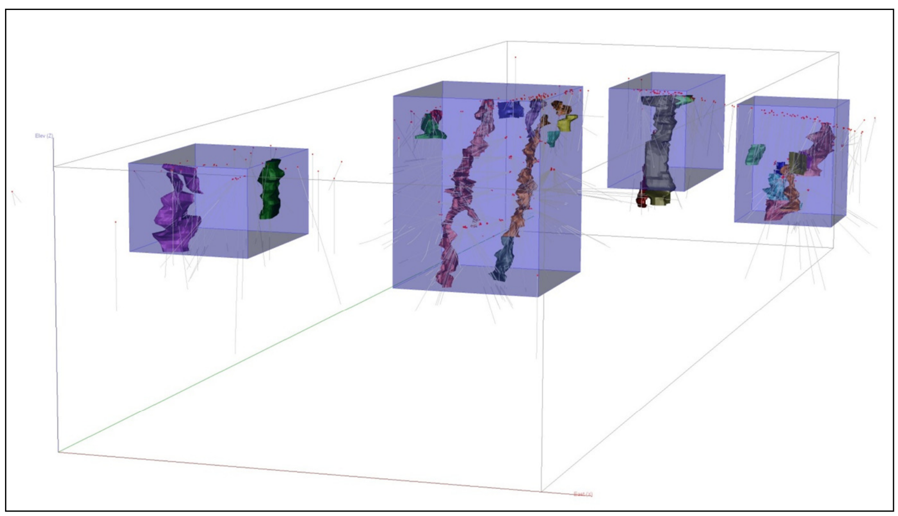
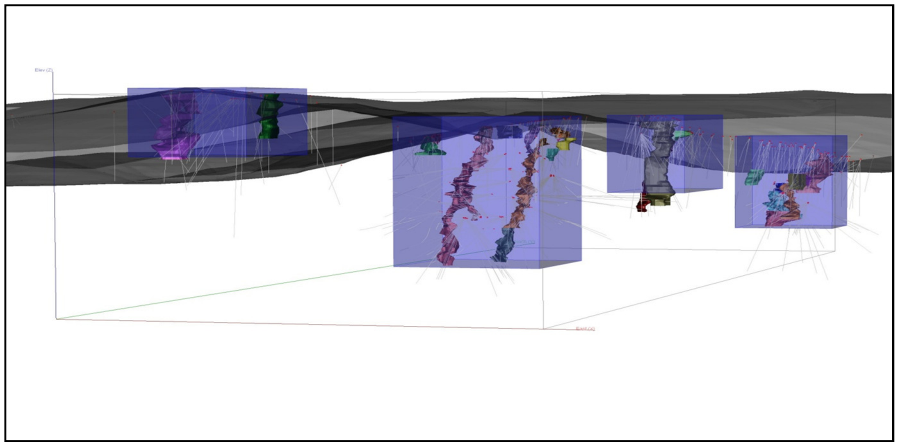


Figure 17-9 Block Models Looking North Showing Surface Topography



17.1.8 Grade Interpolation

Flanders, Giroux & Rawsthorne (2010, pp. 62-64) report

> Gold grades were interpolated into blocks using ordinary kriging. The kriging exercise was completed in a series of passes with the search ellipse dimensions determined by the semivariogram range. I all cases a 1st pass was completed requiring a minimum of four composites to be located within a search ellipse with dimensions equal to ¼ the semivariogram range. For blocks not estimated in Pass 1 the search ellipse was expanded to ½ the semivariogram range and Pass 2 was completed. A third pass using the full range and in some cases a fourth pass using twice the range completed the kriging exercise. In all passes if more than 12 composites were found the closest 12 were used. In all passes a maximum three composites from any drill hole was set which insured a minimum of two holes were used to estimate any block.
>
> Kriging within Model 1 was completed for the North Star (NS) solid and the Whalen Solid using only composites from within each solid. Both estimates used the Whalen variography.
>
> Within Model 2 the 6 individual solids making up the 3000 zone were estimated from composites coded as 3000 zone. The two solids that made up the 3300 zone and the two solids that made up the 3500 zone were estimated from 3300 zone and 3500 zone composites respectively. The variography for the 3000 zone was used to estimate the 3000 zone solids while the 3300 variography was used to estimate both the 3300 and 3500 solids.
>
> A summary of Kriging parameters by zone is shown in Figure 17-9.

Table 17-9 Summary of Kriging Parameters by Zone

ZONE	SOLID	PASS	NUMBER	AZ/DIP	DIST. (m)	AZ/DIP	DIST. (m)	AZ/DIP	DIST. (m)
Whalen	Whal	1	5694	Omni Dir.	10				
	100% Est.	2	4369	Omni. Dir	20				
		3	738	Omni Dir.	40				
North Star	NS	1	2366	Omni Dir.	10				
	100% Est.	2	8005	Omni. Dir	20				
		3	1259	Omni Dir.	40				
3000 Zone	3000D	1	5	15 / 0	3.75	285 / -30	2.5	105 / -60	1.5
	99% Est.	2	576	15 / 0	7.5	285 / -30	5	105 / -60	3
		3	2271	15 / 0	15	285 / -30	10	105 / -60	6
		4	789	15 / 0	30	285 / -30	20	105 / -60	12
	3000M	1	236	15 / 0	3.75	285 / -30	2.5	105 / -60	1.5
	100% Est.	2	4238	15 / 0	7.5	285 / -30	5	105 / -60	3
		3	6104	15 / 0	15	285 / -30	10	105 / -60	6
		4	1239	15 / 0	30	285 / -30	20	105 / -60	12
	3000X	1	0	15 / 0	3.75	285 / -30	2.5	105 / -60	1.5
	100% Est.	2	9	15 / 0	7.5	285 / -30	5	105 / -60	3
		3	771	15 / 0	15	285 / -30	10	105 / -60	6
		4	825	15 / 0	30	285 / -30	20	105 / -60	12
	3000Z	1	13	15 / 0	3.75	285 / -30	2.5	105 / -60	1.5
	90.7% Est.	2	128	15 / 0	7.5	285 / -30	5	105 / -60	3
		3	246	15 / 0	15	285 / -30	10	105 / -60	6
		4	190	15 / 0	30	285 / -30	20	105 / -60	12
	3077	1	35	15 / 0	3.75	285 / -30	2.5	105 / -60	1.5
	100% Est.	2	980	15 / 0	7.5	285 / -30	5	105 / -60	3
		3	1688	15 / 0	15	285 / -30	10	105 / -60	6
		4	68	15 / 0	30	285 / -30	20	105 / -60	12
	3200	1	0	15 / 0	3.75	285 / -30	2.5	105 / -60	1.5
	94.8% Est.	2	36	15 / 0	7.5	285 / -30	5	105 / -60	3
		3	918	15 / 0	15	285 / -30	10	105 / -60	6
		4	1147	15 / 0	30	285 / -30	20	105 / -60	12
3300 Zone	3300-300	1	6800	10 / 0	4.5	280 / -40	4	100 / -50	9
	100% Est.	2	8273	10 / 0	9	280 / -40	8	100 / -50	18
		3	2235	10 / 0	18	280 / -40	16	100 / -50	36
		4	62	10 / 0	36	280 / -40	32	100 / -50	72
	3300-383	1	727	10 / 0	4.5	280 / -40	4	100 / -50	9
	100% Est.	2	3056	10 / 0	9	280 / -40	8	100 / -50	18
		3	1750	10 / 0	18	280 / -40	16	100 / -50	36
		4	82	10 / 0	36	280 / -40	32	100 / -50	72

Table 17-9 continued...

ZONE	SOLID	PASS	NUMBER	AZ/DIP	DIST. (m)	AZ/DIP	DIST. (m)	AZ/DIP	DIST. (m)
3500 Zone	3500N	1	129	10 / 0	4.5	280 / -40	4	100 / -50	9
	100% Est.	2	551	10 / 0	9	280 / -40	8	100 / -50	18
		3	282	10 / 0	18	280 / -40	16	100 / -50	36
		4	3	10 / 0	36	280 / -40	32	100 / -50	72
	3510	1	1089	10 / 0	4.5	280 / -40	4	100 / -50	9
	100% Est.	2	2050	10 / 0	9	280 / -40	8	100 / -50	18
		3	210	10 / 0	18	280 / -40	16	100 / -50	36
J5 Zone	J5	1	1408	180 / -40	5	90 / 0	6.25	270 / 0	6.25
	100% Est.	2	5806	180 / -40	10	90 / 0	12.5	270 / 0	12.5
		3	3941	180 / -40	20	90 / 0	25	270 / 0	25
		4	68	180 / -40	40	90 / 0	50	270 / 0	50
	SC	1	294	180 / -40	5	90 / 0	6.25	270 / 0	6.25
	100% Est.	2	1171	180 / -40	10	90 / 0	12.5	270 / 0	12.5
		3	1485	180 / -40	20	90 / 0	25	270 / 0	25
		4	26	180 / -40	40	90 / 0	50	270 / 0	50
	3100	1	380	45 / 0	7.5	315 / 0	2.5	0 / -90	7.5
	98.5% Est.	2	3877	45 / 0	15	315 / 0	5	0 / -90	15
		3	5122	45 / 0	30	315 / 0	10	0 / -90	30
		4	4500	45 / 0	60	315 / 0	20	0 / -90	60
Mystery	M1	1	3923	30 / 0	10	300 / -45	7.5	120 / -45	10
	100% Est.	2	2884	30 / 0	20	300 / -45	15	120 / -45	20
		3	112	30 / 0	40	300 / -45	30	120 / -45	40
	M3	1	2285	30 / 0	10	300 / -45	7.5	120 / -45	10
	100% Est.	2	561	30 / 0	20	300 / -45	15	120 / -45	20
		3	10	30 / 0	40	300 / -45	30	120 / -45	40
	M4	1	623	30 / 0	10	300 / -45	7.5	120 / -45	10
	100% Est.	2	859	30 / 0	20	300 / -45	15	120 / -45	20
		4	69	30 / 0	40	300 / -45	30	120 / -45	40
	M5	1	815	30 / 0	10	300 / -45	7.5	120 / -45	10
	100% Est.	2	292	30 / 0	20	300 / -45	15	120 / -45	20
	M6	1	3671	30 / 0	10	300 / -45	7.5	120 / -45	10
	100% Est.	2	596	30 / 0	20	300 / -45	15	120 / -45	20
	M7	1	1680	30 / 0	10	300 / -45	7.5	120 / -45	10
	100% Est.	2	573	30 / 0	20	300 / -45	15	120 / -45	20
		3	5	30 / 0	40	300 / -45	30	120 / -45	40
	MYSTS	1	346	30 / 0	10	300 / -45	7.5	120 / -45	10
	100% Est.	2	523	30 / 0	20	300 / -45	15	120 / -45	20
		3	88	30 / 0	40	300 / -45	30	120 / -45	40
	MYSTS2	1	901	30 / 0	10	300 / -45	7.5	120 / -45	10
	100% Est.	2	669	30 / 0	20	300 / -45	15	120 / -45	20
		3	105	30 / 0	40	300 / -45	30	120 / -45	40

17.1.9 Classification

Flanders, Giroux & Rawsthorne (2010, pp. 65-67) report

> Based on the study herein reported, delineated mineralization of the estimated zones within the Nixon Fork Deposit are classified as a resource according to the following definitions from National Instrument 43-101 and from CIM (2005):
>
> > "In this Instrument, the terms "mineral resource", "inferred mineral resource", "indicated mineral resource" and "measured mineral resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition

Standards on Mineral Resources and Mineral Reserves adopted by CIM Council, as those definitions may be amended."

The terms Measured, Indicated and Inferred are defined by CIM (2005) as follows:

"A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge."

"The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase 'reasonable prospects for economic extraction' implies a judgement by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports."

Inferred Mineral Resource

"An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, workings and drill holes."

"Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies."

Indicated Mineral Resource

"An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed."

"Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions."

Measured Mineral Resource

"A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate 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 that are spaced closely enough to confirm both geological and grade continuity."

"Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit."

Geologic continuity has been established through underground mapping and mining and through drill hole interpretation via cross sections and level plans. The grade continuity can be quantified through the use of semivariograms. By tying the search ellipses to the semivariogram range and orientations grade continuity can be used to classify the deposit. Until Fire River completes a current drill program none of this resource is classified as Measured.

Within the Model 1 area:

Whalen – Blocks estimated during pass 1 and 2 using up to ½ the semivariogram range are classified as Indicated. All other blocks are inferred.

North Star - Since there was insufficient data to determine semivariograms all blocks are classified as Inferred.

Within the Model 2 area:

3000M Zones – All blocks are mined out

Remaining 3000 Zones - Blocks estimated in Pass 1 or 2 using up to ½ the semivariogram range were classified as Indicated. All other estimated blocks were classified as Inferred.

3300 Zones – Blocks estimated in Pass 1 or 2 using up to ½ the semivariogram range were classified as Indicated. All other estimated blocks were classified as Inferred.

3500 Zones – Blocks estimated in Pass 1 or 2 using up to ½ the semivariogram range were classified as Indicated. All other estimated blocks were classified as Inferred.

Within the Model 3 area:

J5 Zone - Blocks estimated in Pass 1 or 2 using up to ½ the semivariogram range were classified as Indicated. All other estimated blocks were classified as Inferred.

3100 Zone - Blocks estimated in Pass 1 or 2 using up to ½ the semivariogram range were classified as Indicated. All other estimated blocks were classified as Inferred.

Southern Cross Zone - Since there was insufficient data to determine semivariograms all blocks are classified as Inferred.

Within the Model 4 area:

Mystery Zones - Blocks estimated in Pass 1 or 2 using up to ½ the semivariogram range were classified as Indicated. All other estimated blocks were classified as Inferred.

17.1.10 Resource Estimates by Zone

Flanders, Giroux & Rawsthorne (2010, pp. 67-77) report

The results are summarized cutoff and tabulated by zone at 5 and 10g/t cut-off grades in Table 17-10. These results assume one could mine to the boundaries of the solids and no external edge dilution has been considered.

Table 17-11through to Table 17-26 show the resource estimates varied by cut-off grade for each zone.

Table 17-10 Summary of Resource within Mineralizes Zones at 5 and 10 g/t

Classification	Zone	Au Cutoff (g/t)	Tonnes> Cutoff (tonnes)	Grade > Cutoff Au (g/t)	Ounces Gold
INDICATED	3000	5	20,000	30.4	19,570
	3300	5	112,100	19.7	70,867
	3500	5	5,300	8.3	1,421
	Whalen	5	8,900	7.0	2,008
	J5	5	18,000	11.1	6,408
	3100	5	7,700	7.1	1,761
	Mystery*	5	57,800	14.9	27,763
	TOTAL	**5**	**229,800**	**17.6**	**129,798**
INFERRED	3000	5	63,000	20.6	41,743
	3300	5	30,300	22.6	22,015
	3500	5	40	5.3	7
	Whalen	5	170	6.4	35
	J5	5	2,500	8.7	700
	3100	5	4,900	7.0	1,097
	Mystery	5	360	9.3	107
	NS	5	2,000	6.0	383
	SC	5	19,800	14.5	9,208
	TOTAL	**5**	**123,070**	**19.0**	**75,295**
INDICATED	3000	10	15,500	37.3	18,578
	3300	10	68,900	27.5	60,809
	3500	10	1,200	11.7	452
	Whalen	10	630	11.2	227
	J5	10	7,500	16.7	4,020
	3100	10	560	11.3	204
	Mystery*	10	27,400	23.7	20,878
	TOTAL	**10**	**121,690**	**26.9**	**105,168**
INFERRED	3000	10	37,600	29.7	35,892
	3300	10	20,900	29.5	19,840
	3500	10	0	0.0	0
	Whalen	10	10	10.2	3
	J5	10	660	13.6	288
	3100	10	410	12.4	163
	Mystery	10	100	18.9	61
	NS	10	0	0.0	0
	SC	10	11,100	19.6	7,009
	TOTAL	**10**	**70,780**	**27.8**	**63,257**

*Note: There is a reported 5,410 tonnes averaging 9.49 g/t Au that has been mined from the Mystery Zones but there is no indication of where this has come from or at what cutoff the mining used.

3000 Zones

(Minus the 3000M which has been mined out)

Table 17-11 Indicated Resource withn 3000 Zone

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	26,600	23.61	20,192
2.00	24,400	25.60	20,081
3.00	22,800	27.25	19,974
4.00	21,300	28.93	19,814
5.00	20,000	30.44	19,570
6.00	18,900	31.93	19,405
7.00	18,100	33.14	19,282
8.00	17,200	34.40	19,022
9.00	16,300	35.89	18,807
10.00	15,500	37.28	18,578
11.00	14,600	38.91	18,263
12.00	13,800	40.55	17,989

Table 17-12 Inferred Resource within 3000 Zone

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	99,000	14.18	45,124
2.00	89,400	15.53	44,640
3.00	80,200	17.04	43,927
4.00	70,200	18.96	42,786
5.00	63,000	20.61	41,743
6.00	56,300	22.41	40,566
7.00	50,300	24.31	39,314
8.00	45,400	26.16	38,184
9.00	41,500	27.79	37,084
10.00	37,600	29.69	35,892
11.00	35,000	31.13	35,028
12.00	32,600	32.54	34,104

3300 Zone Resource

(Minus mined out sections between 175 to 200, 275 to 295 and 350 to 370 elevations)

Table 17-13 Indicated Resource within 3300 Zone

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	175,900	13.52	76,460
2.00	152,000	15.42	75,371
3.00	138,900	16.65	74,337
4.00	126,000	17.99	72,861
5.00	112,100	19.66	70,867
6.00	100,900	21.23	68,870
7.00	91,800	22.69	66,980
8.00	82,900	24.34	64,873
9.00	76,000	25.77	62,961
10.00	68,900	27.45	60,809
11.00	63,600	28.86	59,017
12.00	58,600	30.36	57,192

Table 17-14 Inferred Resource within 3300 Zone

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	36,600	19.27	22,672
2.00	35,700	19.72	22,628
3.00	34,000	20.57	22,483
4.00	31,900	21.69	22,243
5.00	30,300	22.60	22,015
6.00	27,400	24.41	21,503
7.00	25,800	25.56	21,198
8.00	23,800	27.08	20,719
9.00	22,200	28.36	20,243
10.00	20,900	29.53	19,840
11.00	19,700	30.67	19,425
12.00	18,900	31.51	19,145

3500 Zone Resource

Table 17-15 Indicated Resource within 3500 Zone

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	28,100	3.51	3,171
2.00	17,400	4.80	2,682
3.00	12,000	5.82	2,245
4.00	7,700	7.14	1,767
5.00	5,300	8.34	1,421
6.00	4,200	9.05	1,222
7.00	3,400	9.74	1,065
8.00	2,800	10.20	918
9.00	2,100	10.80	729
9.50	1,700	11.22	613
10.00	1,200	11.71	452
11.00	630	12.89	261
12.00	460	13.45	199

Table 17-16 Inferred Resource within 3500 Zone

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	3,900	2.71	339
2.00	2,500	3.41	274
3.00	1,800	3.75	217
4.00	490	4.39	69
5.00	40	5.25	7

Whalen Zone Resource

Table 17-17 Indicated Resource within Whalen Zone

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	62,800	3.05	6,160
2.00	38,600	4.04	5,017
3.00	23,700	5.03	3,833
4.00	14,700	6.02	2,844
5.00	8,900	7.02	2,008
6.00	5,500	7.98	1,411
7.00	3,600	8.78	1,016
8.00	2,400	9.47	730
9.00	1,300	10.26	429
10.00	630	11.20	227
11.00	390	11.67	146
12.00	30	12.49	12

Table 17-18 Inferred Resource within Whalen Zone

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	5,100	2.55	417
2.00	2,900	3.33	310
3.00	1,400	4.11	185
4.00	650	4.96	104
5.00	170	6.36	35
6.00	50	8.48	14
7.00	50	8.59	14

J5 Zone Resource

Table 17-19 Indicated Resource within J5 Zone

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	69,900	4.56	10,252
2.00	43,600	6.45	9,041
3.00	31,500	7.97	8,074
4.00	23,100	9.59	7,125
5.00	18,000	11.07	6,408
6.00	14,800	12.28	5,843
7.00	12,200	13.49	5,292
8.00	10,400	14.55	4,866
9.00	8,800	15.65	4,428
10.00	7,500	16.67	4,020
11.00	6,600	17.51	3,716
12.00	5,800	18.37	3,426

Table 17-20 Inferred Resource within J5 Zone

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	35,200	2.33	2,633
2.00	12,400	3.99	1,589
4.00	3,400	7.64	835
5.00	2,500	8.71	700
6.00	2,000	9.60	617
7.00	1,600	10.39	534
8.00	1,100	11.66	412
9.00	830	12.76	341
10.00	660	13.58	288
12.00	460	14.86	220

3100 Resource

Table 17-21 Indicated Resource within 3100 Zone

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	36,200	3.52	4,099
2.00	25,700	4.38	3,622
3.00	18,000	5.19	3,004
4.00	11,000	6.29	2,223
5.00	7,700	7.11	1,761
6.00	5,100	7.97	1,307
7.00	3,400	8.69	950
8.00	2,100	9.50	641
9.00	1,100	10.38	367
10.00	560	11.34	204
11.00	340	11.97	131
12.00	150	12.59	61

Table 17-22 Inferred Resource within 3100 Zone

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	59,800	2.38	4,584
2.00	24,200	3.78	2,941
3.00	12,800	5.00	2,057
4.00	7,600	6.08	1,486
5.00	4,900	6.97	1,097
6.00	2,900	7.97	743
7.00	1,700	9.05	494
8.00	1,000	10.33	332
9.00	590	11.49	218
10.00	410	12.38	163
11.00	300	13.09	126
12.00	180	14.10	82

Mystery Zone Resource

Table 17-23 Indicated Resource within Mystery Zones

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	175,100	6.58	37,037
2.00	123,600	8.71	34,608
3.00	94,100	10.67	32,284
4.00	73,200	12.74	29,978
5.00	57,800	14.94	27,763
6.00	48,200	16.83	26,078
7.00	41,800	18.40	24,732
8.00	36,000	20.18	23,358
9.00	31,000	22.04	21,968
10.00	27,400	23.70	20,878
11.00	24,500	25.27	19,901
12.00	22,300	26.65	19,109

Table 17-24 Inferred Resource within Mystery Zones

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	2,300	3.33	246
2.00	1,300	4.84	202
3.00	730	6.54	153
4.00	530	7.81	133
5.00	360	9.27	107
6.00	170	13.52	74
7.00	100	18.46	59
8.00	100	18.46	59
9.00	100	18.85	61
10.00	100	18.85	61
11.00	100	18.85	61
12.00	100	19.06	61

Note: There is a reported 5,410 tonnes averaging 9.49 g/t Au that has been mined from the Mystery Zones but there is no indication of where this has come from or at what cutoff the mining used.

North Star Resource

Table 17-25 Inferred Resource within North Star Zone

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	60,600	2.23	4,335
2.00	24,000	3.42	2,635
3.00	14,000	4.11	1,850
4.00	5,800	4.98	929
5.00	2,000	5.96	383
6.00	800	6.89	177
7.00	400	7.28	94

Southern Cross Resource

Table 17-26 Inferred Resource within SC Zone

Au Cutoff (g/t)	Tonnes> Cutoff	Grade > Cutoff (g/t)	Ounces Gold
1.00	28,900	10.67	9,913
2.00	25,400	11.93	9,744
3.00	22,200	13.32	9,505
4.00	20,600	14.06	9,314
5.00	19,800	14.46	9,208
6.00	18,600	15.04	8,992
7.00	17,300	15.67	8,716
8.00	15,700	16.52	8,337
9.00	13,600	17.75	7,760
10.00	11,100	19.64	7,009
11.00	9,100	21.58	6,314
12.00	8,500	22.36	6,112

During several periods of mining at Nixon Fork by NGI and later in 2007 by St. Andrews stopes have been developed on the Mystery, 3000M zone and at several elevations on the 3300 zone. The exact volume of rock mined is unknown at this time as access to most of these zones is presently unavailable.

Table 17-27 outlines the estimated tonnage and grade recovered from these various stopes. This material with the exception of Mystery has been removed from the Resource Tables shown above. For each of the mined areas an estimate of material that was present (taken from the Block Models) prior to mining is tabulated below. It is very difficult to compare mined with estimated blocks since the original cutoffs are unknown as are the exact mined shapes. In general the model estimates more tonnes at lower grades at a 1 g/t cutoff.

Table 17-27 Reported and Modelled Grade and Tonnages from Prior Mining

Zone	Tonnes Mined	Au (g/t) Recovered	Company	Elev. Range Of Zone	Est. Tonnes From Model (1.0 g/t Cutoff)	Estimated Au (g/t) from Model
Mystery	5,410	9.5	NGI			
3000M	98,656	46.7	NGI	145 to 390	144,100	25.9
3300	5,440	23.6	NGI	350 to 370	12,500	19.9
3300	8,198	21.8	St. Andrews	275 to 295	9,440	12.4
3300	7,433	16.0	St. Andrews	175 to 200	18,000	13.4
Total	125,137	40.6			184,040	23.6

17.2 Mineral Reserve

NI 43-101 defines a Mineral Reserve by reference to the CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines, 2005 (CIM 2005), thus:

> "A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined."

This study is a "preliminary assessment" (NI 43-101 clause 2.3 (3)) and is not intended to satisfy the requirements of a "preliminary feasibility study" or "feasibility study" (CIM 2005). Therefore, the mining inventories reported in this assessment are not Mineral Reserves, and there is no certainty that all or part of the reported mining inventories will eventually be converted to Mineral Reserves.

Any reference to "ore" in this report is to be taken as a generic term and is not intended to imply any expectation or likelihood that the material referred to will eventually be demonstrated to be economic.

18 Other relevant data and information

18.1 Geotechnical study

18.1.1 Scope of Study

Snowden's geotechnical consultant, Dr. Walter Keilich, visited the Nixon Fork gold mine, located in Alaska, USA, between the 2nd and the 8th of November 2010 to undertake geotechnical mapping and core logging.

The scope of work for the site visit included the following tasks:

- Geotechnically map underground development drives adjacent to the 3000X, J5A and 3300 orebodies (3000X and 3300 are in the Crystal zone, J5A is in the Southern Cross zone)
- Geotechnically log core obtained from the 3000X, 3000Z, J5A, 3000, 3300, Whalen and Northstar orebodies
- Develop a preliminary geotechnical database
- Provide preliminary recommendations for underground stope design.

18.1.2 Geotechnical recommendations

Snowden provides the recommendations in Table 18-1 (sublevel stoping) and Table 18-2 (shrinkage/cut and fill stoping) based on underground mapping, geotechnical logging, data provided and the results from the underground stability analyses. Snowden understands that the intended mining methods at Nixon Fork are either shrinkage or cut and fill stoping with a likely maximum entry size of 5 m wide by 4 m high. The geotechnical assessment indicates that sublevel open stoping, together with shrinkage or cut and fill stoping are viable mining options at Nixon Fork.

Snowden recommends that for sublevel open stopes, where stope walls require support for stability, cable bolt lengths should extend 7 m – 8 m into the abutment beyond the stope boundary, with a minimum support density of 0.35 cables/m^2. Snowden also recognizes this may prove to be uneconomic for some orebodies; therefore consideration may also be given to reducing stope dimensions or mining by cut and fill stoping to improve stability, especially in the Whalen and Northstar orebodies.

For shrinkage/cut and fill stoping, Snowden is of the opinion that the minimum bolt spacing and length should be 1 m and 1.8 m respectively, with a minimum anchorage capacity of 10 tonnes/m. Assuming that this support regime is followed, Snowden would expect that the dilution for shrinkage/cut and fill stoping would be in the order of 5%.

Table 18-1 Recommended sublevel open stope design parameters

Stope	Hydraulic Radius Hangingwall (m)	Footwall (m)	Back (m)	End (m)	Support
J5A	5.9	3.5	3.1	2.9	Hangingwall requires cable-bolts at density of 0.35 cables/m^2
3000X	4.0	3.5	5.3	3.5	
3000Z	4.5	4.0	2.8	2.6	
3000	2.0	3.5	1.5	1.0	
3300	4.0	6.6	3.1	2.7	Footwall requires cable-bolts at density of 0.35 cables/m^2
Whalen	1.0	1.0	1.0	1.0	
Northstar	1.5	1.5	4.0	2.5	

Table 18-2 Recommended shrinkage/cut and fill stoping support regime

Orebody	Bolt Spacing (m)	Bolt Length (m)
J5A	1.2	1.8
3000X	1.3	1.8
3000Z	1.3	1.8
3000	1.2	1.8
3300	1.3	1.8
Whalen	1.0	1.8
Northstar	1.2	1.8

18.1.3 Underground mapping

Three orebodies were selected for underground mapping:

1. J5A
2. 3000X
3. 3000

Mining priority, accessibility and time constraints were the main factors in the selection process. Figure 18-1 and Figure 18-2 illustrate the underground mapping positions.

The scan-line method of mapping was employed, with the horizontal scan-lines being 6 m, 10 m and 6 m long for the 3000X, J5A and 3000 orebodies respectively. Vertical scan-lines were also mapped and were 1.8 m long for all three orebodies.

Figure 18-1 Underground mapping locations for 3000X and J5A orebodies

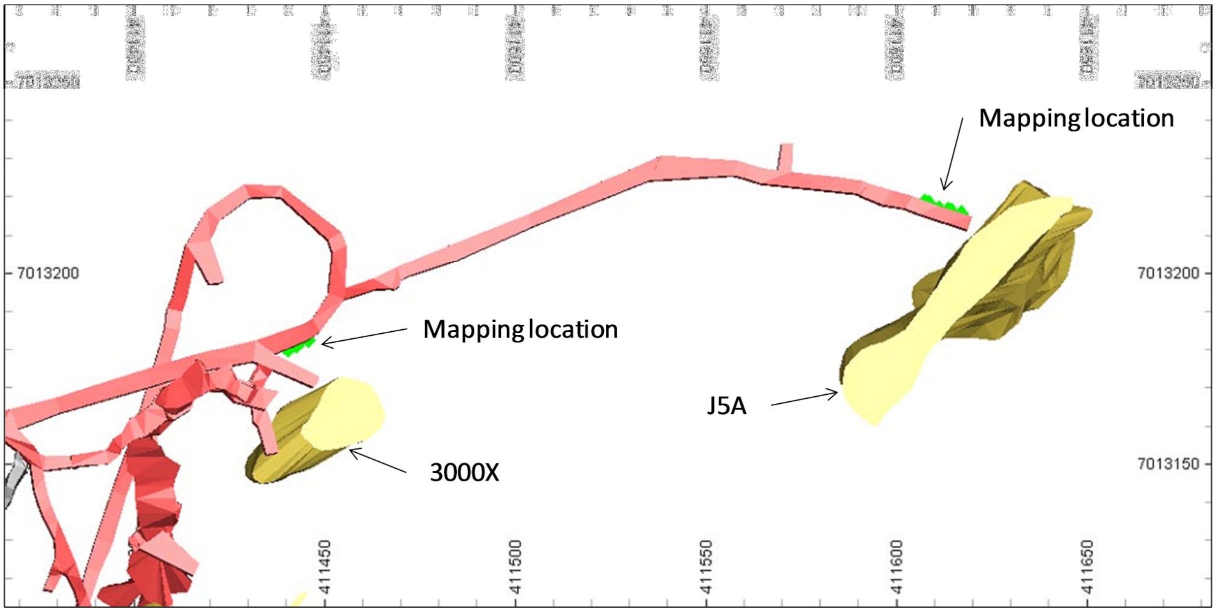
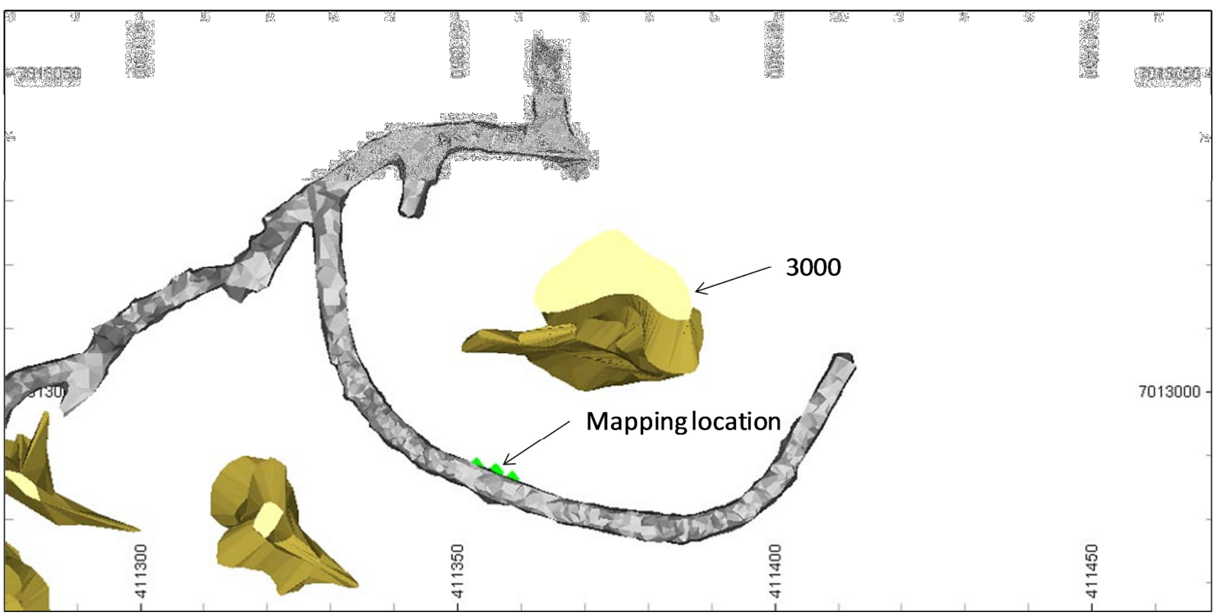


Figure 18-2 Underground mapping location for 3000 orebody



The Dips software package (Rocscience, 2010) was used to assess the mapping data. Joint sets appeared to be consistent across the three orebodies, with three joint sets identified; orientation format is dip / dip direction:

- Joint set 1 – 79° / 182°

- Joint set 2 – 76° / 126°
- Joint set 3 – 81° / 234°

Figure 18-3 illustrates the concentration of the combined mapped data, and Figure 18-4 illustrates the three joint sets with associated dip and dip direction.

Figure 18-3 Concentration of combined mapped data

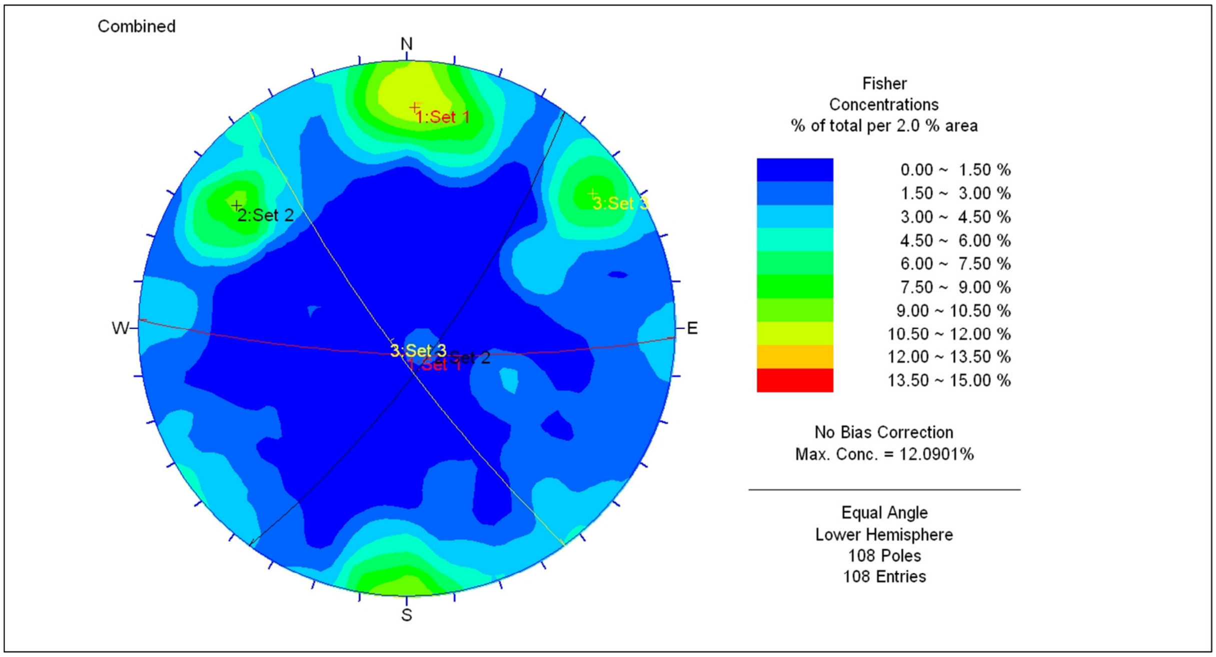
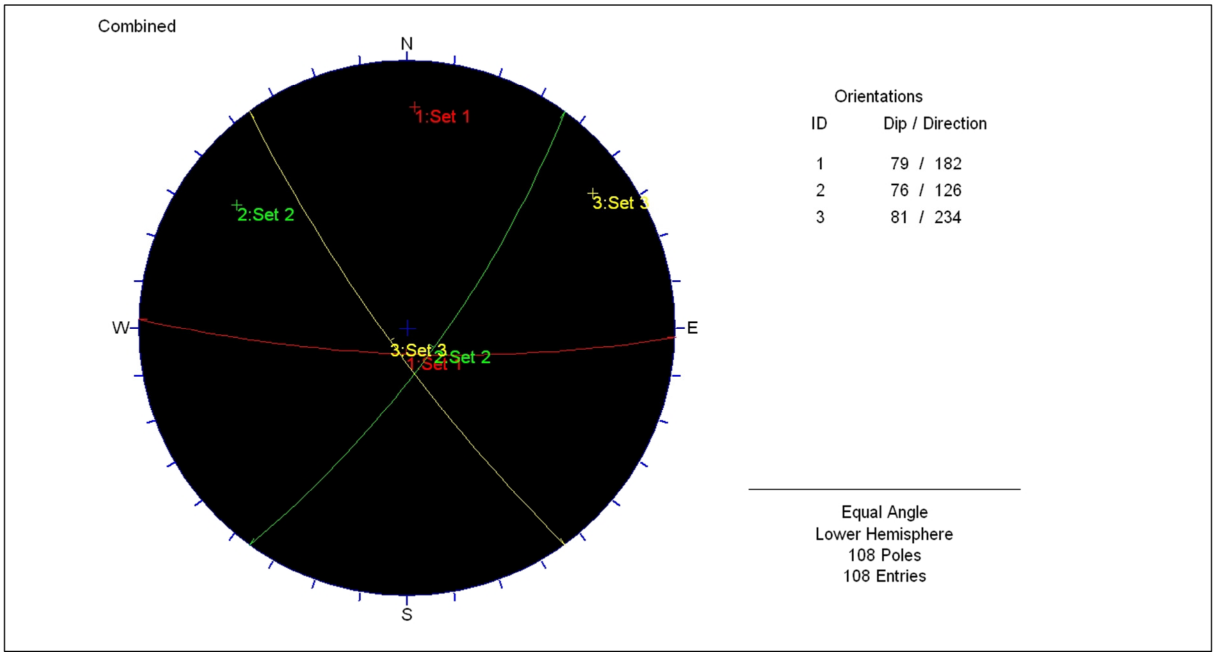


Figure 18-4 Joint sets with dip and dip direction



It is evident from Figure 18-4 that the major joint sets are steep dipping, which presents no major stability issues for the backs of stopes and development drives, but may present wedge/slabbing problems in the hangingwall and footwall of stopes and in the walls of development drives.

For the purpose of this preliminary level analysis, Snowden has assumed that the same joint sets are consistent for the J5A, 3000X, 3000Z, 3000, 3300, Whalen and Northstar orebodies.

18.1.4 Core logging

Table 18-3 contains the drillholes that were geotechnically logged and the orebodies the drillholes intersected.

Table 18-3 Geotechnically logged drillholes

Drillhole	Orebody	Logged Length (m)
N10-001	Whalen	42.2
N10-007	Northstar	34.0
N10U-006	J5A	30.6
N10U-010	3300 Upper	39.6
N10U-023	3000X	45.2
N10U-028	3000Z	32.0
N10U-039	Base of 3300	40.0
N10U-041	Base of 3000	34.2

A set of holes selected for geotechnical logging prior to arrival on site proved to be of little value, as they had been extensively sampled, damaged from mishandling or discarded. Several of the holes in Table 18-3 had also been sampled but the overall integrity was slightly greater than the previously selected holes.

The drillholes were logged for rock strength, rock quality designation (RQD), fracture frequency and fracture characteristics. A 20 m section on both sides of the intersected orebody was logged to capture the characteristics of the hangingwall and footwall. Sampling results were not available at the time of logging, therefore Snowden was not provided with the orebody intersections in all holes. Snowden has estimated the boundaries of the ore zones in each drillhole using the wireframes supplied by FAU.

The collar locations of the drillholes provided to Snowden may not reflect the actual locations, as it is common practice for drillers to relocate the collar; however, the margin of error is expected to be within a few metres and will not influence the outcomes of this assessment.

The orebody wireframes supplied to Snowden do not distinguish between mined voids and solid ore and may not be accurate. The drillholes selected for logging did not intersect mining voids in the area of interest.

All drillholes except for N10U-039 and N10U-041 contained sections that were sampled, making RQD and fracture frequency measurements impossible in these sections. Where this occurred, Snowden utilized RQD logs provided by FAU. For these sections fracture frequency was estimated using a relationship derived from the logging sections that permitted RQD and fracture frequency logging.

Due to the condition of the drillholes and the quality of the data provided, Snowden has adopted a conservative approach for logging RQD and strength.

18.1.5 Geotechnical model

The geotechnical core logs were analyzed in two sections, the first comprising of the initial 5 m interval of the hangingwall and footwall, and the second comprising of the subsequent 15 m interval in the hangingwall and footwall. This was done to investigate if any weak zones were adjacent to the orebodies.

The results are summarized in Table 18-4, Table 18-5, Figure 18-5 and Figure 18-6.

Table 18-4 Rockmass parameters for 0 m – 5 m sections of hangingwall and footwall

BHID	Orebody	Region	RQD (%)	Strength (MPa)
N10-001	Whalen	Hangingwall	79%	50
		Footwall	18%	17
N10-007	Northstar	Hangingwall	49%	38
		Footwall	63%	75
N10U-006	J5A	Hangingwall	76%	70
		Footwall	31%	37
N10U-010	Upper 3300	Hangingwall	81%	63
		Footwall	86%	38
N10U-023	3000X	Hangingwall	57%	61
		Footwall	87%	38
N10U-028	3000Z	Hangingwall	97%	75
		Footwall	33%	38
N10U-039	Base of 3300	Hangingwall	88%	75
		Footwall	76%	75
N10U-041	Base of 3000	Hangingwall	61%	61
		Footwall	41%	15

Table 18-5 Rockmass parameters for 5 m – 20 m sections of hangingwall and footwall

BHID	Orebody	Region	RQD (%)	Strength (MPa)
N10-001	Whalen	Hangingwall	45%	36
		Footwall	70%	31
N10-007	Northstar	Hangingwall	46%	32
		Footwall	29%	75
N10U-006	J5A	Hangingwall	79%	132
		Footwall	20%	38
N10U-010	Upper 3300	Hangingwall	48%	34
		Footwall	31%	22
N10U-023	3000X	Hangingwall	79%	75
		Footwall	62%	31
N10U-028	3000Z	Hangingwall	62%	45
		Footwall	68%	38
N10U-039	Base of 3300	Hangingwall	62%	66
		Footwall	75%	71
N10U-041	Base of 3000	Hangingwall	73%	71
		Footwall	81%	59

Figure 18-5 RQD results for initial 5 m and subsequent 15 m of hangingwall and footwall

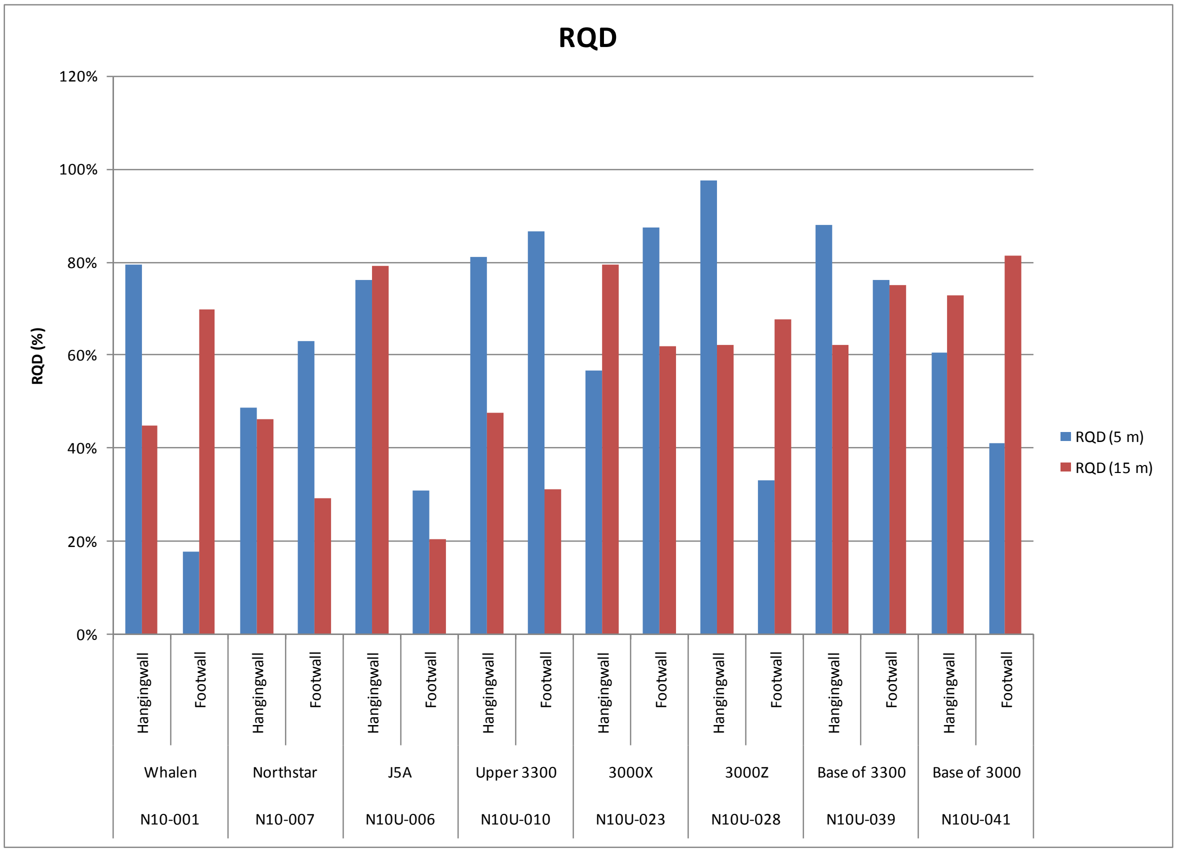
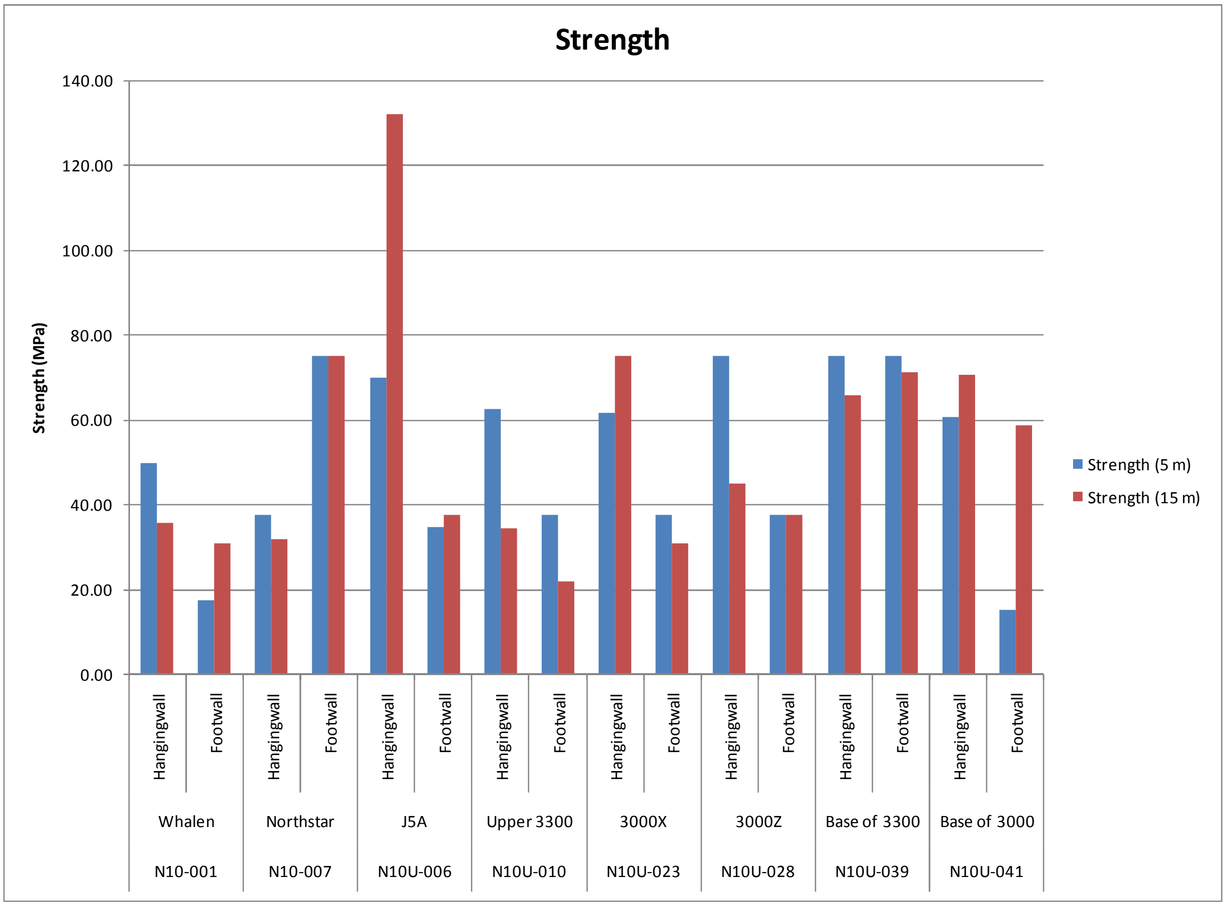


Figure 18-6 **Strength results for initial 5 m and subsequent 15 m of hangingwall and footwall**



It can be seen in Figure 18-5 and Figure 18-6 that in the majority of holes, the hangingwall is comprised of better geotechnical quality material than the footwall. It can also be seen that the initial 5 m of the hangingwall and footwall appears to be more competent than the subsequent 15 m of material.

18.1.6 Stability assessment and conceptual stope design

Sublevel stoping

The stability graph method (Hoek, Kaiser & Bawden, 2000) was used to assess the stability of the conceptual stopes and support density. Stability was assessed on the basis that the stopes are non-man entry and the hangingwall and footwall surfaces will be vertical. In this context, some minor sloughing may occur particularly where blast damage occurs, but overall stability is maintained and dilution is kept within normal, acceptable limits.

To assess stability of sublevel stoping, a series of conceptual stopes were designed for the J5A, 3000X, 3000Z, 3000, 3300, Whalen and Northstar orebodies. The dimensions of the stopes were based on the strike length of the orebody and a sub level interval of 20 m (floor to floor). The hangingwall and footwall of each stope was assumed to be vertical. The details of these stopes are summarized in Table 18-6.

Table 18-6 Conceptual stope details

Orebody	Stope Back Elevation (m)	Length (m)	Width (m)	Height (m)
J5A	385	29.0	8.0	20
3000X	365	29.0	16.5	20
3000Z	335	26.5	7.0	20
3000	137	18.0	5.5	20
3300	130	38.0	7.5	20
Whalen	410	74.5	15.0	20
Northstar	430	74.5	19.0	20

Equation 1 is the relationship used to calculate the hydraulic radius (HR) for each stope face.

Equation 1

$$HR = \frac{width \times height}{2(width) + 2(height)}$$

Note: for horizontal faces (stope backs), substitute length for height in Equation 1.

No information on the in-situ stress regime was provided to Snowden. The horizontal stress to vertical stress ratio (K) was assumed to be 1.5, based on data from the World

Stress Map (Heidbach et al., 2008) that shows the stress measurements in the general vicinity of Nixon Fork indicate the area has a strike-slip stress regime in which the magnitude of vertical stress lies between the major and minor horizontal stress magnitudes.

Two cases were assessed for each orebody:

1. all stope walls were assumed to consist of the first 5 m of hangingwall and footwall material as defined by logging

2. all stope walls were assumed to consist of the subsequent 15 m of hangingwall and footwall material as defined by logging.

The lowest stability number was selected to represent a worst case scenario.

Stopes for the upper 3300 orebody were not assessed due to uncertainty of stope location.

Stopes for the Mystery ore zones were not assessed due to lack of suitable core.

Dilution has been quantified using the equivalent linear overbreak/slough (ELOS) method proposed by Clark and Pakalnis (1997). This method utilizes empirical observations of stability number, N' and HR to estimate the horizontal distance beyond the designed stope boundary that will slough.

Dilution is defined as the waste that is not segregated from the ore during mining (Noble, 1992). Dilution serves to decrease the grade of the ore and increase the mined tonnage, thereby increasing mining costs.

According to Stewart and Trueman (2008), empirical evidence suggests that reducing the stope span will only reduce dilution if the causes of dilution are geotechnical. If dilution is independent of stope span, then dilution is unlikely to be caused by geotechnical factors. Stress damage is the only geotechnical cause of dilution that would not be span dependant.

Stability results are summarized in Table 18-7 and Figure 18-7 to Figure 18-13.

Table 18-7 Results of Stability Assessments

Stope	Wall	HR	N'	Stability	Comments
J5A	HW	5.92	2.86	Stable with support	Support required - conservative
	FW	5.92	1.90	Stable with support	Re-design
	Back	3.14	6.10	Stable	No support required
	End	2.86	1.90	Stable	No support required
3000X	HW	5.92	5.81	Unsupported transition zone	No support required, ELOS = 0.5 m – 1.0 m
	FW	5.92	3.58	Unsupported transition zone	No support required, ELOS = 1.0 m – 2.0 m
	Back	5.26	10.46	Stable	No support required
	End	4.52	3.25	Unsupported transition zone	No support required, ELOS = 1.0 m – 2.0 m
3000Z	HW	5.70	6.55	Unsupported transition zone	No support required, ELOS = 0.5 m – 1.0 m
	FW	5.70	3.08	Unsupported transition zone	No support required, ELOS = 1.0 m – 2.0 m
	Back	2.77	7.91	Stable	No support required
	End	2.59	2.62	Stable	No support required
3000	HW	4.74	3.52	Unsupported transition zone	No support required, ELOS = 1.0 m – 2.0 m
	FW	4.74	0.41	Stable with support	Re-design
	Back	2.11	0.15	Unsupported transition zone	No support required, ELOS = > 2.0 m
	End	2.16	0.05	Unsupported transition zone	No support required, ELOS = > 2.0 m
3300	HW	6.55	4.38	Stable with support	Re-design
	FW	6.55	4.00	Stable with support	Support required - conservative
	Back	3.13	3.47	Stable	No support required
	End	2.73	1.49	Stable	No support required
Whalen	HW	7.88	0.11	Caved zone	Re-design
	FW	7.88	0.01	Caved zone	Re-design
	Back	6.24	0.11	Stable with support	Re-design
	End	4.29	0.06	Stable with support	Re-design
Northstar	HW	7.88	0.22	Caved zone	Re-design
	FW	7.88	0.18	Caved zone	Re-design
	Back	7.57	5.80	Stable with support	Re-design
	End	4.87	1.80	Unsupported transition zone	Re-design

Figure 18-7 Stability graph for J5A

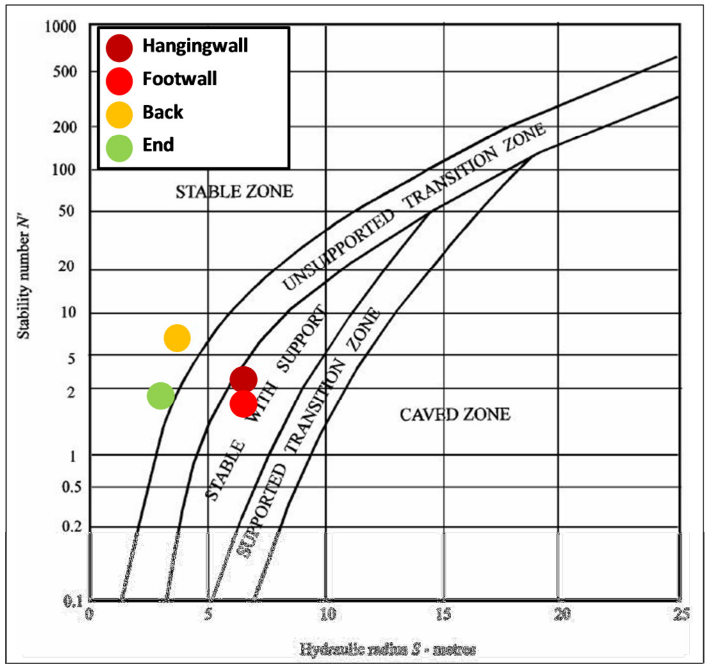


Figure 18-8 Stability graph for 3000X

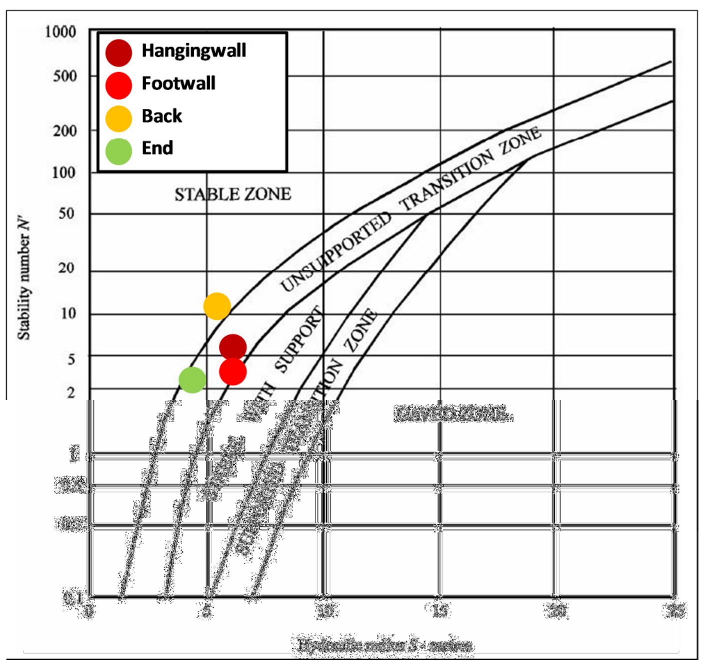


Figure 18-9 Stability graph for 3000Z

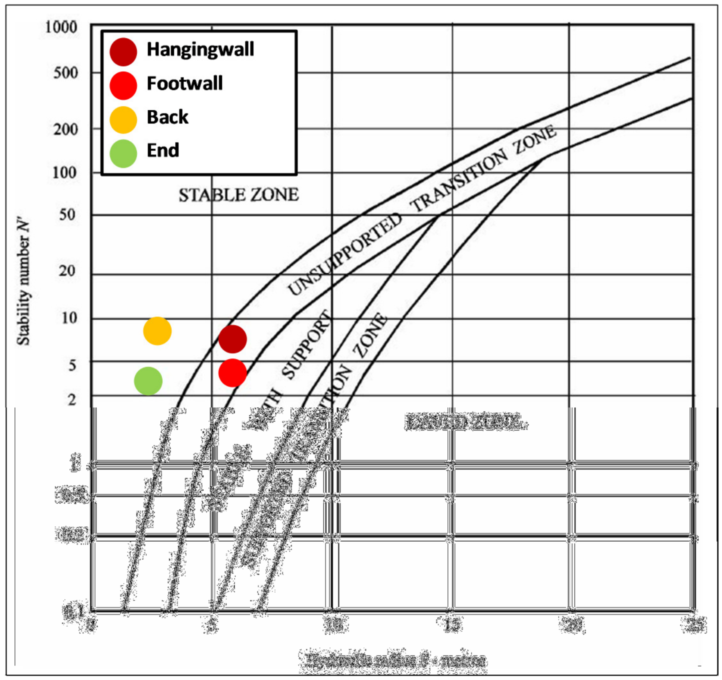


Figure 18-10 Stability graph for 3000

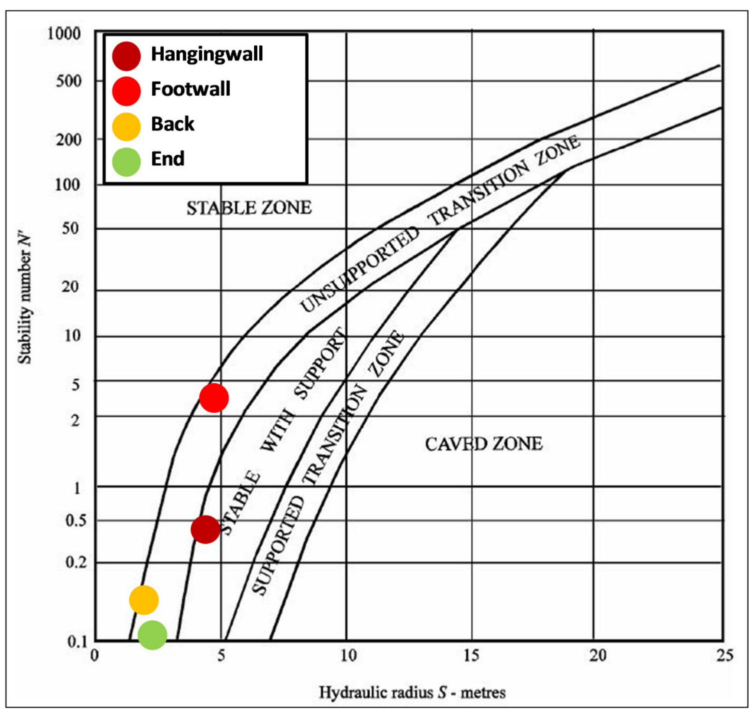


Figure 18-11 Stability graph for 3300

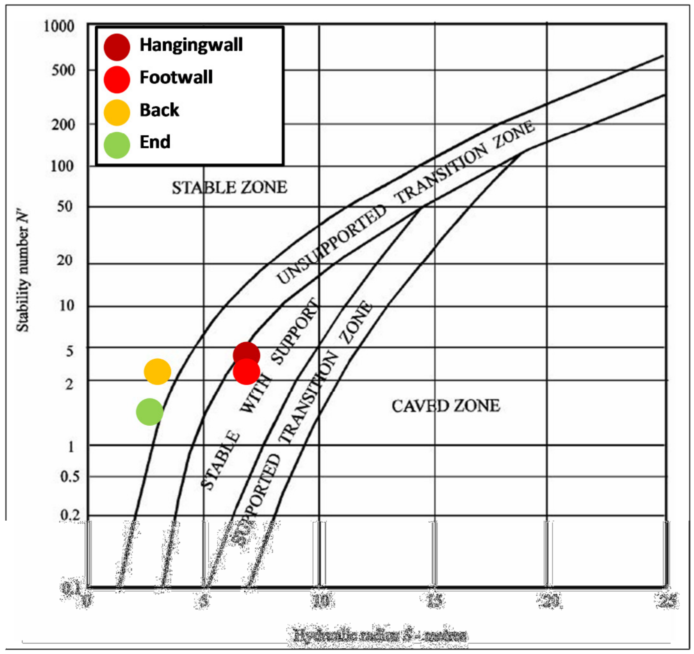


Figure 18-12 Stability graph for Whalen

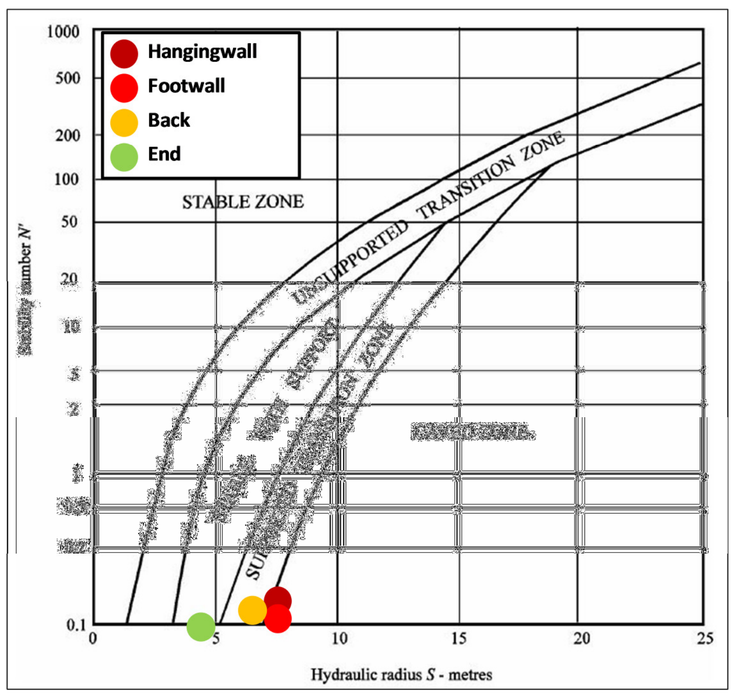


Figure 18-13 Stability graph for Northstar

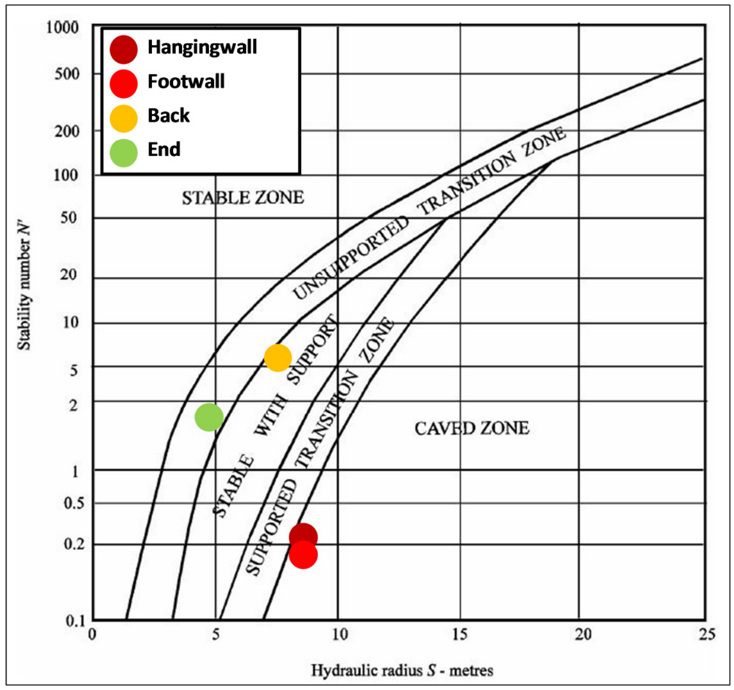


Figure 18-7 shows that the J5A stope back and end walls are stable without support. Support is required to stabilize the hangingwall and footwall. The hangingwall should be stabilized with a support density of 0.35 cables/m² but the footwall cannot be stabilized due to the poor geotechnical conditions indicated by the low RQD. Re-designing the footwall to a HR of 3.5 should be sufficient for stability with minimal dilution.

Figure 18-8 shows that the 3000X stope should be stable without support, although dilution may occur from the hangingwall, footwall and end walls. To minimize dilution, the hangingwall, footwall and small dimension walls should be re-designed to a HR of 4.0, 3.5 and 3.5 respectively.

Figure 18-9 shows that the 3000Z stope should be stable without support, although dilution may occur in the hangingwall and footwall. To minimize dilution, the hangingwall and footwall should be re-designed to a HR of 4.5 and 4.0 respectively.

Figure 18-10 shows that the 3000 stope footwall, back and end walls should be stable without support although dilution may occur. Re-designing the stope back to a HR of 1.5 and the end walls to a HR of 1.0 should be sufficient for stability and minimize dilution. The hangingwall requires support to be stable but cannot be stabilized due to low RQD. Re-designing the hangingwall to a HR of 2.0 and the footwall to a HR of 3.5 should be sufficient for stability with minimal dilution.

Figure 18-11 shows that the 3300 stope back and end walls should be stable without support. Support is required to stabilize the hangingwall and footwall. The footwall should be stabilized with a support density of 0.35 cables/m² but the hangingwall cannot be stabilized due to low RQD. Re-designing the hangingwall to a HR of 4.0 should be sufficient for stability with minimal dilution.

Figure 18-12 shows that the Whalen stope should to be completely re-designed due to very poor geotechnical conditions. The stability graph indicates that a HR of 1.0 for all stope walls would be required to achieve stability without support.

Figure 18-13 shows that the Northstar stope should be completely re-designed due to very poor geotechnical conditions. Re-designing the hangingwall and footwall to a HR of 1.5 would be required to achieve stability without support. The stope back cannot be stabilized with support so a re-design to a HR of 4.0 should achieve stability and minimize dilution. The end walls are stable without support but dilution is expected. To minimize dilution, the HR of the small dimension walls should be reduced to 2.5.

Shrinkage/cut and fill stoping

The NGI Q system (Barton, Lien & Lunde, 1974) and the updated design chart (Grimstad & Barton, 1993) were used to assess support requirements for a man-entry stope with a span of 5 m in a shrinkage/cut and fill stoping scenario. An excavation support ratio (ESR) of 4 (temporary mine openings) was used in the assessment for all stoping areas.

As noted in Section 18.1.4, the precise configuration of the final mining shape will coincide with the highest mineralization, which will be established through the course of mining. The exact ground that will be exposed as the final hangingwall and footwall of the stope are therefore not pre-determined.

The support design results for shrinkage/cut and fill stoping are presented in Table 18-8.

Table 18-8 Support regime for shrinkage/cut and fill stoping

Orebody	Average Min Q	Bolt Spacing (m)	Bolt Length (m)
J5A	1.7	1.2	1.8
3000X	3.1	1.3	1.8
3000Z	3.4	1.3	1.8
3000	1.7	1.2	1.8
3300	3.0	1.3	1.8
Whalen	0.4	1.0	1.8
Northstar	1.4	1.2	1.8

18.1.7 Geotechnical - detailed findings and recommendations

The HR and support recommendations for sublevel stoping are presented in Table 18-9.

The support regime recommendations for shrinkage/cut and fill stoping are presented in Table 18-10.

Table 18-9 Recommended sublevel open stope design parameters

Stope	Hydraulic Radius Hangingwall (m)	Footwall (m)	Back (m)	End (m)	Support
J5A	5.9	3.5	3.1	2.9	Hangingwall requires cable-bolts at density of 0.35 cables/m²
3000X	4.0	3.5	5.3	3.5	
3000Z	4.5	4.0	2.8	2.6	
3000	2.0	3.5	1.5	1.0	
3300	4.0	6.6	3.1	2.7	Footwall requires cable-bolts at density of 0.35 cables/m²
Whalen	1.0	1.0	1.0	1.0	
Northstar	1.5	1.5	4.0	2.5	

The following guidelines to minimize dilution are provided by Stewart and Trueman (2008):

Minimizing geotechnical causes of dilution

- *More detailed structural analysis using scanline mapping and stereonets*
- *Assess relaxation potential*
- *Cost benefit analysis for cablebolting and/or stope span reduction including fill cycle times*
- *Ore drive development under good geological control including appropriate drive profile*
- *Stress damage related dilution can be minimized by evaluation of extraction sequence against damage criterion, increasing the number of rings fired per blast and where practical avoiding shrinking central pillar extraction sequence*

Minimizing drill and blasting related dilution

- *Selected appropriate blast pattern*
- *Survey drill holes and analyze results in a systematic manner*
- *Drill and blast trials should be randomized trials*
- *Blast damage minimization*
- *Smooth-wall blasting*

Table 18-10 Recommended shrinkage/cut and fill stoping support regime

Orebody	Bolt Spacing (m)	Bolt Length (m)
J5A	1.2	1.8
3000X	1.3	1.8
3000Z	1.3	1.8
3000	1.2	1.8
3300	1.3	1.8
Whalen	1.0	1.8
Northstar	1.2	1.8

18.2 Underground mining inventory

18.2.1 Site visit

On the 3rd and 4th of August 2010, Mr. Anthony Finch, responsible for the mine engineering component of the study, visited the Nixon Fork site. During the visit Mr. Finch inspected the entire site, including but not limited to the camp, the offices, the workshops, the mobile mining equipment, the tailings and mineral processing facilities, access and site roads, infrastructure locations, surface expressions of the mineralisation, existing underground workings in both the Crystal and Mystery, drill core from various locations on the site to view properties of both mineralisation and host rocks, and the air strip associated.

18.2.2 Mining assessment

A study to assess the economic potential of underground mining at Nixon Fork was conducted using Snowden's in house consulting tools Stopesizor and Evaluator. The underground study looked at all the modelled zones in conjunction with the existing workings.

Stopesizor is a software tool designed to identify mining inventories based on minimum mining geometries and a selected range of cut-off grades. In this way, it is possible to incorporate planned dilution where necessary and exclude outlying and isolated resource blocks from the inventory to give a realistic indication of the potentially extractable inventory.

Evaluator is a software tool used to incorporate prices, costs, capacities and scheduling constraints into an analysis in order to be able to consistently and rapidly compare the economic potential of differing inventories and strategies developed with Stopesizor.

By using these tools, a wide range of scenarios can be rapidly considered, enabling potentially value adding opportunities to be identified and reducing the likelihood of progressing a sub-optimal strategy to the design stage.

Stopesizor software modifies a geological block model to identify the optimum extraction outline for a range of cut-off values (usually grade). This is done by constructing a model comprising selective mining blocks (SMB), where the SMB represents a user defined minimum practical geometry. Each SMB comprises a contiguous group of resource blocks that honours the user defined minimum dimensions, bearing and dip constraints for each axis.

At Nixon Fork two candidate mining methods were identified; long hole open stoping (LHOS) and cut and fill (CAF) (or shrinkage – both of which have the same SMB). Inventories were determined in Stopesizor for each method and subjected to an economic analysis to select the preferred method.

The original resource models provided had a block dimension of 2.0mX by 2.0mY by 2.0mZ. Each block also contained a percentage field which reflected the percentage of the block that is within the geologic wireframe. Because Stopesizor does not tolerate partial blocks (percentages) the model was subdivided to a block size of 1.0mX by 1.0mY by 1.0mZ, after which any blocks outside the geological wireframes were deleted. This process was undertaken to reflect the potential selectivity of cut and fill (CAF) mining and to reduce the dilution that was inherent in the supplied resource model (if the percentages were averaged over the entire block). The resulting block model was then reconciled to the original. The reconciliation report is provided in Table 18-11 below.

Table 18-11 Model reblocking reconciliation report

DOMAIN	CUT-OFF	Stopesizor 1x1x1 model				Resource report						comparison (stopesizor/resource report)			
		INDICATED		INFFERRED		INDICATED			INFFERRED			INDICATED		INFFERRED	
		TONNES	AUMET (g)	TONNES	AUMET (g)	TONNES	GRADE (g/t)	AUMET (g)	TONNES	GRADE (g/t)	AUMET (g)	TONNES	AUMET (g)	TONNES	AUMET (g)
	1	28,287	99,181	3,901	10,580	28,100	3.51	98,631	3,900	2.71	10,569	101%	101%	100%	100%
	5	5,312	44,331	33	172	5,300	8.34	44,202	40	5.25	210	100%	100%	82%	82%
3500	10	1,235	14,468			1200	11.71	14,052			-	103%	103%		
	1	179,094	2,396,932	36,936	710,260	175,900	13.52	2,378,168	36,600	19.27	705,282	102%	101%	101%	101%
	5	113,743	2,218,575	30,462	689,170	112,100	19.66	2,203,886	30,300	22.60	684,780	101%	101%	101%	101%
3300	10	69,164	1,895,581	21,072	622,449	68,900	27.45	1,891,305	20,900	29.53	617,177	100%	100%	101%	101%
	1	62,592	191,386	5,085	12,886	62,800	3.05	191,540	5,100	2.55	13,005	100%	100%	100%	99%
WHA	5	8,906	62,645	156	992	8,900	7.02	62,478	170	6.36	1,081	100%	100%	92%	92%
LEN	10	635	7,115	3	26	630	11.20	7,056			-	101%	101%		
	1	27,279	654,135	85,456	1,402,787	26,600	23.61	628,026	99,000	14.18	1,403,820	103%	104%	86%	100%
	5	20,694	636,132	62,277	1,334,415	20,000	30.44	608,800	63,000	20.61	1,298,430	103%	104%	99%	103%
3000	10	16,010	601,589	38,727	1,162,926	15,500	37.28	577,840	37,600	29.69	1,116,344	103%	104%	103%	104%
	1			61,060	135,298			-	60,600	2.23	135,138			101%	100%
	5			1,958	11,688			-	2,000	5.96	11,920			98%	98%
NS	10							-			-				
	1	69,893	319,011	35,099	81,788	69,900	4.56	318,744	35,200	2.33	82,016	100%	100%	100%	100%
	5	17,957	198,886	2,511	21,885	18,000	11.07	199,260	2,500	8.71	21,775	100%	100%	100%	101%
J5	10	7,553	125,830	661	8,978	7,500	16.67	125,025	660	13.58	8,963	101%	101%	100%	100%
	1			28,924	307,984			-	28,900	10.67	308,363			100%	100%
	5			19,788	285,692			-	19,800	14.46	286,308			100%	100%
SC	10			11,024	216,337			-	11,100	19.64	218,004			99%	99%
	1	36,244	127,640	59,715	142,398	36,200	3.52	127,424	59,800	2.38	142,324	100%	100%	100%	100%
	5	7,665	54,503	4,891	34,050	7,700	7.11	54,747	4,900	6.97	34,153	100%	100%	100%	100%
3100	10	555	6,291	403	4,996	560	11.34	6,350	410	12.38	5,076	99%	99%	98%	98%
	1	174,872	1,151,628	2,307	7,542	175,100	6.58	1,152,158	2,300	3.33	7,659	100%	100%	100%	98%
MYS	5	57,849	863,953	348	3,214	57,800	14.94	863,532	360	9.27	3,337	100%	100%	97%	96%
TERY	10	27,412	649,632	92	1,755	27,400	23.70	649,380	100	18.85	1,885	100%	100%	92%	93%
	1	141	426	16,468	60,481			-			-				
3200	5			3,251	23,391			-			-				
	10			353	4,831			-			-				

The model was then loaded into Stopesizor wherein various inventories were developed for each of the SMB's.

The SMB dimensions used are shown in Table 18-12 below.

Table 18-12 Stopesizor SMB dimensions

Axis	Units	Cut and fill or Shrinkage	LHOS
1 (E)	m	4	4
2 (N)	m	4	4
3 (Z)	m	5	15

Stopesizor allows for a rapid evaluation of various mining methods on the overall mining inventory, and, by applying a notional cost to each mining method, those inventories can be ranked in terms of value delivered to the owner (rather than tonnage or dilution).

The first part of this process is an examination of the inventors afforded by each mining method.

The resource model provided to Snowden was in 4 parts. The models and their respective parts are shown in Table 18-13, below.

Table 18-13 Model - zone mapping

Model number	Zones contained within
Model 1	Northstar (NS), Whalen (WH)
Model 2	3000D, 3000M, 3000X, 3000Z, 3077, 3200, 3300_300, 3300_383, 3500N, 3510
Model 3	3100, J5, Southern Cross (SC)
Model 4	M1, M3, M4, M5, M6, M7, MYSTS, MYSTS2

Stopesizor stopes for LHOS and CAF for a COG of 15 g/t are shown in the subsequent figures (Figure 18-14 to Figure 18-18).

Figure 18-14 Northstar (left) and Whalen (right) CAF stopes above COG 15 g/t

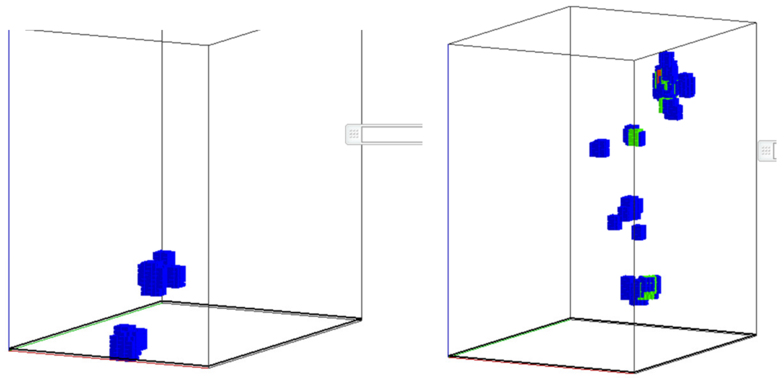


Figure 18-15 Whalen LHOS stopes above COG 15/gt - Northstar returned no stopes.

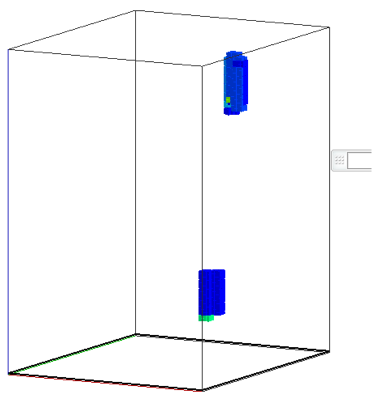


Figure 18-16 Model 2 - Crystal stopes above COG 15g/t - CAF on left, LHOS on right.

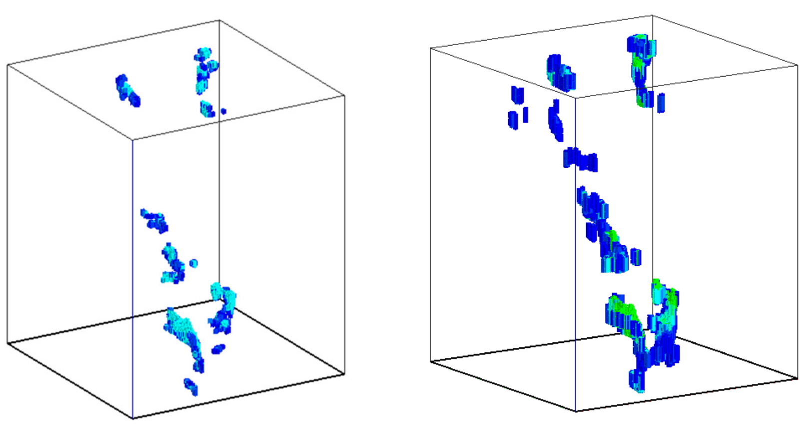


Figure 18-17 Model 3 - Southern Cross stopes above COG 15g/t - CAF on left, LHOS on right

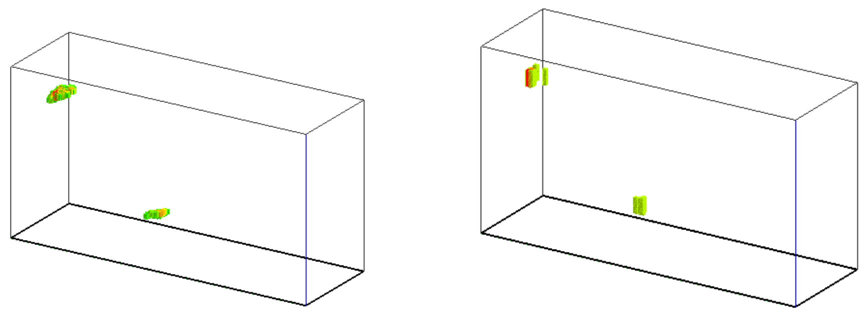
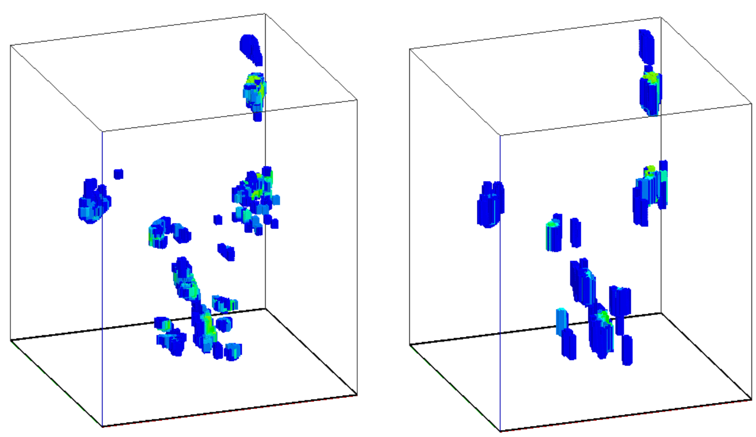
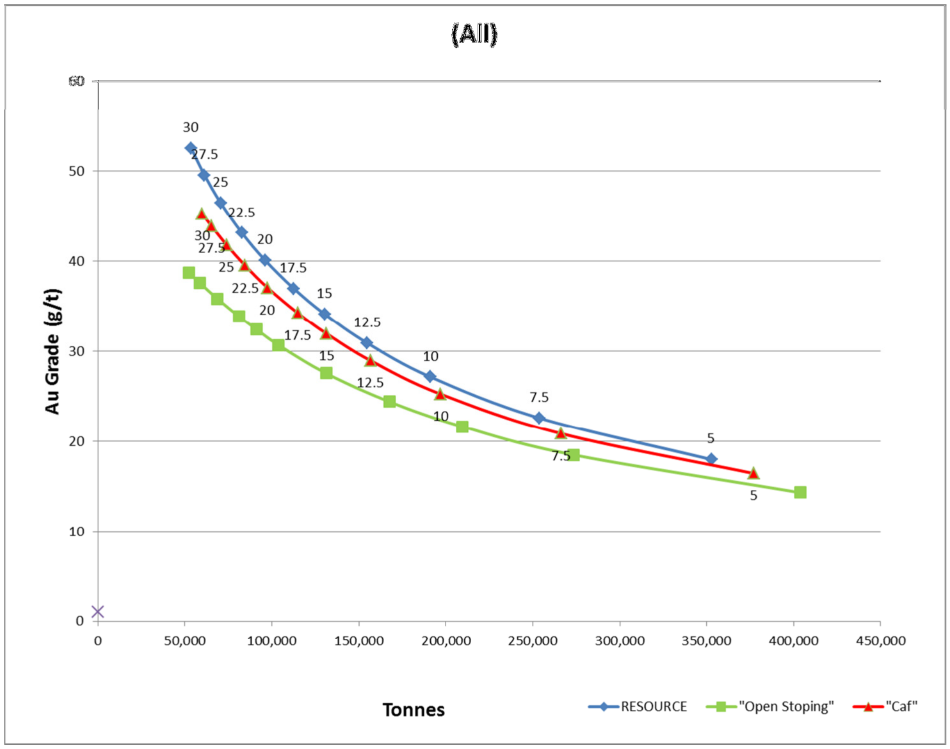


Figure 18-18 Model 4 - Mystery stopes above COG 15g/t - CAF on left, LHOS on right



The inventories generated in Stopesizor can be quantitatively examined using grade tonnage curves. Curves for the Resource, CAF, and LHOS are presented in Figure 18-19 for all zones combined.

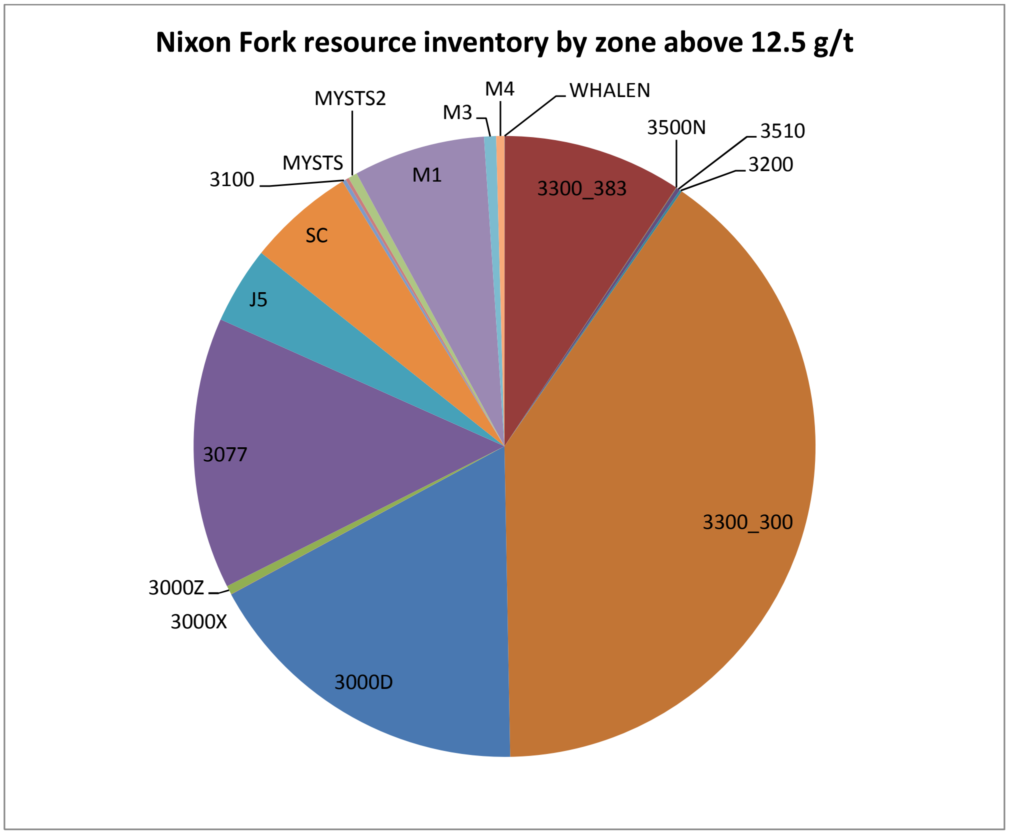
Figure 18-19 Grade tonnage curve for all zones



It can be seen in Figure 18-19 that there is a drop in grade when one moves from the resource to the CAF inventory to the LHOS inventory. For example at a COG of 15 g/t the resource has an inventory of about 133,000 t at an average grade of 34 g/t whereas the inventory for CAF for the same COG is135,000t with an average grade of 32 g/t, and for LHOS at that COG (15g/t) the inventory is 136,000t at an average grade of 27 g/t. This is indicating that on the whole the deposit is very sensitive to mining selectivity and that significant dilution can be incurred using less selective mining techniques.

Often in a multi-zonal deposit like Nixon Fork, a single dominant zone may be driving the behaviour of the grade tonnage curve. Snowden investigated this by examining all the zones individually. The distribution of the resource above a COG of 12.5g/t by zone is shown in Figure 18-20 below.

Figure 18-20 Distribution of the resource above 12.5g/t by zone.



From Figure 18-20 it can be seen that most of the resource is in the 3300_300, 3000D, 3077, 3300_383, J5, M1 and SC zones. Grade tonnage curves are presented for each of these zones in the subsequent figures.

Figure 18-21 Zone 3300-300 grade tonnage curves

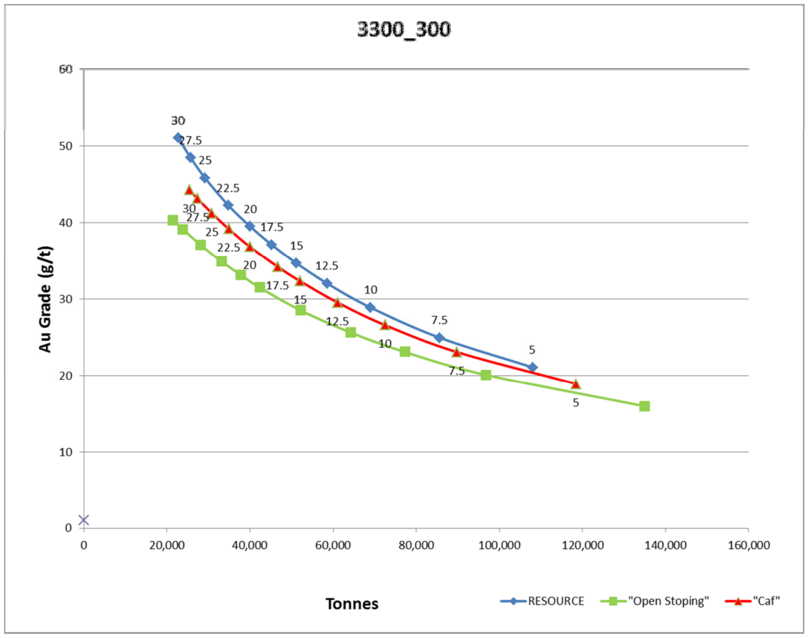


Figure 18-22 Zone 3000D grade tonnage curves.

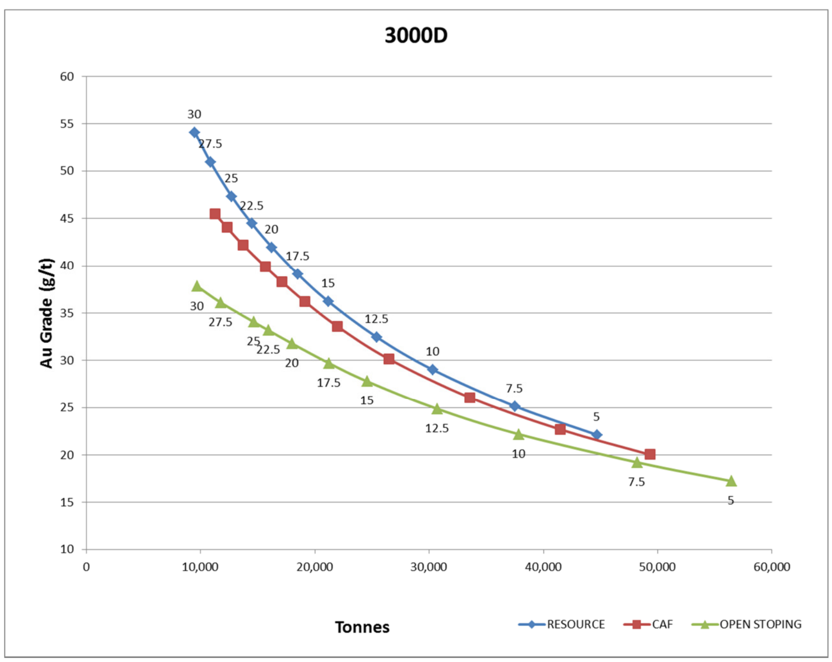


Figure 18-23 Zone 3077 grade tonnage curves.

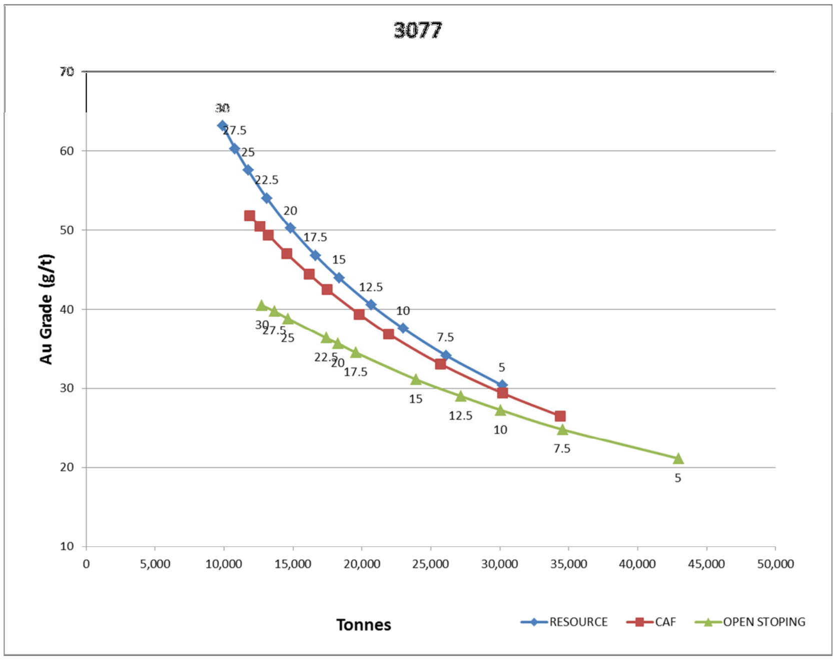


Figure 18-24 Zone 3300_383 grade tonnage curves.

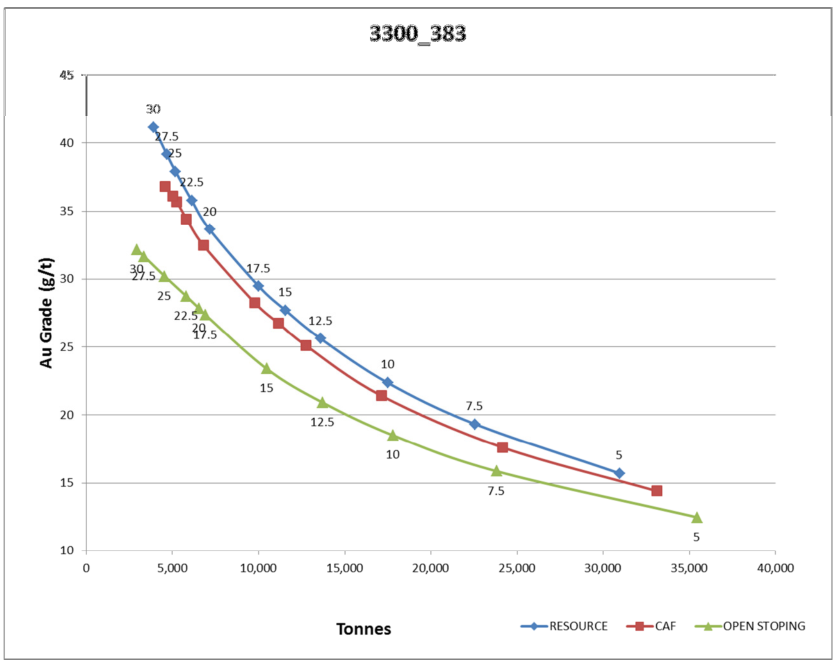


Figure 18-25 Zone J5 grade tonnage curves.

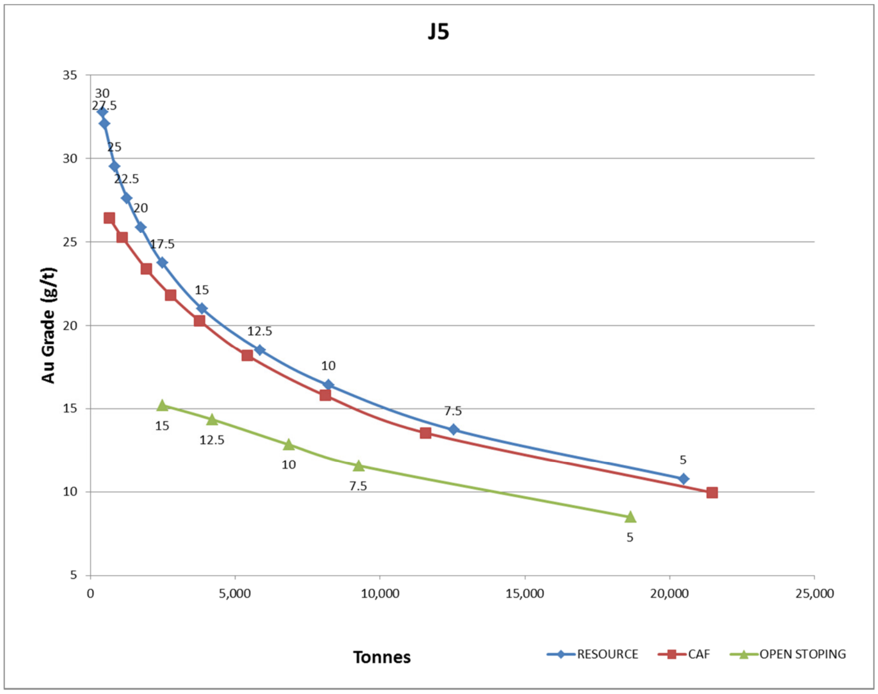


Figure 18-26 Zone M1 grade tonnage curves.

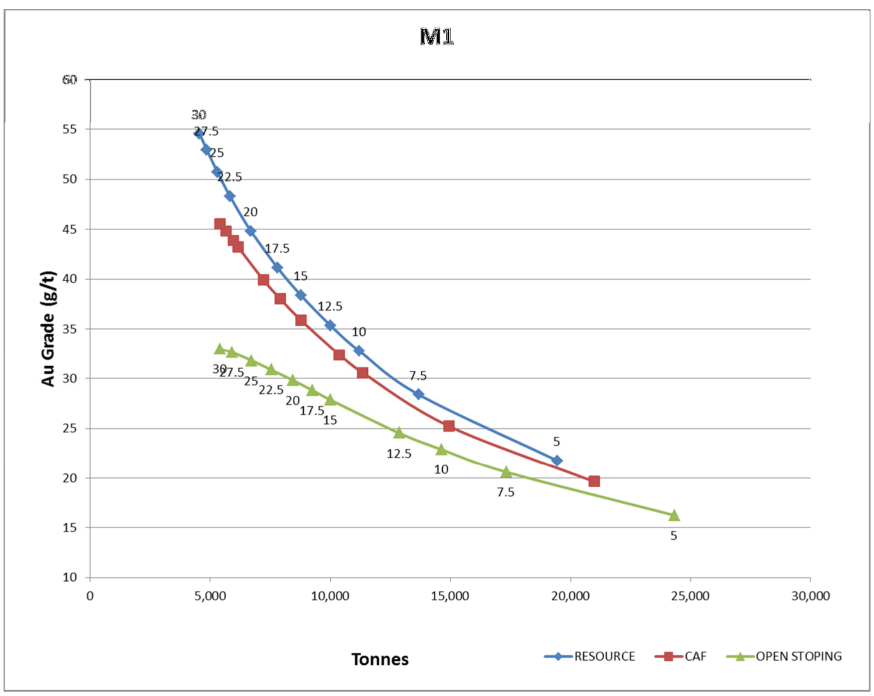


Figure 18-27 Zone M6 grade tonnage curves.

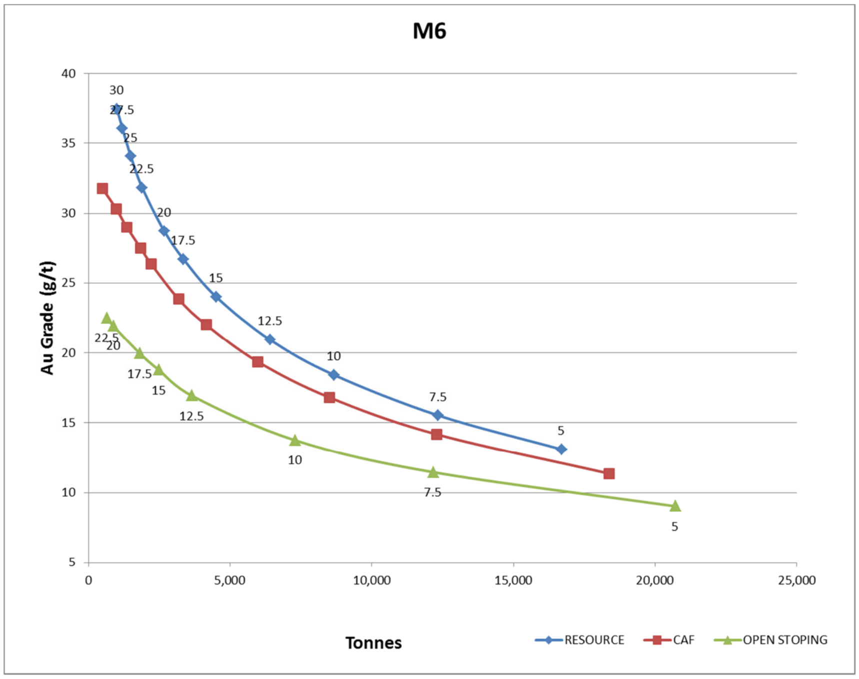


Figure 18-28 Zone M3 + M5 grade tonnage curves

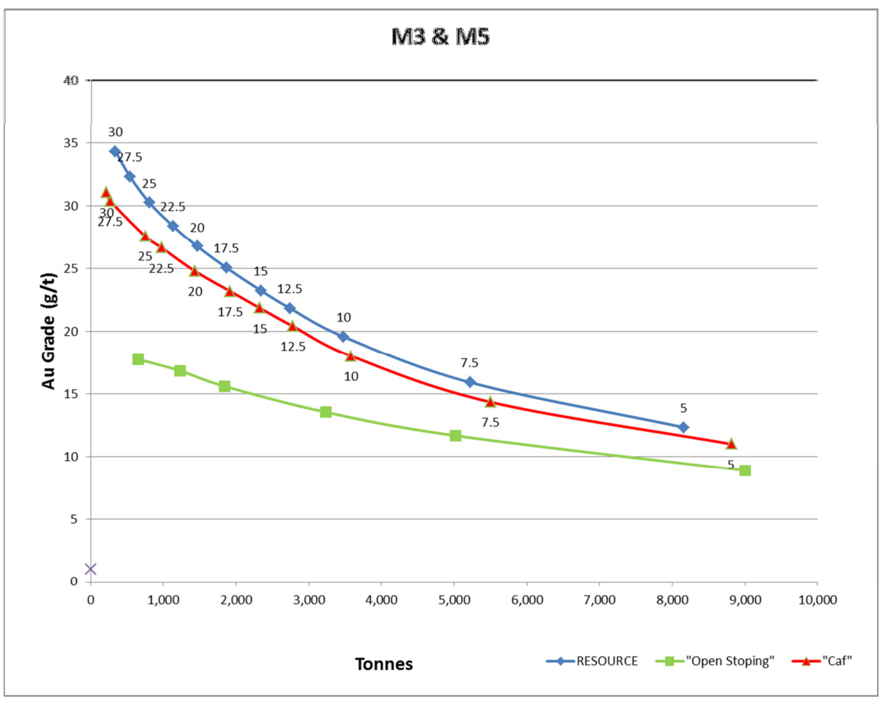
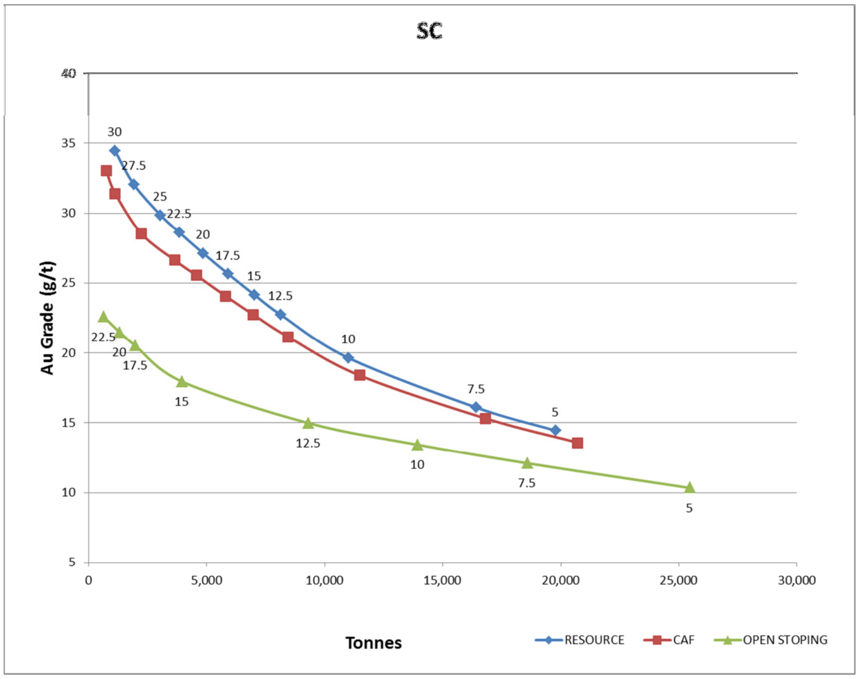


Figure 18-29 Zone SC grade tonnage curves.



All the curves in Figure 18-21 through Figure 18-29 show similar behaviour to Figure 18-19 in that there is significant dilution encountered when long hole stoping is used compared to cut and fill mining. This is more pronounced in the smaller zones but still of concern in the larger zones. In some cases the loss of metal (dilution) can be offset by the reduction in costs encountered in when using less selective mining methods. In the case of Nixon Fork it is anticipated that LHOS will be lower cost than CAF or shrinkage mining.

By applying mining cost for each method to the material in the zone, and then using diluted tonnes and grade for each mining method to calculate the revenue that could be derived Snowden was able to calculate for each zone the "Net Value" for each mining method for each COG by zone. The "Net Value" does not include the cost of development, the cost of capital, and does not include the effects of the time value of money, however, it does provide a way or ranking mining methods in the context of the value they provide to the owner vs the cost of them.

Snowden derived some costs for Nixon Fork by examining budgets proposed by FAU, and making minor modifications to them as was deemed appropriate.

The assumptions used for the "Net Value" calculation is provided in Table 18-14 below.

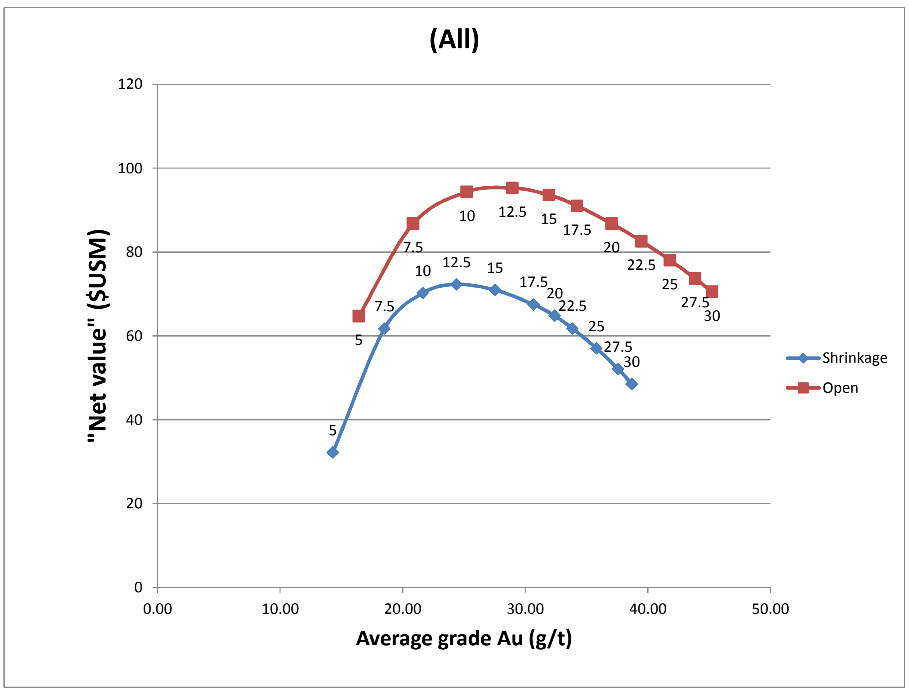
Table 18-14 Assumptions for net value calulations.

Assumption	Amount	Unit
Process feed rate	150	Tonnes per day
G&A costs	190	$US/tonne
Process costs	120	$US/tonne
Unplanned mining losses	5	%
Unplanned dilution	5	%
Process Au recovery	95	%
AU price	1200	$US/Oz
Open stoping mining cost	90	$US/tonne
*Shrinkage mining cost	108	$US/tonne

* *shrinkage and CAF costs were calculated, shrinkage costs were used in this evaluation as they are lowest, and deliver the same selectivity as CAF.*

By applying the above assumption to the grade tonnage curves derived from Stopesizor Snowden was able to derive the "Net Value" chart show in below.

Figure 18-30 "Net Value" for all zones for two mining methods



From Figure 18-30 it is clear that the selectivity afforded by shrinkage stoping returns much higher value than that of open stoping, the value differential being in the order of 20%. Figure 18-30 also shows that for both cases peak value is attained at a COG of around 12.5 g/t.

Looking at the zones individually shows similar results, these individual results are presented in the subsequent figures (Figure 18-31, Figure 18-32, Figure 18-33, Figure 18-34, Figure 18-35, Figure 18-36, Figure 18-37, and Figure 18-39).

Figure 18-31 "Net Value" zone 3330_300.

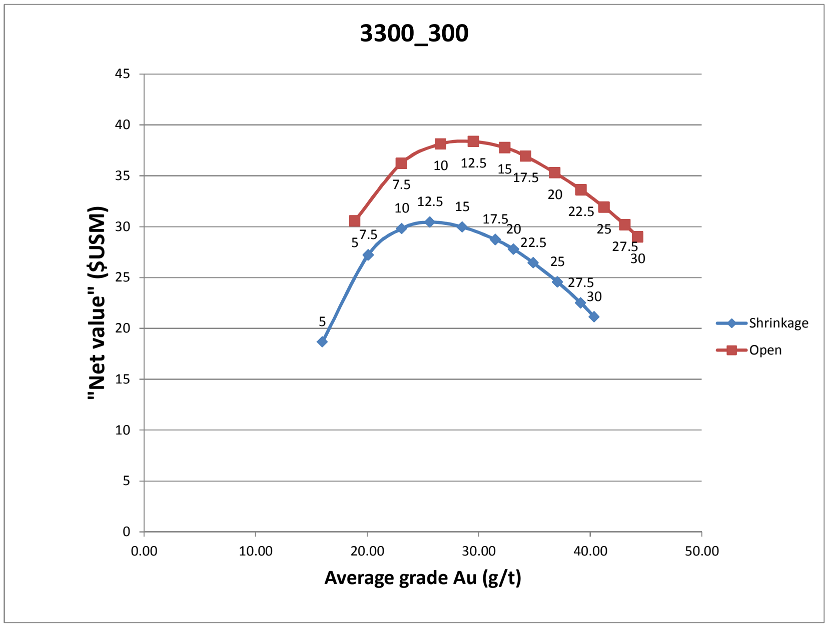


Figure 18-32 "Net Value" zone 3000D.

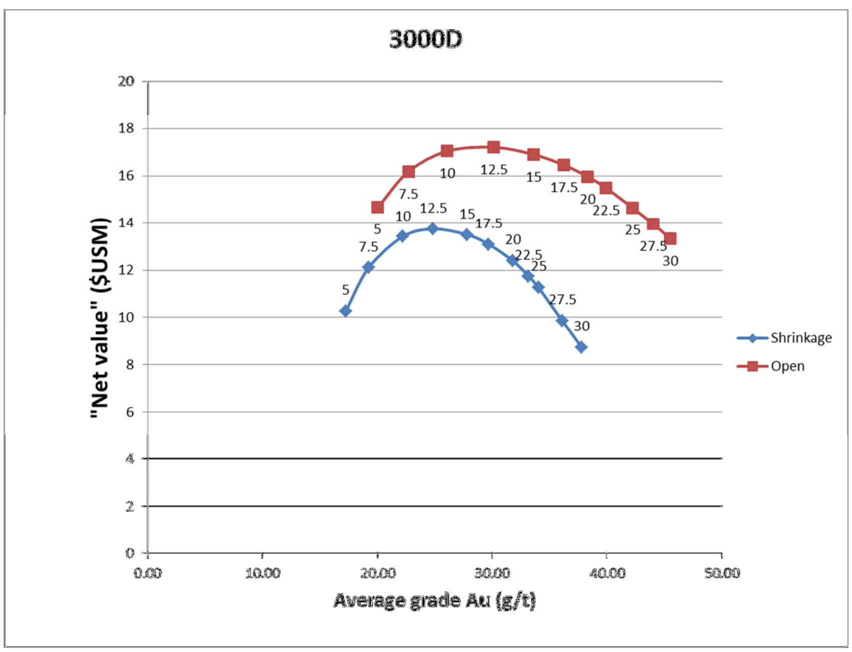


Figure 18-33 "Net Value" zone 3077

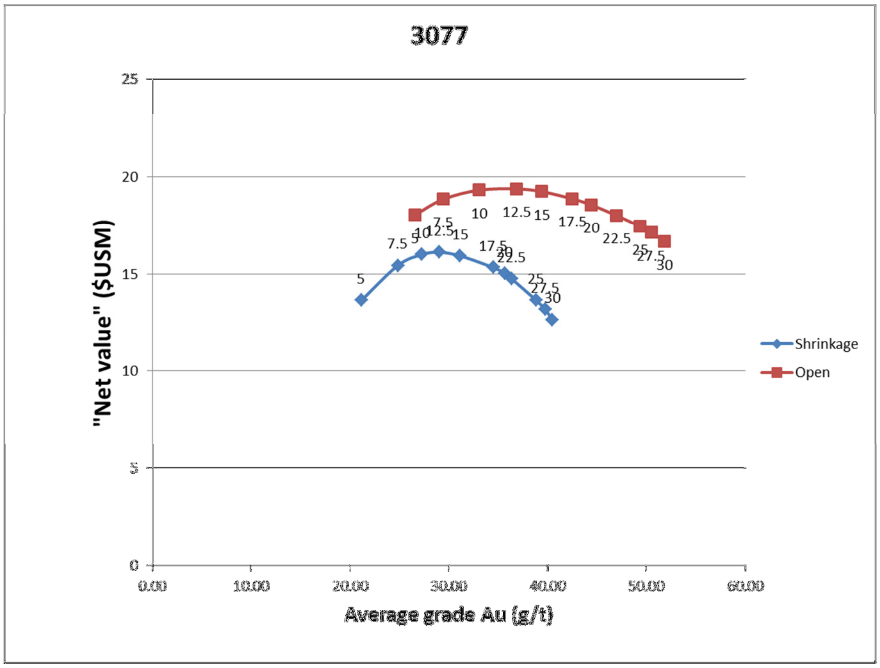


Figure 18-34 "Net Value" zone 3300_383.

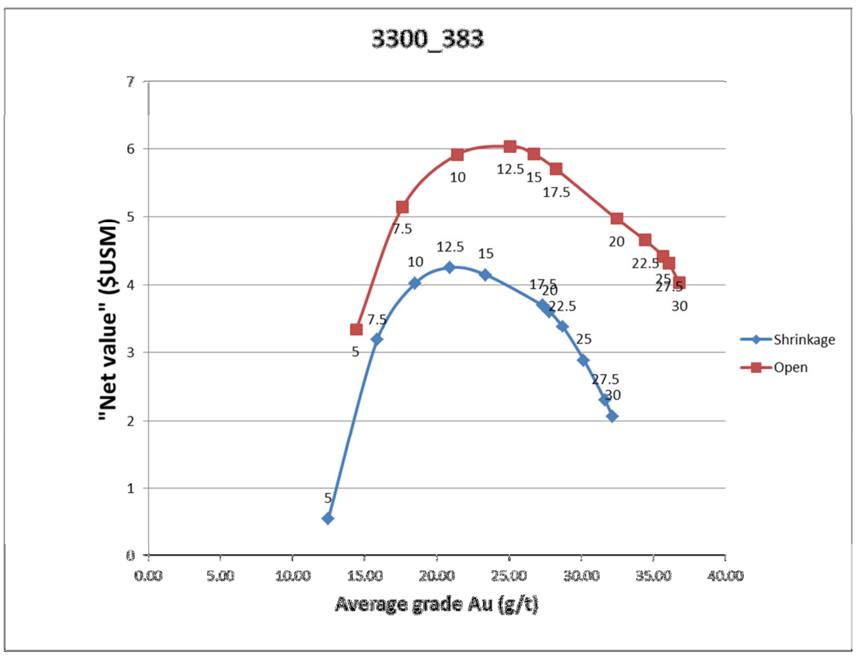


Figure 18-35 "Net Value" zone J5.

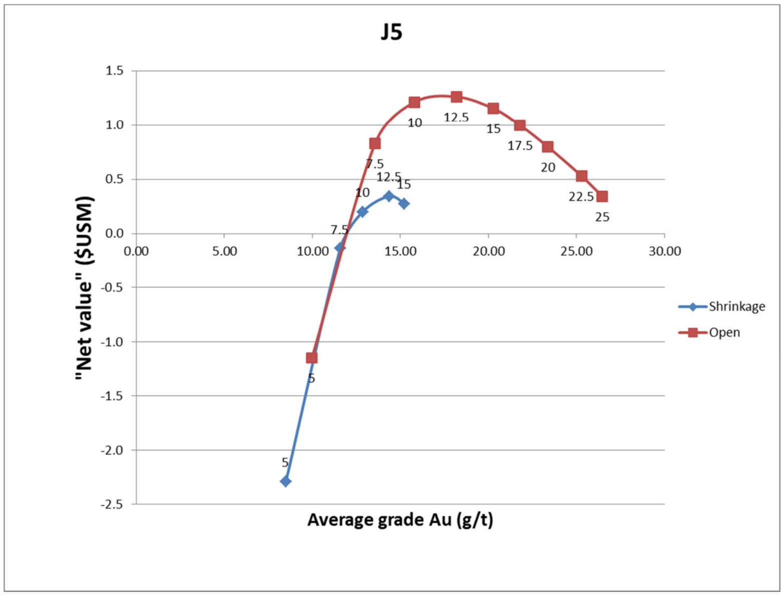


Figure 18-36 "Net Value" zone M1.

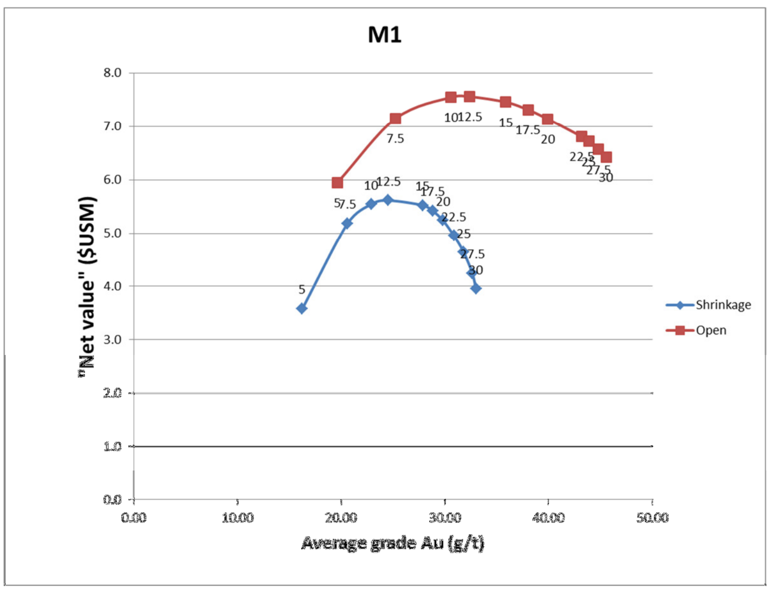


Figure 18-37 "Net Value" zone M6.

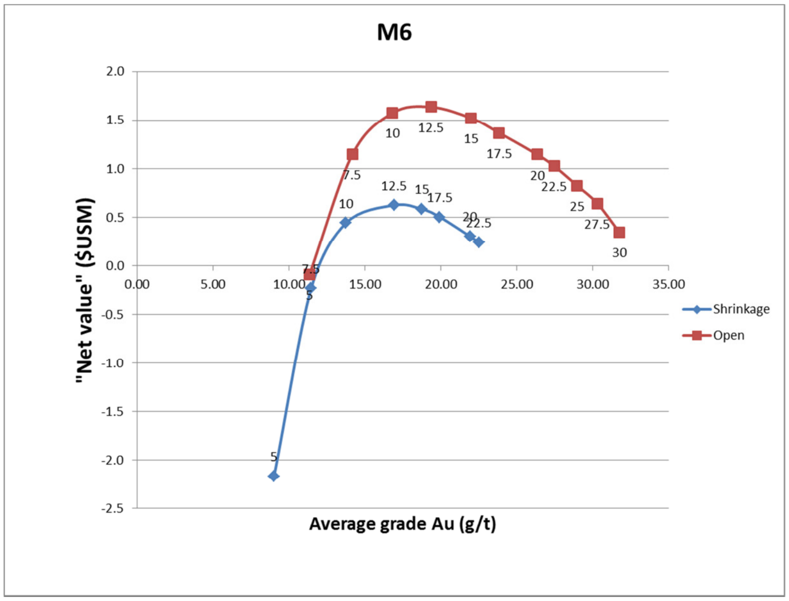


Figure 18-38 "Net Value" Zone M3 + M5

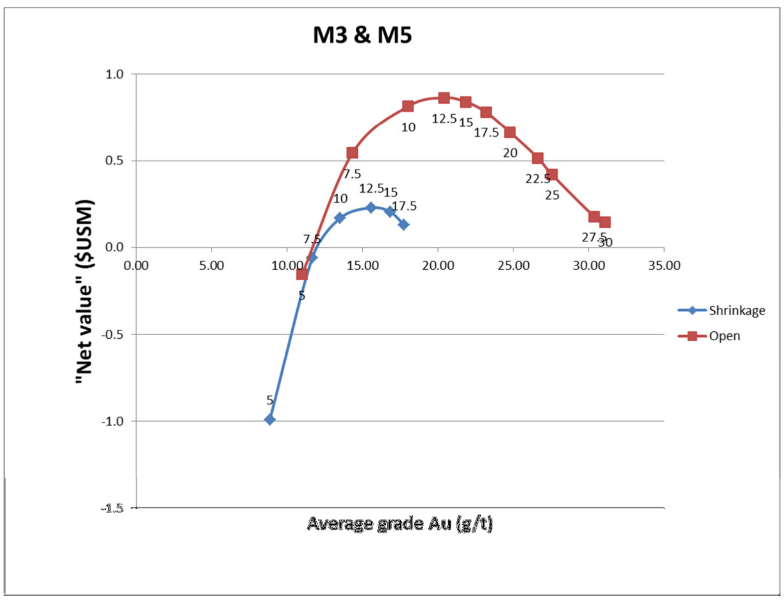
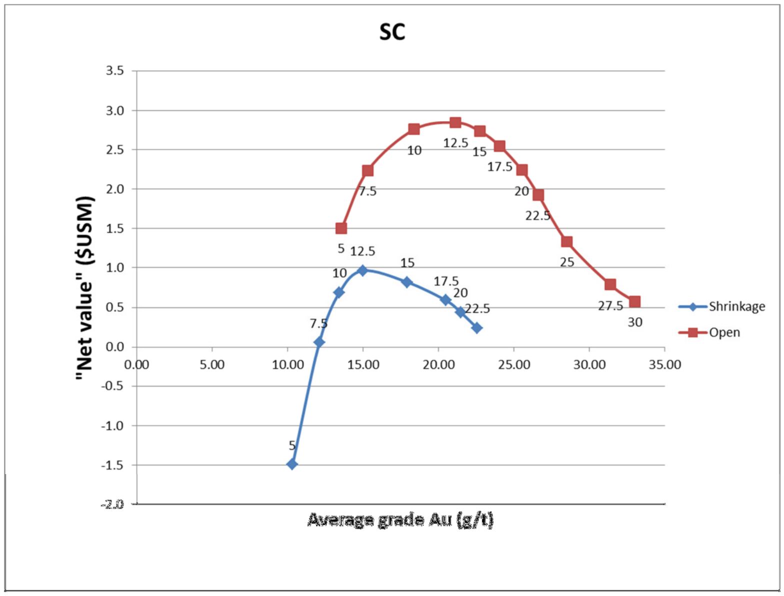


Figure 18-39 "Net Value" zone SC.



The analysis above presents a compelling case to use a highly selective mining method like shrinkage or CAF and also shows that because of the low daily production (150 t), and the high unit costs that this rate generates, that minimization of dilution will be the key to success at Nixon Fork.

18.2.3 Underground design

Sublevel intervals

Existing underground workings have a level interval of between about 15m and 30m. Because mining selectivity is a so important to success at Nixon fork, Snowden has determined that a sublevel interval of 15m would be appropriate for new development. This is a reasonable level spacing for shrinkage stoping as it allows several stopes to be developed simultaneously in a zone, and offers flexibility and selectivity in stope design and production.

Underground development

Drives have been designed with dimensions of 4.0 mH by 4.0 mW, suitable for small mechanized equipment. Declines and level accesses have arch profiles to improve long term stability. Sill drives have a square profile for improved access during drilling and mucking. Declines have been designed at a grade of 15%. Conceptual development was generated to access all zones regardless of whether or not the mineral inventory justified development costs. This is because in the next step of the evaluation, Snowden's Evaluator tool will be populated with the conceptual development, and make a determination on a stope by stope basis if that proposed conceptual development is of value (that is if the accesses to the zone and level is profitable).

Underground water

The property is operated as a zero discharge facility. This means that water cannot be pumped from the underground excavations and discharged off-site.

About 50% of the current potential mining inventory is below the current maximum depth of the mine (the 156m elevation). This depth also approximates the known water table. There is no physical constraint to continuing to mine below this depth apart for the requirement to maintain zero discharge from the facility.

Above the ramp bottom the existing workings appear quite dry. According to FAU inflow around the 156m level is ongoing but modest, estimated to be less than 1.0 l/s. FAU states that over the period of August to December 2010, water was pumped from the bottom of the ramp to use as drill water for one diamond drill operating in the underground mine. This modest consumption caused the water level in the mine to decrease steadily to the point that fresh water had to be drawn from surface for drilling.

FAU states that sudden inflows of water have been experienced by the mine in past operations, the most significant raising the water level at the bottom of the mine by 20 m. These have been caused by accessing perched water in the natural voids which, when drained, did not continually flow.

According to FAU a survey performed over one year from June 1998 to May 1999 showed the water level at the bottom of the mine to fluctuate seasonally by 6 m, with the maximum elevation of 158 m in mid-September and a minimum elevation of 152 m in mid-October. The full range of this fluctuation represents only 900,000 litres of water.

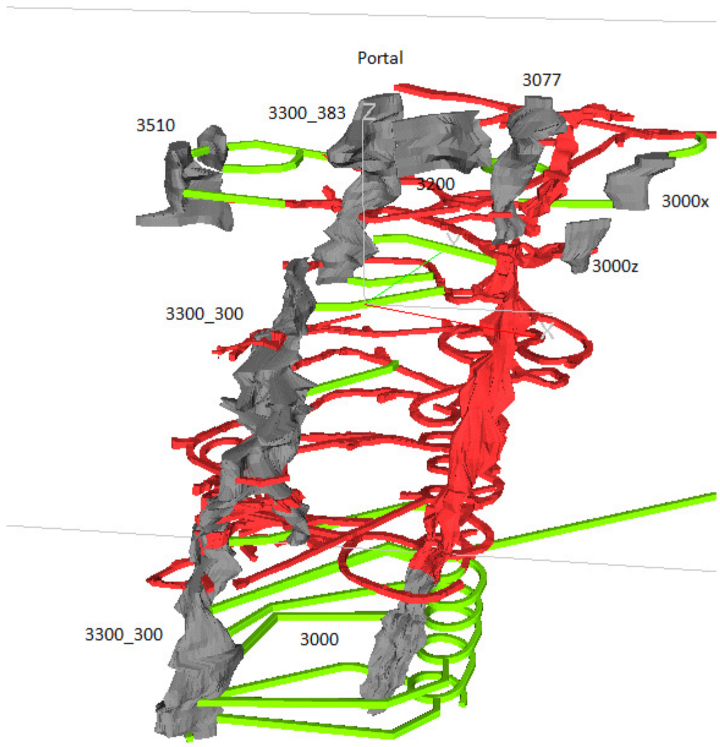
Although considered a moderate to low risk, FAU intends to deal with potential water ingress as a consequence of carrying out mining activities below the 156m level in the following way:

- A bulk water discharge permit may be applied for. Water quality data is being accumulated in support of this application, which is anticipated to be a lengthy process, taking as long as 18 months.
- A dam will be designed and installed inside the mine at 190 elevation to act as a reservoir for mine use, primarily as drill water. Water from the bottom of the ramp will be pumped to this reservoir.
- If necessary, excess water will be evaporated at the mine portal using misting sprayers. These are capable of releasing around 7.5 l/s, particularly in dry climates such as at the site.

Crystal Area

The underground concept for the crystal area is shown in Figure 18-40. The existing development and workings are shown in red; the proposed conceptual development is shown in green; and mineralised zones are in grey.

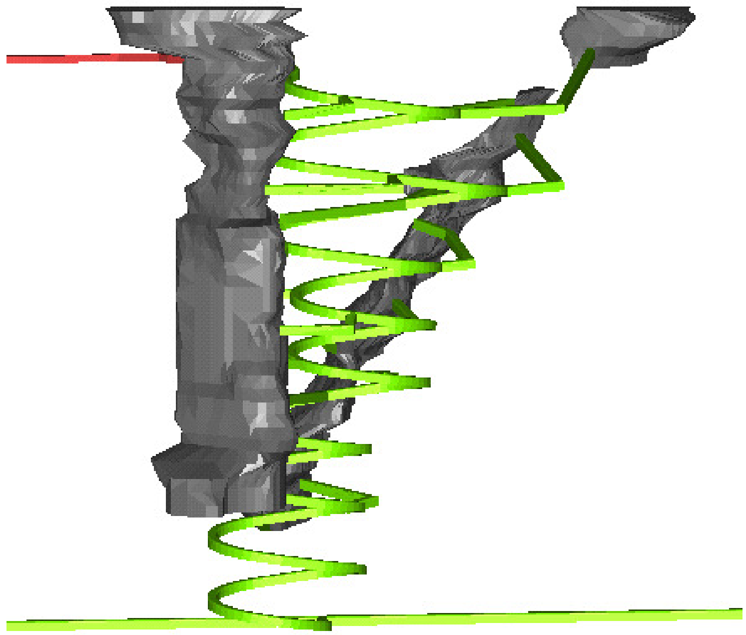
Figure 18-40 Crystal area conceptual design



J5, Southern cross, and 3100 conceptual development

The conceptual design for J5, SC, and 3100 is shown Figure 18-41. This area is accessed by extending a decline from the top and by joining that decline to a long drift from 3000 at the 155 mRL. This same long drift accesses the Mystery zone. The existing development and workings are shown in red; the proposed conceptual development is shown in green; and mineralised zones are in grey.

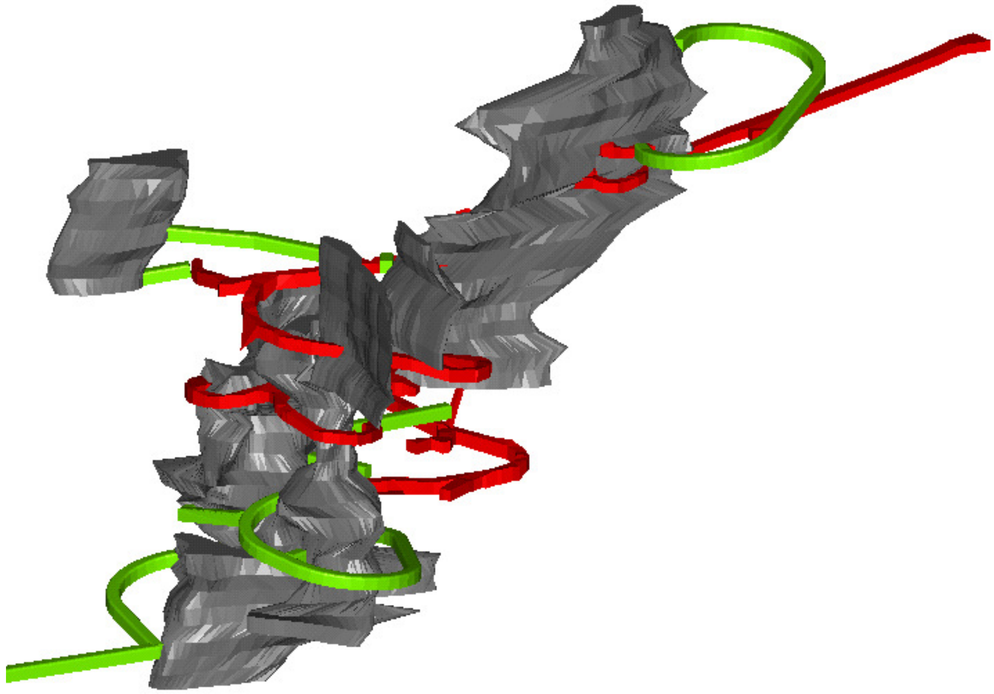
Figure 18-41 J5, SC, and 3100 conceptual design



Mystery

The conceptual development for the Mystery area is shown in Figure 18-42. Mineralised zones can be accessed from the existing decline and some extensions to the existing declines. The whole Mystery area can be accessed underground from the Crystal area via the drift at the 155 level at the bottom of the zone. The existing development and workings are shown in red; the proposed conceptual development is shown in green; and mineralised zones are in grey.

Figure 18-42 Mystery area conceptual design



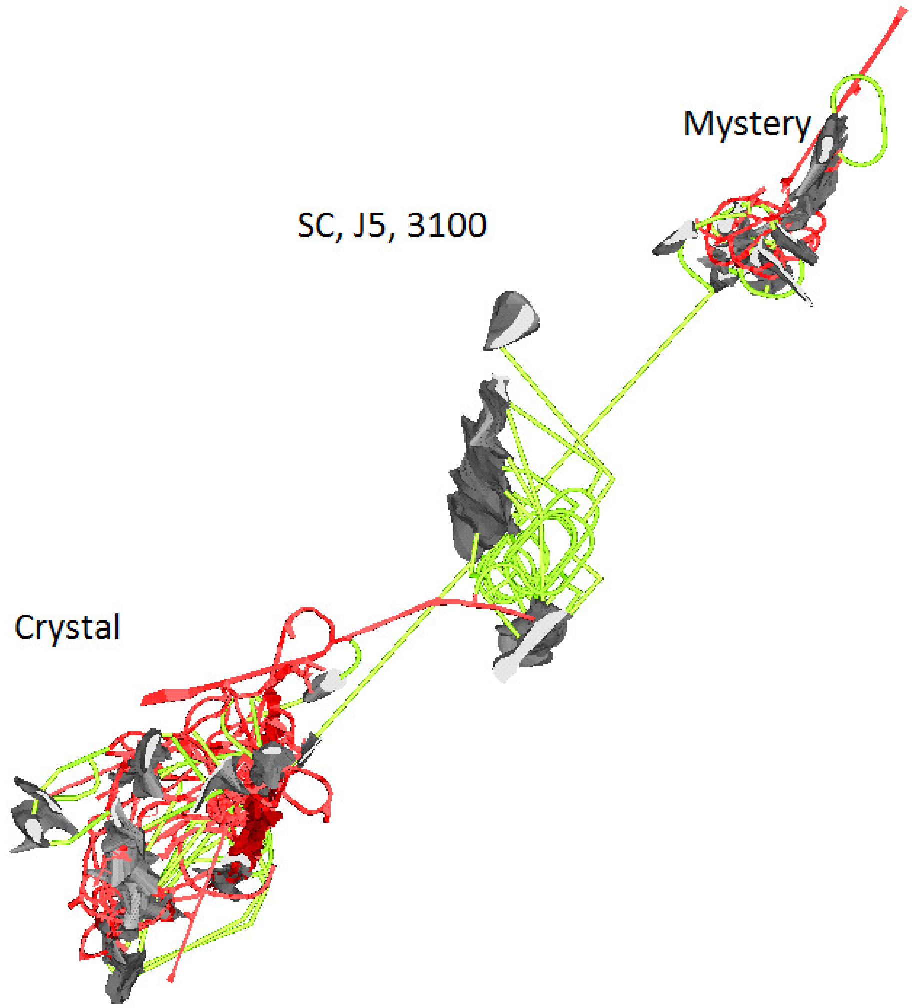
Underground summary

A summary of the physical parameters associated with the preliminary underground designs is presented in Table 18-15 and in a plan view in Figure 18-43. It should be noted that these inventories are based on the conceptual designs and may not include areas identified as potential feed during the Stopesizor analysis, or some stopes which may be identified as being not beneficial when specific development and access requirements are considered.

Table 18-15 Summary of underground conceptual designs

Item	Units	Crystal	3100,J5, SC	Mystery
Development	km	2.4	2.1	1.25
Potential feed above 12.5g/t COG	Kt	119.4	14.2	21.2
Grade of potential feed above 12.5g/t COG	Au g/t	32.7	20.8	31.2

Figure 18-43 Plan view of project areas.



18.3 Mining schedule

A preliminary schedule for the underground designs has been prepared using Snowden's Evaluator software.

Evaluator is a scheduling package based on a Mixed Integer Programming (MIP) formulation. The software enables multiple sources (e.g. underground production and underground development) to be modelled together and optimized by schedule and cut-off grade simultaneously, for a net present value (NPV) objective. The modelling of each location incorporates the input of economic and technical parameters. The economic parameters include price, operating cost, capital cost, and discount rate. The technical parameters include mining recovery and dilution as well as metallurgical recovery. The optimization honours sequencing and physical capacity limit constraints to ensure a feasible solution. Given the complexities of this project, particularly the dependencies between the various deposits and underground development options, Evaluator was deemed an appropriate and necessary tool to provide guidance for strategy selection.

Schedule parameters

Snowden has identified underground trucking as a potential bottleneck at Nixon Fork. Existing trucking capacity is 3 x 10 t trucks, whose average speed (up and down the decline) is about 3 km/h. Some preliminary work by Snowden indicated that the existing truck fleet would be unlikely to meet the needs of the mine, and as a result of this work FAU has scheduled the purchase of 2x20t trucks to replace the 3x10t trucks prior to commencing production. So that trucking capacity could be used as a constraint in the scheduling work, Snowden calculated the trucking capacity of a single truck by level and by zone. The assumptions presented in Table 18-16 were used in deriving trucking capacity calculations

Table 18-16 Truck productivity assumptions.

Assumption	Value	Unit
Truck capacity (new)	20	Tonnes
Distance from portal to dumping point (used for ore and waste)	150	M
Time to dump	1	Minute
Time to be loaded	5	Minute
Decline grade	15	%
Average speed	5.0	Km/hr.
Truck availability	75	%
Truck utilization (of available hours)	85	%

By combining the assumptions in Table 18-16 with the conceptual development and existing development detailed in Figure 18-40, Figure 18-41, Figure 18-42, and Figure 18-43 Snowden calculated the capacity of a single truck by zone and by level. This capacity includes the travel time from the zone/level to the ROM pad at the Crystal portal. The results of these calculations are provided in Table 18-17, below.

Table 18-17 Single truck productivity by zone and by level (new trucks)

Level	Zone truck single truck productivity (t/h)		
	Crystal	J5/3100/NS	Mystery
400	54.36	43.07	
385	46.44	37.95	
370	40.54	33.91	
355	35.96	30.65	
340	32.32	27.96	
325	29.34	25.71	
310	26.87	23.79	8.17
295	24.78	22.14	8.38
280	22.99	20.7	8.61
265	21.45	19.44	8.85
250	20.09	18.32	9.10
235	18.90	17.32	9.37
220	17.84		9.65
205	16.90		9.95
190	16.05		10.27
175	15.28		10.62
160	14.58		10.98
145	13.94		
130	13.36		
115	12.82		
100	12.32		
85	11.87		
70	11.44		

Evaluator can be configured to purchase additional capacity should it enhance value. In the case of the trucking capacity, Evaluator was allowed to purchase additional trucking capacity at a cost of $US400,000 per unit.

The technical parameters required for Evaluator were determined from public domain research, advice from FAU and FAU's consultants, and from reference to Snowden's database of relevant performance data.

The Evaluator parameters are shown in Table 18-18.

Table 18-18 Evaluator parameters and assumptions

Item	Unit	Value
Ultimate process capacity (in the schedule production is ramped up over 4 months)	t/month	4500
Process cost	$/t	120
G&A cost	$/t	190
Mining cost (excluding trucking)	$/t	86
Gold price	$/tOz	1200
Gold cost*	$/tOz	74.6
Discount rate	%/year	5.0
Trucking and loading cost	$/t	120
Final process plant recovery (in the schedule recovery is ramped up from 90 to 95% over 6 months)	%	95
Development maximum rate	m/month	82
Development cost (excluding trucking)	$/m	1,173
Unplanned mining loss (including pillars)	%	5
Unplanned dilution	%	5
Dilution grade	g/t	0

* *Gold cost consists of: 6% royalty, refining charge of $0.75/tOz, and refiner return of 99.875%*

Copper and Silver revenues

This evaluation does not include revenues that may be derived from the sale of Silver in Dore and Silver in the Copper concentrate, and Copper from the Copper concentrate. This is because these metals were not estimated in the resource model, so accordingly their likely concentrations in the predicted feed are unknown. It should be noted however, that in the past, at Nixon Fork, revenues have been generated from Copper and Silver and are likely to be generated in the future. These additional revenues would represent further upside on this preliminary assessment.

Schedule results

The results from the Evaluator schedule are summarized in Figure 18-44 to Figure 18-48, and in Table 18-20 to Table 18-22. Key findings include:

- The two year schedule was supported by the current resources
- No additional trucks are purchased (other than the initial planned purchase of 2x20t trucks)
- Only the top two levels of the J5/3100/SC decline are developed as this is the area where most of the higher grade ore exists

- The underground connection to Mystery is developed, with the lower areas of that zone mined in the latter part of the schedule.
- Development runs at full capacity (82m/month) for most of the schedule.
- The COG and resulting average mined grade that Evaluator used for each area is presented in Table 18-19 below

Table 18-19 COG and mined grade determined by the optimisation

Area	Evaluator determined COG (AU g/t)	Average mined grade (Au g/t)
3077 (Crystal)	12.5	38.6
3000D (Crystal)	15.0	36.3
3300_300 (Crystal)	15.0	32.2
3300_383 (Crystal)	15.0	25.9
SC (Southern Cross)	17.5	21.1
M1 (Mystery)	15.0	35.4
M6 (Mystery)	17.5	23.8
M3 (Mystery)	17.5	20.5
M5 (Mystery)	17.5	22.3

Figure 18-44 Schedule of potential feed

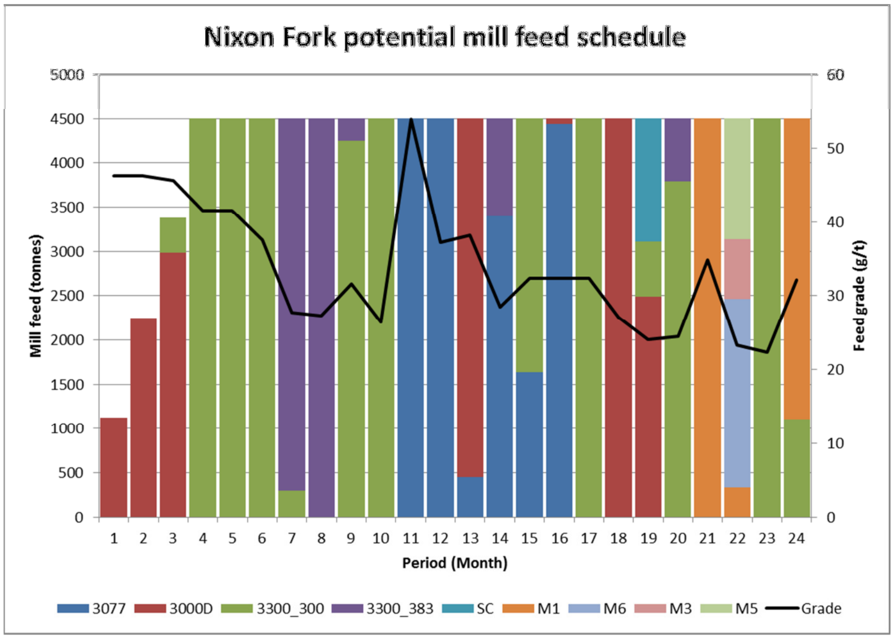


Figure 18-45 Schedule of gold production by category of resource source

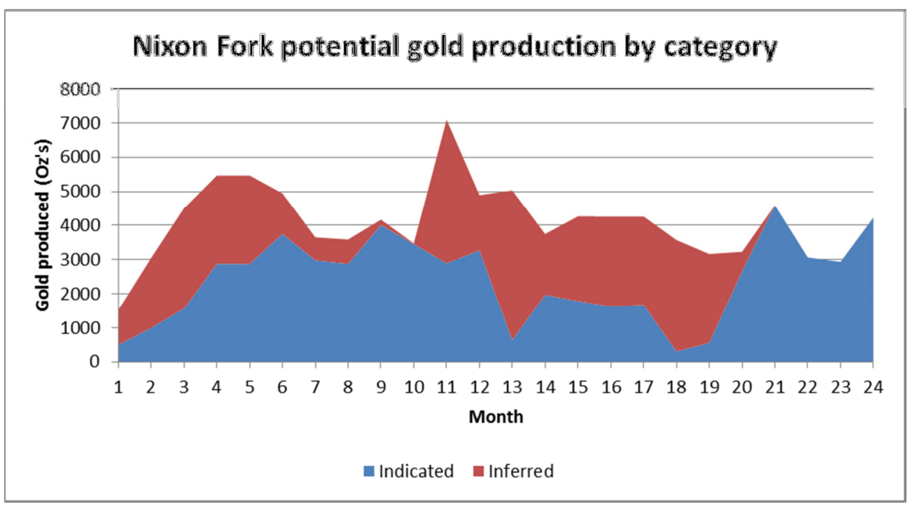


Figure 18-46 Schedule of potential cash flow

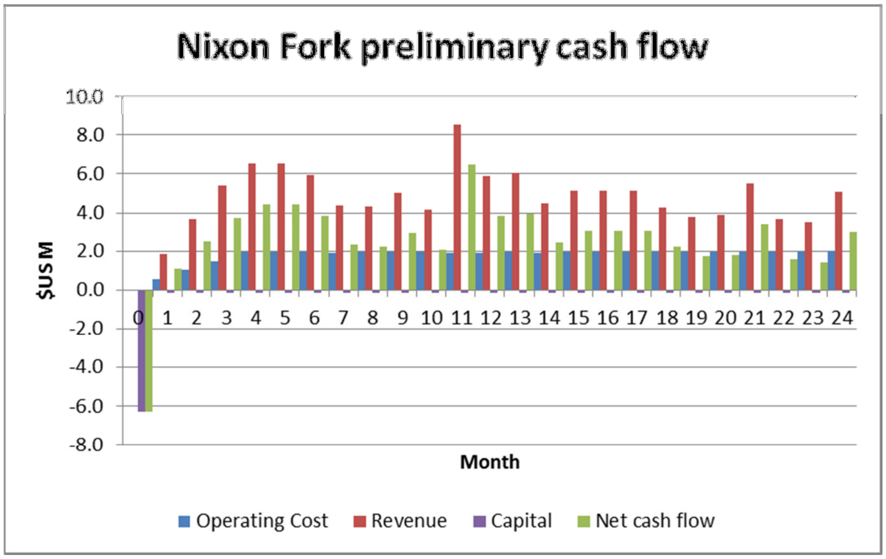


Figure 18-47 Development schedule

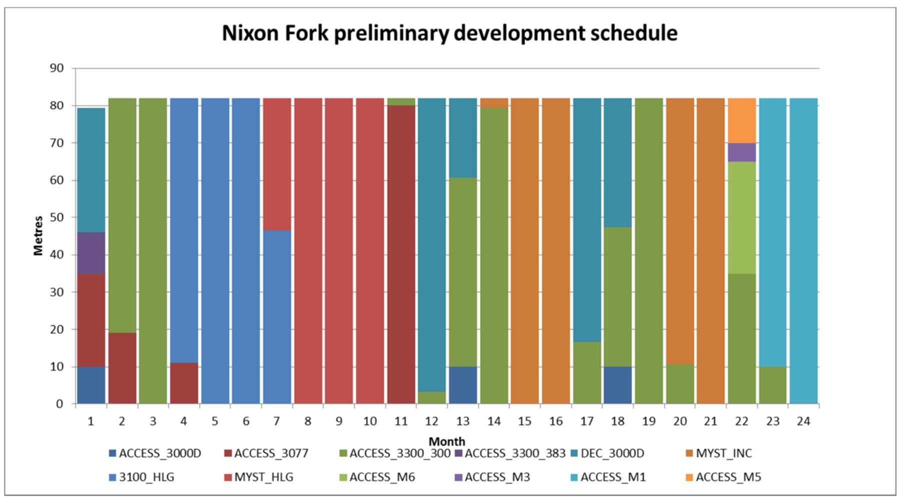


Figure 18-48 Schedule of potential production by category of resource source

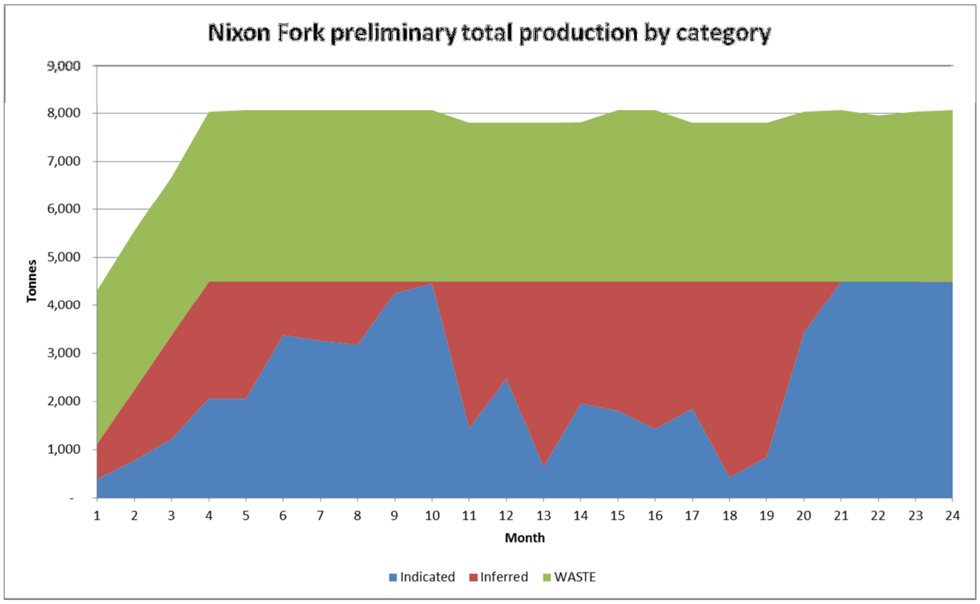


Table 18-20 Project schedule - material movement and feed grades

		Month																								
Zone	Item	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**Total**
3077	tonnes	-	-	-	-	-	-	-	-	-	-	4,500	4,500	452	3,398	1,638	4,443	-	-	-	-	-	-	-	-	**18,931**
	g/t											49.0	33.8	44.4	28.4	27.3	29.4									**35.1**
3000D	tonnes	1,125	2,250	2,982	-	-	-	-	-	-	-	-	-	4,048	-	-	57	-	4,500	2,486	-	-	-	-	-	**17,448**
	g/t	42.1	42.1	42.0										33.7			33.7		24.7	23.9						**33.0**
3300_300	tonnes	-	-	393	4,500	4,500	4,500	298	-	4,248	4,500	-	-	-	-	2,862	-	4,500	-	624	3,795	-	-	4,500	1,110	**40,331**
	g/t			37.8	37.8	37.8	34.2	31.3		29.4	24.0					30.7		29.4		19.8	23.1			20.3	16.9	**29.3**
3300_383	tonnes	-	-	-	-	-	-	4,202	4,500	252	-	-	-	-	1,102	-	-	-	-	-	705	-	-	-	-	**10,760**
	g/t							24.8	24.8	18.2					18.2						18.2					**23.5**
SC	tonnes	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1,390	-	-	-	-	-	**1,390**
	g/t																			19.2						**19.2**
Mystery		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4,500	4,500	-	3,389	**12,389**
																						31.7	21.2		33.3	**28.3**
Total mill feed	tonnes	**1,125**	**2,250**	**3,375**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,499**	**101,249**
Feed grade	g/t	**42.1**	**42.1**	**41.5**	**37.8**	**37.8**	**34.2**	**25.2**	**24.8**	**28.8**	**24.0**	**49.0**	**33.8**	**34.8**	**25.9**	**29.5**	**29.4**	**29.4**	**24.7**	**21.9**	**22.3**	**31.7**	**21.2**	**20.3**	**29.2**	**30.2**
Cost	**$/oz**	**371.9**	**337.9**	**330.9**	**356.0**	**356.5**	**393.3**	**527.9**	**537.9**	**467.1**	**561.1**	**268.5**	**392.5**	**386.8**	**512.0**	**456.6**	**453.0**	**456.7**	**546.4**	**613.4**	**605.5**	**431.5**	**641.1**	**665.6**	**469.4**	**447.4**
	$/t	**504.0**	**458.0**	**441.5**	**432.4**	**433.1**	**432.0**	**428.3**	**428.9**	**432.5**	**433.0**	**423.3**	**426.8**	**432.4**	**426.5**	**432.6**	**428.7**	**432.3**	**433.6**	**431.2**	**434.4**	**439.9**	**436.6**	**434.1**	**441.4**	**433.8**

Table 18-21 Detailed underground development schedule

		Period																							
Zone	Level	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	**Total**
ACCESS_3000D	145	10																								10
	130													10												10
	115																		10							10
ACCESS_3077	385	25	19		2																					46
	370				9							80														89
ACCESS_3300_300	250											2	3					17					35	10		67
	145		63	82																						145
	130													51	79											130
	115																		37	82	11					130
ACCESS_3300_383	385	11																								11
DEC_3000D	145	33																								33
	130												79	21												100
	115																	65	35							100
MYST_INC	190																					53				53
	175																				71	29				100
	160															18	82									100
	145														3	64										67
3100_HLG	155				71	82	82	47																		281
MYST_HLG	155							35	82	82	82															281
ACCESS_M6	190																						30			30
ACCESS_M3	190																						5			5
ACCESS_M1	310																							72	82	154
ACCESS_M5	220																						12			12
Grand Total		**79**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**82**	**1914**

Table 18-22 Detailed underground production schedule

		Period																								
Zone	**Level**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**Total**
3077	400														2,652	1,638										4,290
	385																4,443									4,443
	370												4,500	139	194											4,833
	355														552											552
	340											4,500		313												4,813
3077 Total												**4,500**	**4,500**	**452**	**3,398**	**1,638**	**4,443**									**18,931**
3000D	145	1,125	2,250	2,982																						6,357
	130													4,048			57									4,105
	115																		4,500	2,486						6,986
3000D Total		**1,125**	**2,250**	**2,982**										**4,048**			**57**		**4,500**	**2,486**						**17,448**
3300_300	265																	525		624						1,149
	250																							4,500		4,500
	235																								1,110	1,110
	220						2,513	298		3,156																5,967
	205									1,092	4,500															5,592
	145			393	4,500	4,500	1,987																			11,380
	130															2,862		3,975								6,837
	115																				3,795					3,795
3300_300 Total				**393**	**4,500**	**4,500**	**4,500**	**298**		**4,248**	**4,500**					**2,862**		**4,500**		**624**	**3,795**			**4,500**	**1,110**	**40,331**
3300_383	385									252					1,102						705					2,059
	370							4,202	4,500																	8,702
3300_383 Total								**4,202**	**4,500**	**252**					**1,102**						**705**					**10,760**
SC	385																			1,390						1,390
SC Total																				**1,390**						**1,390**
M1	295																								3,302	3,302
	280																					1,038	333			1,371
	250																					3,462				3,462
	235																								87	87
M1 Total																						**4,500**	**333**		**3,389**	**8,222**
M6	175																						2,130			2,130
M6 Total																							**2129.8**			**2,130**
M3	190																						677.28			677
M3 Total																							**677.28**			**677**
M5	235																						422			421.6
	220																						938.4			938.4
M5 Total																							**1,360**			**1360**
Grand Total		**1,125**	**2,250**	**3,375**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,500**	**4,499**	**101,249**

18.4 Capital and operating cost estimates

18.4.1 Operating costs

The operating costs have been generated by Snowden and are based upon budgets an estimates provided by FAU. Snowden has examined this budgets and estimates in detail and compared them with information contained in Snowden's database, and Snowden has found the estimates to be consistent with Snowden's expectations for the operation that is envisaged at Nixon Fork. Operating costs are detailed in Table 18-18 in Section 18.3

18.4.2 Capital costs

Capital cost estimates have been generated based on information provided by FAU. FAU has already funded the capital for the mill and infrastructure upgrade, so the only capital that has been applied to this estimate is that associated with getting the mine into production.

Note that:

- In this preliminary analysis, all underground development has been treated as an operating cost.
- Sustaining capital has been provided for at 2.5% of start-up costs per annum.

An itemised list of the capital costs included in the PEA is shown in Table 18-23.

Table 18-23 Capital and start-up costs

Item	Units	Unit cost ($M)	Total cost ($M)
Remote loader	1	0.500	0.500
20t underground truck	2	0.400	0.800
Forklift for bolting	1	0.130	0.130
Alimiak/rail/accessories	1	0.300	0.300
Misting sprayer	1	0.075	0.075
First fill supplies	1	0.150	0.150
Subtotal			**1.955**
Contingency	30%		0.590
Working capital	1.5	2.500	3.750
Total			**6.295**

18.5 Economic analysis

Using the Evaluator software package Snowden has calculated the cash flows for the schedule, these are presented in Section 18.3. In undertaking the economic analysis:

- no provision has been made for revenues derived by the processing of tailings, this has been covered in another technical report
- no provision has been made for exploration expenditure attributable to the operation
- no provision has been made for environmental bonds, rehabilitation costs or salvage revenues
- no provision has been made for project financing arrangements or sales arrangements other than spot price contracts
- no provision has been made for depreciation of capital expenditure
- no provision has been made for government royalties or taxes
- no provision has been made for inflation of costs with time
- the life of the project has been limited to 24 months.

The results of the financial modelling for the base case and two other cases are summarised in Table 18-24.

Table 18-24 Summary of financial model

Item	Units	Gold price ($US/tOz)		
		1,033 (3 yr. avg.)	1,200 (base case)	1,500 (upside)
Undiscounted cash flow	$M	47.81	64.28	93.63
NPV @ 5% discount	$M	45.30	60.94	88.86
IRR	%	462	549	853
Payback period	Months	4	3	3

No price or cost variance assumptions were made in the calculation of these financial indicators.

Average unit costs for the two year schedule are presented in Table 18-25.

Table 18-25 Unit cost for two year schedule

Item	$/t processed	$/tOz produced
Mining Cost	124	128
Processing Cost	190	196
G & A Cost	120	124
Total	**434**	**447**

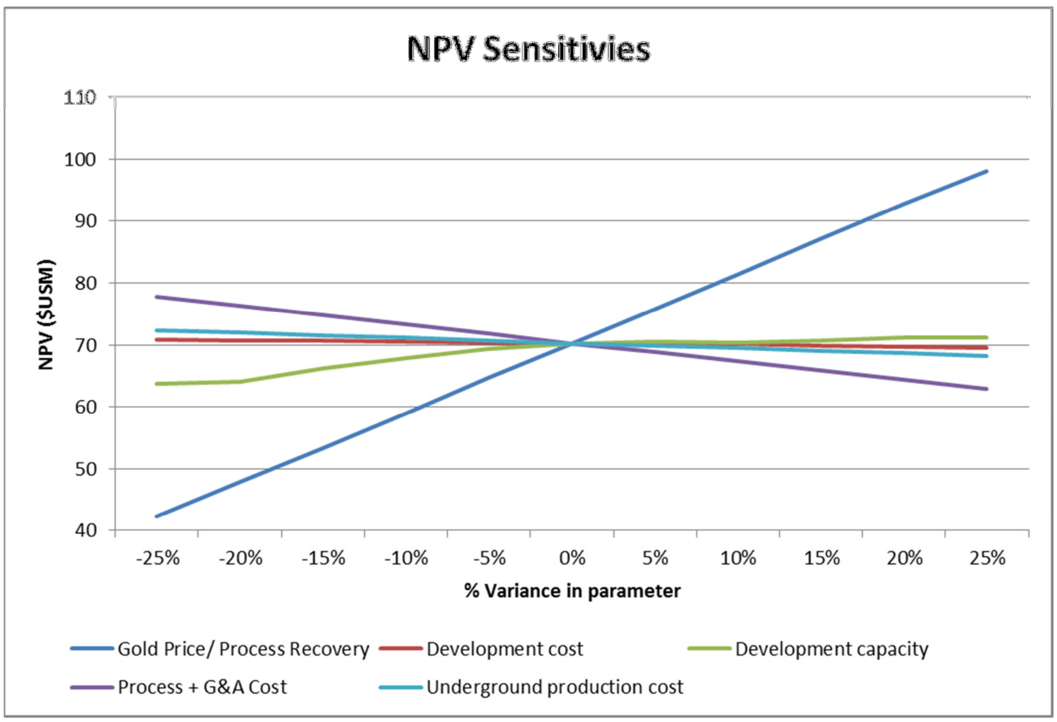
18.5.1 Sensitivity analysis

Sensitivity analyses were undertaken on the Project schedule for the following key parameters:

- gold price from $700 /tOz to $1500 /tOz
- processing and site operating costs
- underground mining production cost (excludes development)
- underground development cost
- underground development capacity

The outcomes of the sensitivity analyses for NPV are presented in Figure 18-49. The results show that the project NPV is most sensitive to the gold price/process recovery. The Project also exhibits a moderate sensitivity to the process and site costs as well as sensitivity to development capacity. The development capacity sensitivity is relatively high for capacities below about 80m/month. This reflects the requirement to put in place the connection to the Mystery zone from the Crystal zone prior to accessing some resources in both Mystery and Crystal. For development rates above 80m/month the project is insensitive to development. Sensitivity to mining costs, and development cost, is low.

Figure 18-49 Sensitivity analysis results - NPV



19 Conclusions and recommendations

It can be concluded from the current study that for the first 24 months of production at Nixon Fork there is potential for a profitable operation.

With a gold price of $US1200/tOz and the other specified financial assumptions, the project delivers an IRR of 549% on an undiscounted cash flow of $64.3 M for this two year plan.

It is recommended that FAU continue with its evaluation of the Nixon Fork Project and progress towards undertaking a Prefeasibility Study to address the remaining material project uncertainties, or commence test mining and processing underground material to demonstrate the economics.

Resource estimation recommendations:

- Maintain a substantial ongoing exploration program so that reserves can be replaced as they are depleted by mining.
- Determine full extent of previously mined material to appropriately deplete the resource model, so that the mining inventory can be more accurately determined.
- Undertake a drilling program so that more of the Resource can be classified as Measured or Indicated which may then be converted into Reserves after completion of a Prefeasibility Study.
- Review the resource confidence classification criteria for future Resource estimates and ensure that all aspects affecting confidence in the Resource estimation are considered, including geological understanding, complexity, and continuity, the sample data density and orientation (including sample grades and bulk density data), the data accuracy and precision as established through the QAQC programs, grade continuity including the spatial continuity of mineralisation, the quality of the estimates, and the results of the estimation validation.

Metallurgical

- One of the principle driving forces for the high COG at Nixon Fork is the low processing rate (which gives rise to high unit costs). Snowden recommends that Nixon Fork investigate cost effective alternatives to increase the mill throughput. By increasing mill throughput, COG's can be reduced and the size of the resource above cut-off will be substantially increased - for example the mining inventory above a cut-off 10g/t is almost double that of 15g/t.

Other

- Handling the moderate water inflows derived from mining below the water table is important for the sustainable exploitation of these resources. In the schedule this represents about 50% of the mined inventory. It is therefore recommended that FAU proceed with evaluation and the implementation of the identified options for dealing with underground water inflows.

20 References

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Note: References noted within quotations from other sources have not been reproduced. Refer to the source of the quotation for details of such references.

21 Dates and signatures

Nixon Fork Project Preliminary Economic Assessment

Alaska, USA

February 25 2011

Issued by:
Fire river gold corp.

[SIGNED]	February 25, 2011
----------------------------------	-------------------------
Anthony Finch	Date
[SIGNED]	February 25, 2011
----------------------------------	-------------------------
Gary Giroux	Date
[SIGNED]	February 25, 2011
----------------------------------	-------------------------
Richard Flanders	Date
[SIGNED]	February 25, 2011
----------------------------------	-------------------------
George Rawsthorne	Date

22 Certificates

CERTIFICATE of QUALIFIED PERSON

I, Anthony Finch, Divisional Manager - Mining, of Snowden Mining Industry Consultants Pty Ltd., Suite 600, 1090 West Pender St, Vancouver, British Columbia, do hereby certify that:

(a) I am the co-author of the technical report titled "Fire River Resources Corp: Nixon Fork Project Preliminary Economic Assessment" and dated January ??. 2011 (the 'Technical Report') prepared for Fire river gold corp.

(b) I graduated with a Bachelor of Engineering (Mining) from The University of Queensland in 1987 and a Bachelor of Economics from The University of Queensland in 1993.

(c) I am a Member of the Australasian Institute of Mining and Metallurgy.

(d) I have worked as a mining engineer continuously for a total of 24 years since my graduation from university in operational, managerial, technical and consulting roles.

(e) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('the Instrument') and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a 'qualified person' for the purposes of the Instrument. During the last 5 years I have been the mine manager of a narrow gold mine, and have been involved in the evaluation of several precious and base metal deposits

(f) I have made a site visit to the Property on 20 July 2010.

(g) I am responsible for the preparation of sections 1, 2, 3, 15, 17.2, 18 and 19 of the Technical Report entitled"Nixon Fork Project, Alaska, USA, Preliminary Economic Assessment" dated February 25, 2011 ("Technical Report").

(h) I am independent of the issuer as defined in section 1.4 of the Instrument.

(i) I have not had prior involvement with the property that is the subject of the Technical Report.

(j) I have read the Instrument and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

(k) As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25th day of February, 2011 at Vancouver BC this

[signed]

Anthony Finch, B Eng.,(Min), B Econ., M AusIMM

Divisional Manager and Principal Consultant – Mining, Snowden Mining Industry Consultants.

CERTIFICATE of QUALIFIED PERSON

I, G.H. Giroux, of 982 Broadview Drive, North Vancouver, British Columbia, do hereby certify that:

(a) I am a consulting geological engineer with an office at #1215 - 675 West Hastings Street, Vancouver, British Columbia.
(b) I am a graduate of the University of British Columbia in 1970 with a B.A. Sc. and in 1984 with a M.A. Sc., both in Geological Engineering.
(c) I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia.
(d) I have practiced my profession continuously since 1970. I have had over 30 years experience in base and precious metal resource estimation and in that time have worked on many skarn deposits including New York Canyon, Merry Widow, Los Filos and Bermejal.
(e) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Policy 43-101.
(f) I am responsible for Section 17 (with the exception of 17.2); in the technical report entitled "Nixon Fork Project, Alaska, USA, Preliminary Economic Assessment" dated February 25, 2011 ("Technical Report"). This report is based on a study of the data and literature available on the Nixon Fork project and a site visit conducted during the period May 3-5, 2010.
(g) I have not previously worked on this property.
(h) As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
(i) I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.
(j) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 25th day of February , 2011

"G.H. Giroux" {signed and sealed}

G. H. Giroux, P.Eng. MASc.

CERTIFICATE of QUALIFIED PERSON

I, RICHARD W. FLANDERS, Certified Professional Geologist #10898, HEREBY CERTIFY THAT:

(a) I am currently employed as the sole owner and operator of Ridgerunner Exploration of 1870 Becker Ridge Rd, Fairbanks, Alaska, 99709, USA.

(b) I am a graduate of Michigan State University, with a B.S. degree in Geology (1975). I am also a graduate of the University of Idaho with an M.S. degree in Geology (1978).

(c) I am a member of the American Institute of Professional Geologists, the Society of Economic Geologists, and the Alaska Miners Association.

(d) I have practiced my profession continuously since 1978. I have been actively employed in various capacities in the mining industry as geologist in numerous locations in North America, Mongolia, and Russia. I have had over 30 years experience in exploration of base and precious metal and in that time have worked on many skarn deposits including Nixon Fork between 1989 and 1994.

Some of the mining and geology jobs I've had in the past 32yrs include but not limited to:

- Project and exploration geologist on underground development and recognisance or drilling programs all over Alaska and in California, Utah, Nevada, and Mongolia exploring for gold deposits of all sorts and, in addition, Sn-W, VMS deposits, and gemstones.
- On one project I was the: Project geologist, underground geologist and mine engineer and the processing plant manager.
- Chief blaster on seismic exploration programs in Antarctica.
- Project Exploration geologist at Nixon Fork periodically over a 5 yr period doing surface and underground exploration.
- Placer gold miner.
- Ran a 100-man winter oil exploration camp in the Russian Far East - Kamchatka

(e) I have read the definition of “Qualified Person” set out in National Instrument 43-101 (NI43-101) and certify that by reason of my education, affiliation with a professional organization (as defined by NI43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI43-101.

(f) I am responsible for preparations of sections 4 to 15 of the report entitled “Nixon Fork Project, Alaska, USA, Preliminary Economic Assessment” dated February 25, 2011 (“Technical Report”).

(g) I managed and worked on the Nixon Fork prospect for Central Alaska Gold Co. and Nevada Goldfields, Inc between 1989 and 1994. Other than this work, I have not had prior involvement with the property that is the subject of the Technical Report.

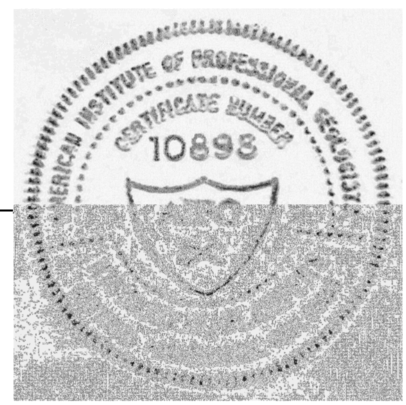
(h) I am not aware of any material fact or material change with respect to the subject matter of this Technical Report that is not reflected in the Technical Report, the omission to disclose which would make the Technical Report misleading.

(i) .I am independent of the issuer applying all of the tests in section 1.4 of NI43-101. I own no interest in any company or entity that owns or controls an interest in the properties which comprise the Nixon Fork project.

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

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

DATED in Fairbanks, Alaska this 25th day of February, 2011.



Richard W. Flanders, BS, MS, CPG#10898

CERTIFICATE of QUALIFIED PERSON

I, Timothy G. Smith, P.Eng, of 8953 Jackpine Drive, Helena, MT, do hereby certify that:

(a) I am an independent consulting engineer.

(b) I am a graduate of McGill University, with a degree in Bachelor of Engineering, Metallurgical (1979).

(c) I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia (license # 15818).

(d) I have practiced my profession since 1979.

(e) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience and affiliation with a professional association, I meet the requirements of a Qualified Person as defined in National Instrument 43-101.

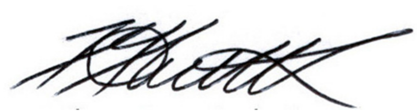
(f) I have reviewed Section 16 of the technical report entitled "Nixon Fork Project, Alaska, USA, Preliminary Economic Assessment" dated February 25, 2011 (Technical Report).

(g) As of the date of this certificate, to the best of my knowledge, information and belief, section 16 of the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

(h) I am independent of the issuer applying all of the tests in section 1.4 of the National Instrument 43-101.

(i) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 25th day of February, 2011



Timothy G. Smith, P.Eng.