Mineral Resource Estimation
Dannemora Magnetit AB
2014

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The information in this report that relates to Mineral Resources and Ore Reserves is compiled by, or under the supervision of Mr. Thomas Lindholm, GeoVista AB, who is a Fellow of the Australasian Institute of Mining and Metallurgy, and a Competent Person qualified to report on mineral resources, based on his training and experience in exploration, mining and mineral resource estimation of iron ore, base and precious metals. The Mineral Resources are reported following the guidelines of the JORC Code, 2012 edition.

Introduction

Since new results from diamond drilling and assays became available during 2010 and 2011, the new data as well as historic information has been used for the Dannemora mineral resource estimations. The first resource estimates, based partly on new data from the Dannemora iron deposit, were published in 2011 and included a verification of the usability of the historic data (pre-1993) and a bulk density determination. These chapters and also sampling and assaying procedures reported 2011 that are still valid are given in Part 1 below with only minor changes. In Part 2, the current Mineral Resource Estimation and relevant new procedures are presented.

All work carried out by Dannemora Magnetit AB, that constitutes the basis for mineral resource estimates, follow current industry standard QA/QC protocols.

For modeling, cross sections spaced 25 or 50 metres apart, geological information from underground mapping, and at one instance susceptibility measurements, respectively have been interpreted to produce strings for solid construction. All assay sections were composited before variography modeling and consecutive grade interpolation of the block models. The block sizes are based on the distribution of drillcore information as well as on the planned mining block size.

The block models have been verified with visual comparison between block values and original assays as well as with regular statistical means. De-clustering plots of block values versus assay composites have also been constructed for each mineralisation.

The resource classification is based on interpreted geological continuity, verified by variography. The absence of information on contents of modelled elements, with emphasis on deleterious elements, has been used to lower the level of confidence so that resources that otherwise classify as measured have been classified as indicated.

The general geological knowledge of the mineralized bodies within Dannemora Iron Deposit, as well as the geological knowledge of entire field, has also been important when concluding that the resource classification is based on a reasonable level of confidence.

Specific gravity has been determined for 1494 assay sections and the relation of SG as a function of the Fe-contents calculated, this has been applied to each individual model block.

The resource estimation has been modeled at a cut-off of 20 % Fe. Reporting does not exclude internal blocks that fall below the cut-off.

The present report includes updated resource estimates for all Dannemora mineralisations except for Diamanten 2, Strömsmalmen, Sjöhag and Lyndon 1 and 3. No new information has been added to these mineralisations during 2014 and thus the resource estimates are identical to those given in the Mineral Resource Estimation Report from 2013.

Note

On May 13, 2014 the Uppsala District Court approved an application for company reconstruction of Dannemora Mineral AB and Dannemora Magnetit AB. Due to this situation core drilling has been limited mainly to infill drilling and a planned exploration program was postponed. From September 15, 2014 and onwards, all forward-oriented measures such as diamond drilling have been stopped in the mine.

Part 1

1. Verification of historic data

The historic exploration database for the Dannemora Iron Ore field consists of 1085 drill holes, totaling 112765 metres of drilling. Of these, a total of 3894 sections, representing 27910 metres of core, have been assayed, in most cases for Fe and Mn. During the pre-1993 mining period several sampling schemes existed, sometimes the core was split and sometimes 0.1-0.2 metre pieces were taken out for each metre of the section to be assayed. In addition, generally only sections considered to contain >30 % Fe underwent assay. Little or nothing is known about sampling and assay QA/QC procedures during that period.

A program set out to verify the usability of historic assays was initiated in 2007. For correlation, a total of 417 sections have been sent for re-assay using an industry standard analytical method, accredited for iron ore. The results show a slight bias, with the old assays having a slightly higher Fe contents for higher Fe grades and slightly lower Fe contents for lower grades. Similarly, Mn was found to be slightly overestimated. Functions to re-calculate older assays have been developed.

The historic assays do rarely include information on deleterious elements, such as S and As. Whenever the distribution of modeled elements, with emphasis on these, has been impossible to assess due to lack of information, such resources has been classified as indicated, even though data density for Fe and Mn would otherwise suffice for a higher classification.

1.1 Re-sampling

Of the 3894 samples analyzed during the mining operation before 1993, 417 samples have been re-analyzed to confirm the quality of the historic assays. During re-sampling, chosen sample lengths are usually shorter than 5 metres, thus often shorter than the original sample lengths. To be able to compare the historic and new assays, new and historic sample start and/or stop positions coincides. For example, a historic 8 meter sample will be divided in two 4 meter samples. The historic 417 samples correspond to a sample length of 2606 metres. With the new shorter sample lengths, this results in 646 new analyses.

Each re-analyzed historic sample is compared with the new samples from the same section. When an old sample is split and re-analyzed as several shorter samples, a calculated length weighed average composition from the new samples is used for comparison against the historic assay (assuming identical or near identical density on consecutive sections).

As described in section 2.3, some historical assays are based on 0.1-0.2 metre samples of full core, selected every metre along the assayed interval. Re-assays for the same interval is, by necessity, based on a split of the remaining portions of core. This procedure increases the uncertainty in comparing and calibrating the two against each other.

1.2 Results iron re-assays

The results are shown in Figure 1. After a preliminary linear regression of the data, the regression line function was refined by calculating the standard deviation on the residuals and excluding points lying more than one standard deviation away from the line. In Figure 1, the data points used in the regression are displayed as blue diamonds. Data points excluded from the regression are displayed as red diamonds. The data conforms to a linear function, giving a low error, with an r^2 of 0.97.

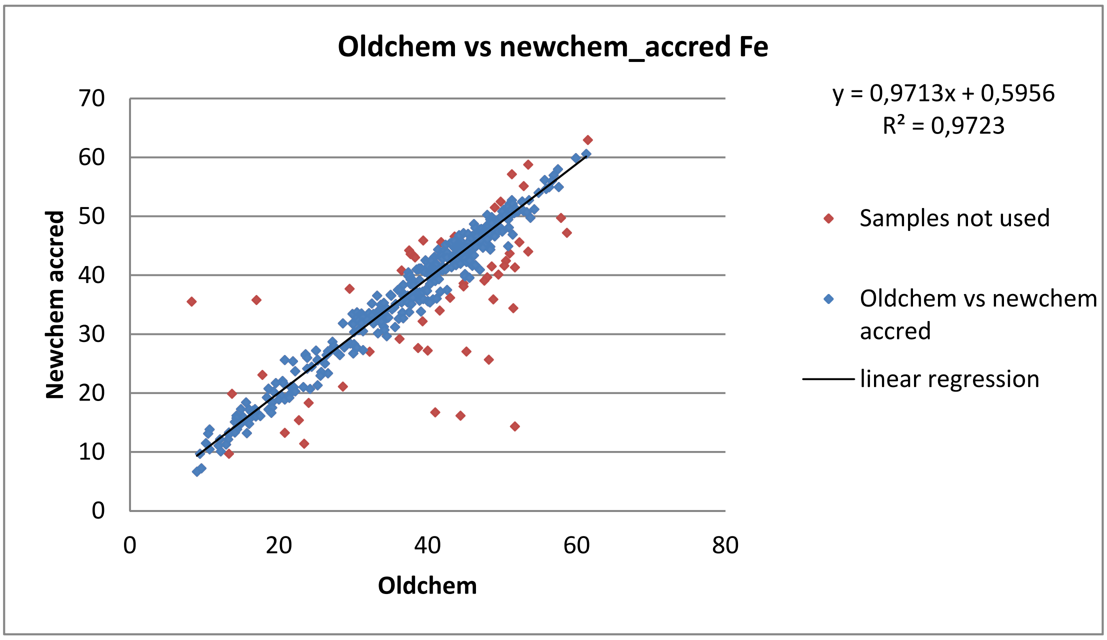

Figure 1. New Fe assays (Newchem accred) plotted against historic Fe assays (Oldchem).

The regression indicates that historic Fe grades lower than 20 % are slightly underestimated, and historic Fe grades higher than 20 % are slightly overestimated. The resulting regression function gives:

$Fe_{tg} = 0.9713 * Fe_{hg} + 0.5956$

where Fe_{tg} is the calculated true grade and Fe_{hg} is the historic Fe grade.

This function is applied to adjust the historic assays before they are used in the update of the mineral resource estimates.

1.3 Results manganese re-assays

The results are shown in Figure 2. After a preliminary linear regression of the data, the regression line function was refined by calculating the standard deviation on the residuals and excluding points lying more than one standard deviation away from the line. In addition to this, extreme grades above 6 % Mn were excluded. In Figure 2, the data points used in the regression are displayed as blue diamonds. Data points excluded from the regression are displayed as red diamonds. The data conforms to a linear function, giving a low error, with an r^2 of 0.97.

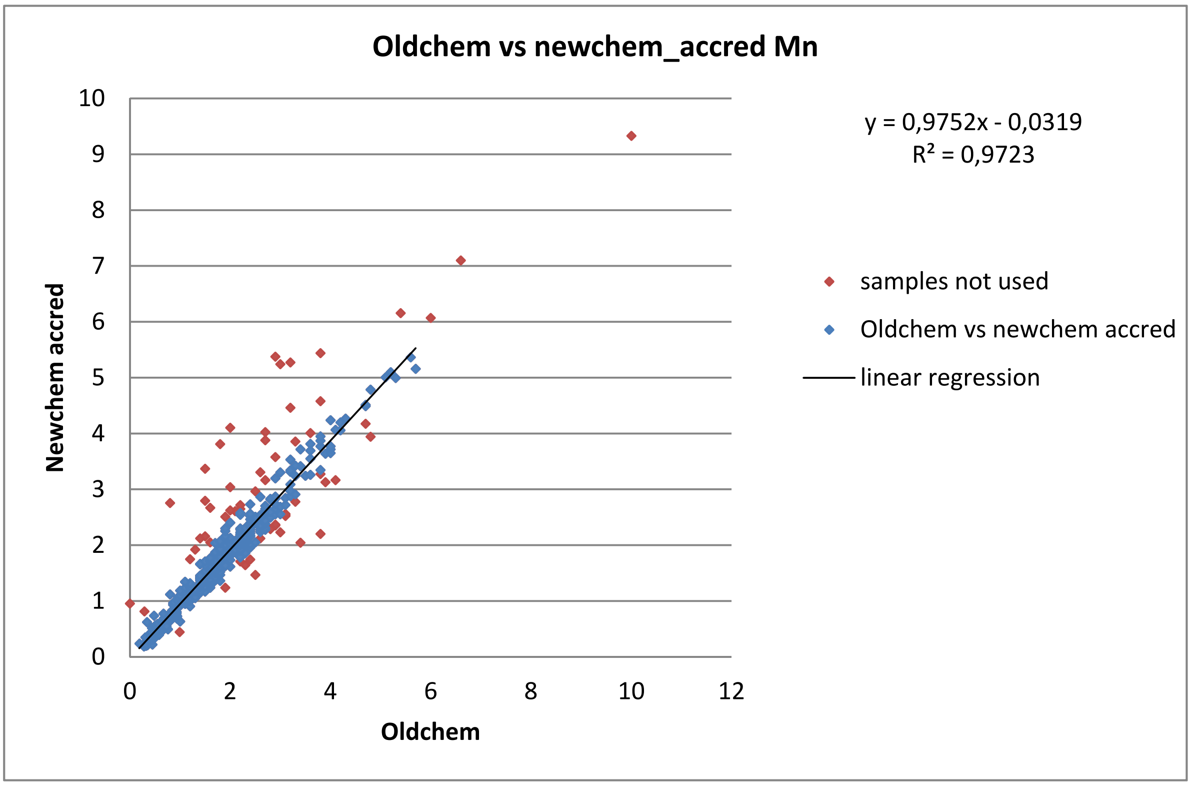

Figure 2. New Mn assays (Newchem accred) plotted against historic Mn assays (Oldchem).

The regression shows that historic Mn grades are slightly overestimated. The resulting regression function gives:

$Mn_{tg} = 0.9752 * Mn_{hg} - 0.0319$

where Mn_{tg} is the calculated true grade and Mn_{hg} is the historic Mn grade.

This function is applied to adjust the historic assays before they are used in the update of the mineral resource estimates.

1.4 Factors concerning sample repeatability

The variation in sample repeatability, when verifying the historic assays, is mainly controlled by factors such as sample size, drill core diameter, drill core recovery, analytical method/methods and historic sampling strategy. The sampling strategy and analytical procedure used during the pre-1993 period have varied, but some concluding assumptions can be made.

- Core recovery has never been documented in the historic drill core logging protocols and no significant core loss has been observed during the re-logging program conducted by Dannemora Mineral.

- The sample size has varied by differences in sample length, core diameter and sampling strategy. A historic long sample was sometimes sampled by using only 10 centimetre whole drill core from every metre, thus the recorded historic sample length does not reflect the sample size.

- From early 1900 to about 1970, the drill core diameter used was commonly 26 millimetres. After 1970, the core diameter was increased to 32 millimetres. During this period, it is assumed that there also were changes in analytical procedures, thus the date of drilling can be a controlling factor.

As there is no historic documentation regarding core recovery and no significant core loss has been found and recorded during re-logging, this factor has not been evaluated. However, based on the rare events of core loss found in the re-logging campaign, it is not considered to be a problem.

When re-sampling core where only 10 % of the core was used in the historic sample, the remaining 90 % of the core was cut and used for analysis. This result in an increased sample size compared to the historic samples. As differences in sample size is also closely related to the drill core diameter, it is effectively evaluated together with variations in drill core diameter and drill date.

To detect possible differences caused by variations in drill core diameter and/or analytical method when re-analyzing a historic assay, results from cores with higher numbers than 1000, drilled 1985 or later, were compared against the results obtained from the entire population. The result is presented in Figure 3, and shows that the grades from cores drilled after 1985 generally follows the same trend as the older cores, although without overestimating grades below 20 % Fe.

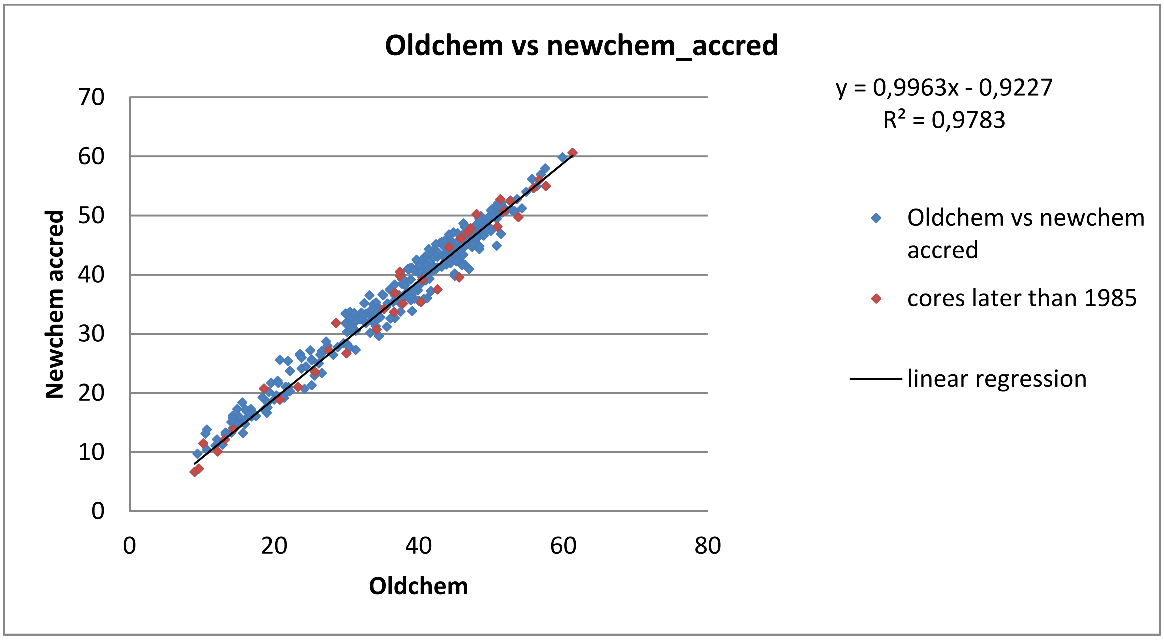

Figure 3. Comparison of the reproducibility of assay results for drill core samples from cores drilled 1985 or later, compared with the results from the entire sample population.

In summary, the exploration database is considered appropriate for resource estimation purposes.

2. Bulk density determination

To understand the relationship between specific gravity and Fe content for the Dannemora iron deposit, a total of 1494 specific gravity determinations have been made.

ALS implements an accredited water displacement laboratory method (method OA-GRA08). A total of 868 samples, mostly from what is considered to be the mineralised bodies, have been analyzed with this method.

The other 626 density determinations (method DIH_2008_01A) have been made in-house, also using the water displacement methods. 203 samples (of the total 1494 samples) have been measured by both methods. The mean deviation between methods is 0.0011 g/cm^3 with a standard error of 0.0047.

A scatter plot has been made to determine the relationship between total iron content and specific gravity. A best-fit line was mathematically applied and a linear equation was constructed using the slope and intercept. To verify the accuracy of the linear equation it was then applied on the 1494 known values. The data was then filtered, by excluding values greater than two standard deviations (sigma=0.17), and a second scatter plot with a polynomial of the fifth degree was produced based on the remaining 1443 values, see Figure 4. The polynomial equation is:

$S.G = 0{,}000000011293436x^5 - 0{,}000001866752200x^4 + 0{,}000116388304134x^3 - 0{,}003234035753488x^2 + 0{,}063293076337187x + 2{,}584595574759420$

$R^2 = 0{,}931103360357905$

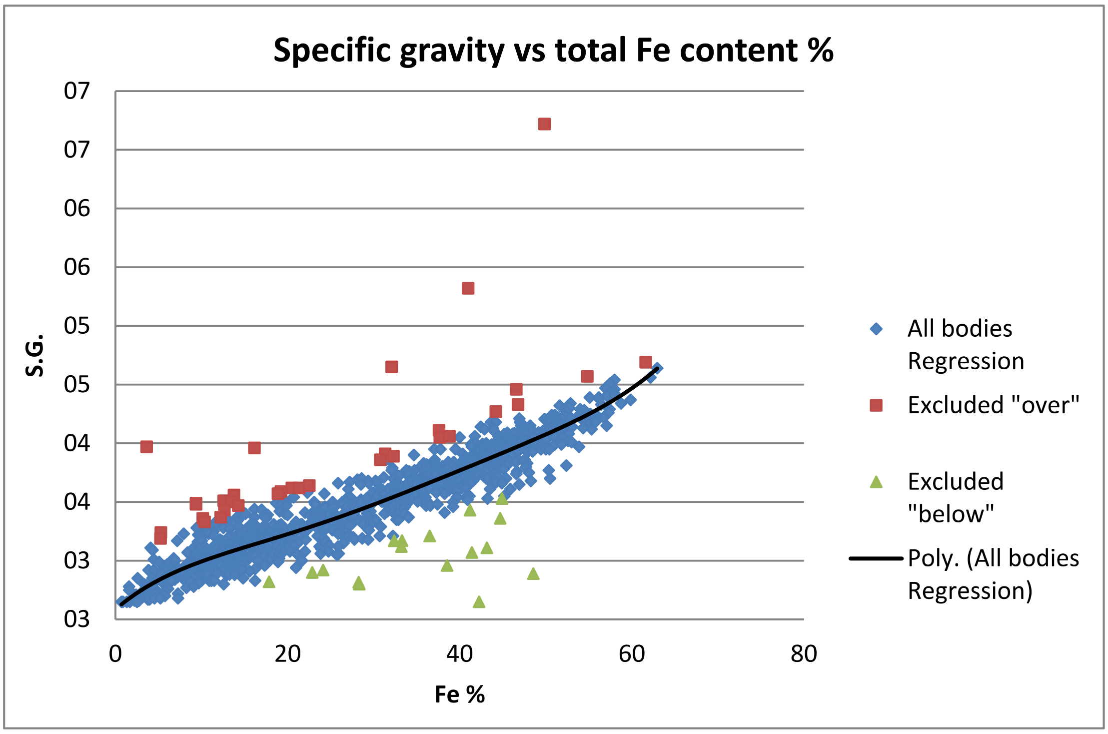

Figure 4. Scatter plot of specific gravity versus total iron content.

3. Sampling and assaying

Samples are labeled using an industry standard system with pre-printed water resistant labels. This allows a sample tag with a sample number to follow the sample during the sampling and shipping procedures. A copy of the sample tag and sample number, including drill core and section information etc, is kept at Dannemora Mineral for sample identification.

All core cutting is done using a diamond saw. Drill core not previously sampled is cut in two halves and one half is then placed back into the core box, the other half is put on a tray for drying, together with the sample number. After drying, the sample and sample tag is placed into PVC bags and sealed using cable ties. Drill core previously sampled is cut so that one quarter of drill core is put back into the core box and one quarter is used for analysis.

All cores are logged for lithology, core recovery, and, since June2013 (dbh 3197), also geotechnical properties. Mineralized parts are sampled for chemical analysis. Sampling follows lithology and no samples are longer than 5 metres. Starting in March 2013, an estimate of recoverable magnetite in each sample has been made, together with a sample description. After sampling, the cores are stored in the core-shed at Dannemora.

Dannemora Magnetite handles the drill core cutting and sample delivery to the Bussgods shipping facility in Österbybruk for final delivery to ALS Minerals´ laboratory in Piteå.

The ALS Minerals facilities are ISO9002 accredited. ALS Minerals in Piteå carries out the sample preparation in steps following methods LOG-22, CRU-31, SPL-21 and PUL-31. The preparation can be summarized as follows:

- The sample is logged into a tracking system and a bar code label is attached.
- Fine crushing of rock chip and core samples to more than 70 % of the sample passing 2 mm.
- Split sample using riffle splitter.
- A sample split of up to 250 g is pulverized to more than 85 % of the sample passing 75 microns.

After sample preparation, 100 grams of the sample pulp is sent from the ALS sample preparation laboratory to an ALS Minerals facility where chemical analysis is conducted. For currently used analytical method, see Part 2.

The assay data is pasted directly from the report from ALS Minerals into an excel spread sheet. The spread sheet automatically re-calculates results reported as oxides to pure element concentration and codes concentrations reported as under detection limit as the detection limit *-1, not analyzed as -99 and concentrations above reportable level as -666. The data is then copied and pasted into a table in the Microsoft Access database.

3.1 Verification of assays

In addition to ALS Minerals internal QA/QC program, Dannemora Mineral inserts standard samples and duplicate/internal standard samples into the sample train. Generally, one standard and one duplicate sample per 40 samples are analyzed. That gives a QC of 5 % of the samples.

The standard used is a mixture of two commercial standard reference materials provided by The National Institute of Standards and Technology, USA; NIST 692 Labrador Iron Ore and NIST 88b Dolomitic Limestone. A mixture of 4 parts of NIST 692 and 3 parts of NIST 88b is a close approximation of the mean composition of the carbonate hosted iron deposit at Dannemora.

Mixing and homogenizing the two NIST reference materials has been done by ALS Minerals in Piteå, using method HOM-01. The standard reference material was delivered to ALS Minerals in the original sealed glass bottles containing 100g NIST 692 and 75g NIST 88b. Prior to homogenization, the reference materials were accurately weighed at ALS Minerals, to allow calculation of the mixed material´s chemical composition. Two batches have been prepared. Expected values for Fe and Mn are:

Batch no. 1: Fe 34.45 % ± 1.72 %; Mn 0.21 % ± 0.01 %
Batch no. 2: Fe 34.24 % ± 1.71 %; Mn 0.21 % ± 0.01 %

The duplicates are six samples with varying Fe content, chosen among the ALS reject pulps from the laboratory, which are re-assayed with new sample ID´s. The purpose is to check the reproducibility of results.

Part 2

4. Drilling, sampling and analytical method during 2014

4.1 Drilling and sampling

With the purpose to enhance information of the limits of the mineralized bodies before mine planning is carried out, 54 underground drill holes were drilled in the Dannemora Iron Deposit from January to September 2014. In addition, five holes were drilled from the surface and underground for technical purposes. The total length of underground drilling was 5921.9 metres and of technical holes 290.0 metres. The cores are numbered 3195, 3196, 3213 to 3250, and 3501 to 3516.

Logging of the cores has included core recovery, lithology, sulphide mineralogy, and geotechnical properties as RQD and RMR. Sampling has followed lithology and each sample has been described. In addition, an estmate of the expected amount of recoverable magnetite, given as a number between 1 and 4, has been made for each sample. Sampling length was generally 4 metres (max 5 m) for the magnetite sections and 2 metres for the wall rock sections. The number of wall rock samples have decreased as it is no longer considered necessary to assay wall rock without recoverable magnetite. After core logging, section intervals are marked in the core box. A total of 238 samples from 38 drill cores were analyzed.

4.2 Analytical method

Samples are analyzed by ALS Minerals using the X-ray fluorescence spectroscopy method XRF21n, following Lithium Metaborate Fusion (Table 1).

Table 1. Elements and ranges for method XRF21n.

Analyte	Units	Sign Figs	Lower Limit	Upper Limit
Al2O3	%	3	0.01	100
As	%	3	0.001	1.5
Ba	%	3	0.001	10
CaO	%	3	0.01	40
Cl	%	3	0.001	6
Co	%	3	0.001	5
Cr2O3	%	3	0.001	10
Cu	%	3	0.001	1.5
Fe	%	3	0.01	75
K2O	%	3	0.01	6.3
MgO	%	3	0.01	40
Mn	%	3	0.001	25
Na2O	%	3	0.005	8
Ni	%	3	0.001	8
P	%	3	0.001	10
Pb	%	3	0.001	2
S	%	3	0.001	5
SiO2	%	3	0.05	100
Sn	%	3	0.001	1.5
Sr	%	3	0.001	1.5
TiO2	%	3	0.01	30
V	%	3	0.001	5
Zn	%	3	0.001	1.5

Zr	%	3	0.001	1
Total	%	3	0.01	100
LOI	%	3	0.01	100

4.3 Results, verification of recent assays

During 2014, 6 standard samples and 6 duplicates/internal standards have been analyzed (for description, see Part 1). Only batch no. 2 of the standard has been used. The results for standards (Fe, Mn) and duplicates (Fe, Mn) are shown in Figures 5, 6, and 7 respectively. A deviation of +/- 5 % of the certified standard values, displayed as a reference line, is regarded as acceptable.

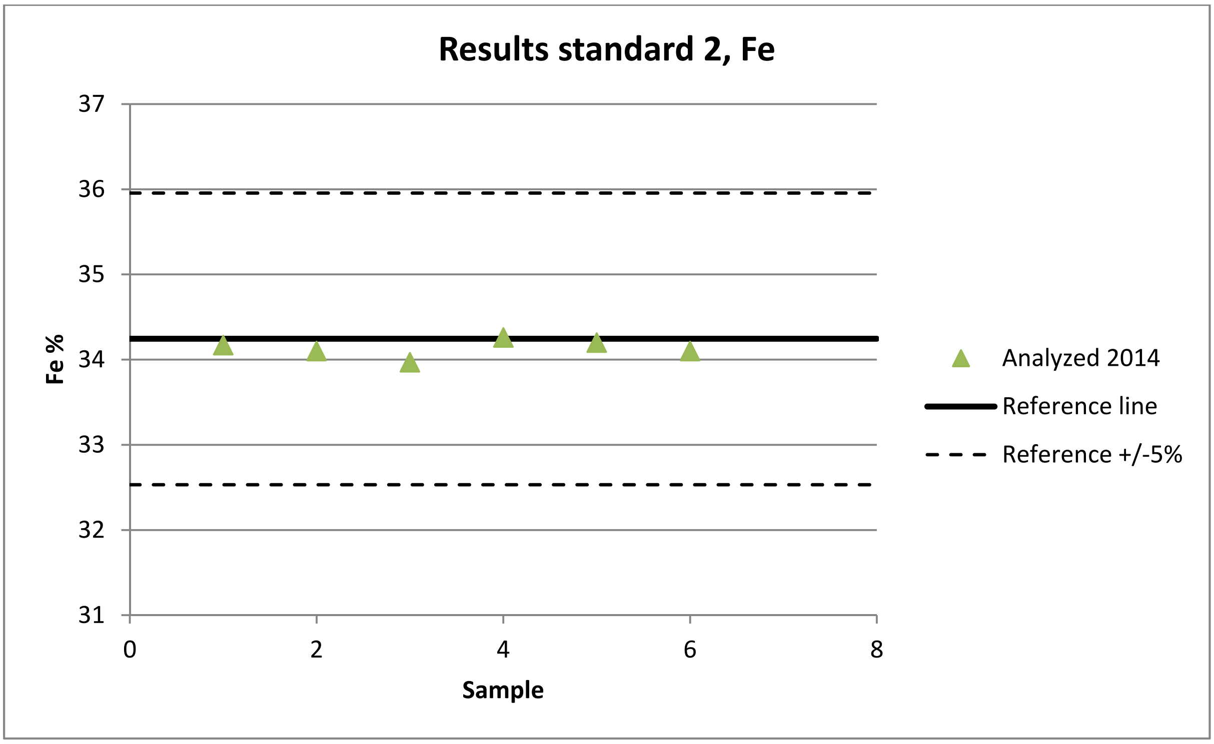

Figure 5A. Fe values for the mixed standard reference material, batch no. 2, analyzed by ALS Minerals.

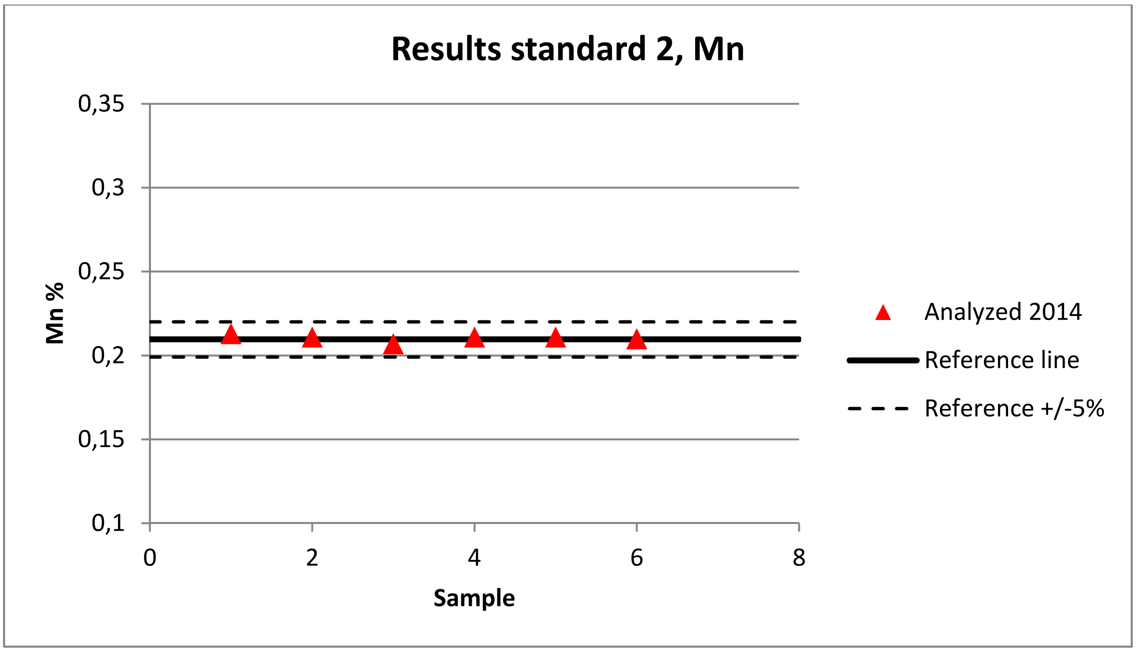

Figure 5B. Mn values for the mixed standard reference material, batch no. 2, analyzed by ALS Minerals.

The average Fe value of the analyzed standard is 0.32 % lower than the reference value and the average Mn value is 0.45 % higher than the reference value. The results suggest that accuracy is well within tolerance.

For duplicate samples, the reproducibility appears to be satisfying, even if the number of analyses is rather small.

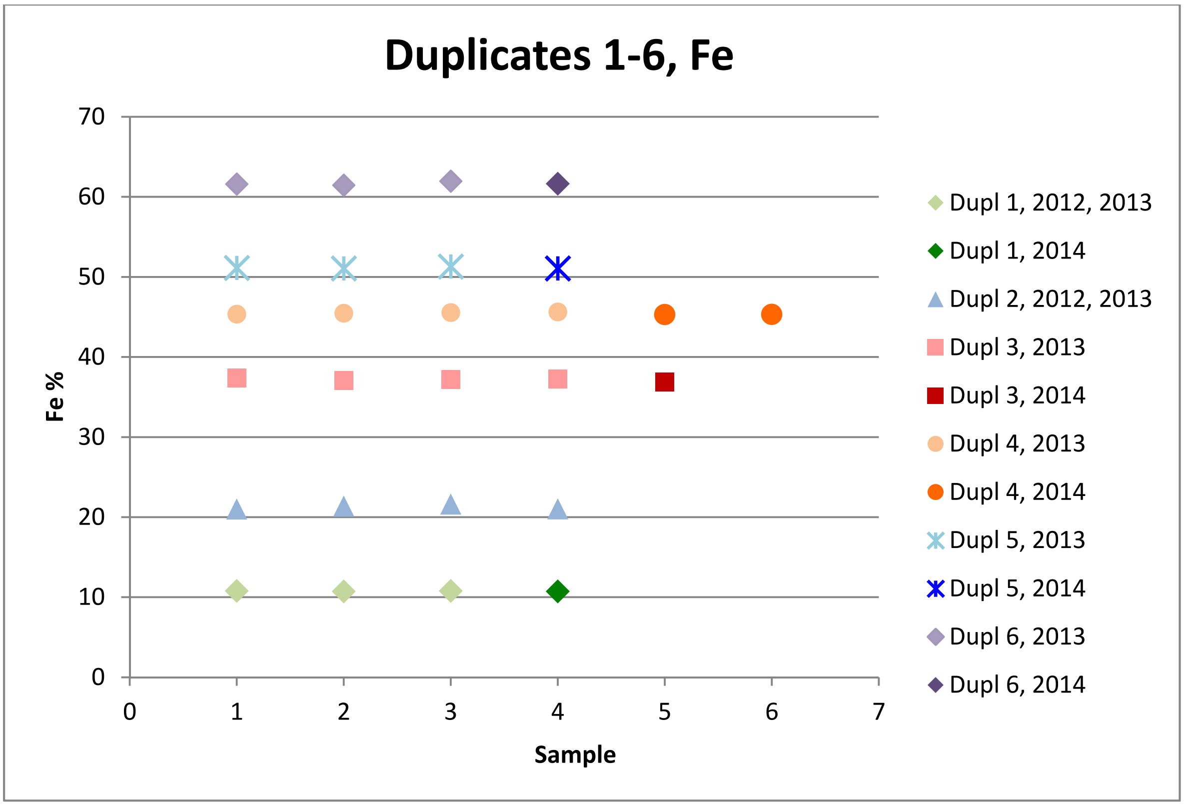

Figure 6. Fe values for duplicates, analyzed 2014 by ALS Minerals. Previous samples added for comparison.

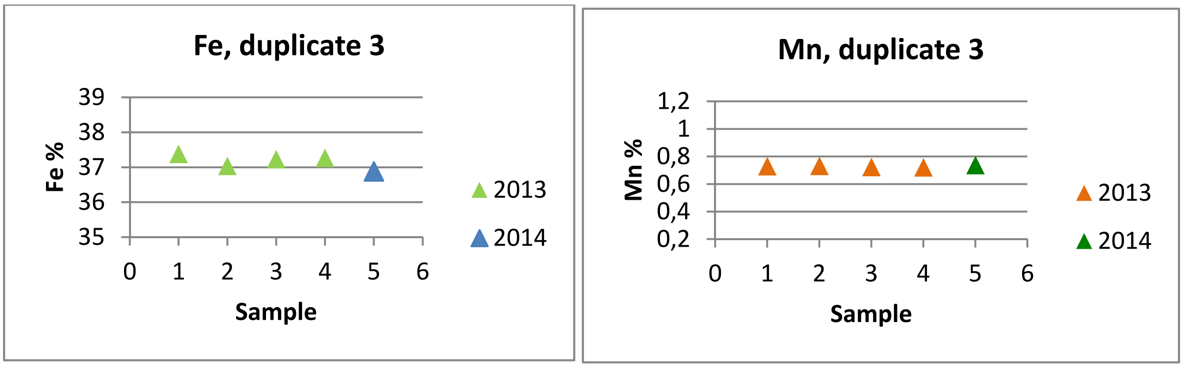

Figure 7. Fe and Mn values respectively for duplicate 3.

5. Mineral processing and metallurgical test work

The mass yield in the existing Dannemora sorting plant* is today around 40 %. Recent testwork, performed at laboratory, pilot, and large scale, has shown that with an additional upgrading step, based on the tailings from the sorting process, it is possible to increase the mass yield to 56-58 %.

The final step of the planned recovery process is wet grinding with wet magnetic separation. The principal design criterion for that type of process is the amount of grinding needed to liberate the magnetite particles in order to achieve the required recoveries and grades.

To properly design and size the wet process part, during November 2013, GTK in Finland performed laboratory wet grinding and wet separation tests based on tailings from the sorting plant.

Test 1
Rod mill grinding to 40.9 % passing 75 µm and wet separation in 1 step resulted in a concentrate grade of 44.0 % Fe. Wet separation in 4 additional steps resulted in a concentrate grade of 52.0 % Fe.
Test 2
Rod mill grinding to 40.9 % passing 75 µm and wet separation in 1 step according to test 1. Grinding in ball mill to 64.0 % passing 75 µm and wet separation in 4 steps resulted in a concentrate grade of 55.6 % Fe
Test 3
Rod mill grinding to 40.9 % passing 75 µm and wet separation in 1 step according to test 1. Grinding in ball mill to 79.7 % passing 75 µm and wet separation in 4 steps resulted in a concentrate grade of 58.8 % Fe.
Total results
The tests show that a Fe content above 55 % (sinter fines specification) with high process yield can be achieved in the planned upgrading process. The results are in line with previous tests and confirm that grinding to 60 % passing 75 µm is sufficient to reach 55 % Fe in the concentrate. As expected, the Fe content in the wet tailings from the upgrading process is generally low, about 8 %, all of which is not present in magnetite. Put into a mass balance for a new beneficiation process this will result in a total process yield of 56-58 %.

*In the current sorting process, crude ore crushed to minus 150 mm passes a first magnetic separation step removing the larger waste. After two further crushing steps and screening, the magnetite-rich material undergoes final magnetic separation for production of lump ore and sinter fines at 50 and 55 % Fe respectively. The waste from the first step is crushed to minus 50 mm and mixed with the tailings from the final separation steps.

6. Resource estimation

6.1 Envelope modelling

For each mineralisation in the Dannemora iron ore deposit, envelopes were constructed by connecting segments with the interpreted outline of the mineralisation. All segments are snapped on sample boundaries with 20 % Fe or higher.

Due to the fact that some of the mineralised bodies are narrow and display a complex structure, waste rock around the mineralisation was treated in one of the following ways:

- Internal waste rock that could be defined as a coherent unit was modeled as a separate envelope not included in the block model.
- Narrow waste rock sections in between mineralised sections or waste rock that could not be excluded from the model was allowed to dilute the block model.

Beyond last sampled section, the envelope was allowed to continue by approximately half the mineralisation width in accordance to geological relationship between segments and/or other geological information such as underground mapping.

6.2 Domains

During modeling, each mineralised envelope was treated as one single domain with respect to Fe and Mn. This is generally true for Fe and Mn, where it is supported both by the geology as well as looking at the data set. Where waste rock material was allowed in the model, mixing of the populations sometimes was evident.

6.3 Composites and block size

Composite lengths were chosen based on sample length statistics (Fig. 8) for different domain data sets, but was concluded for all domains to be either 4 or 5 metres.

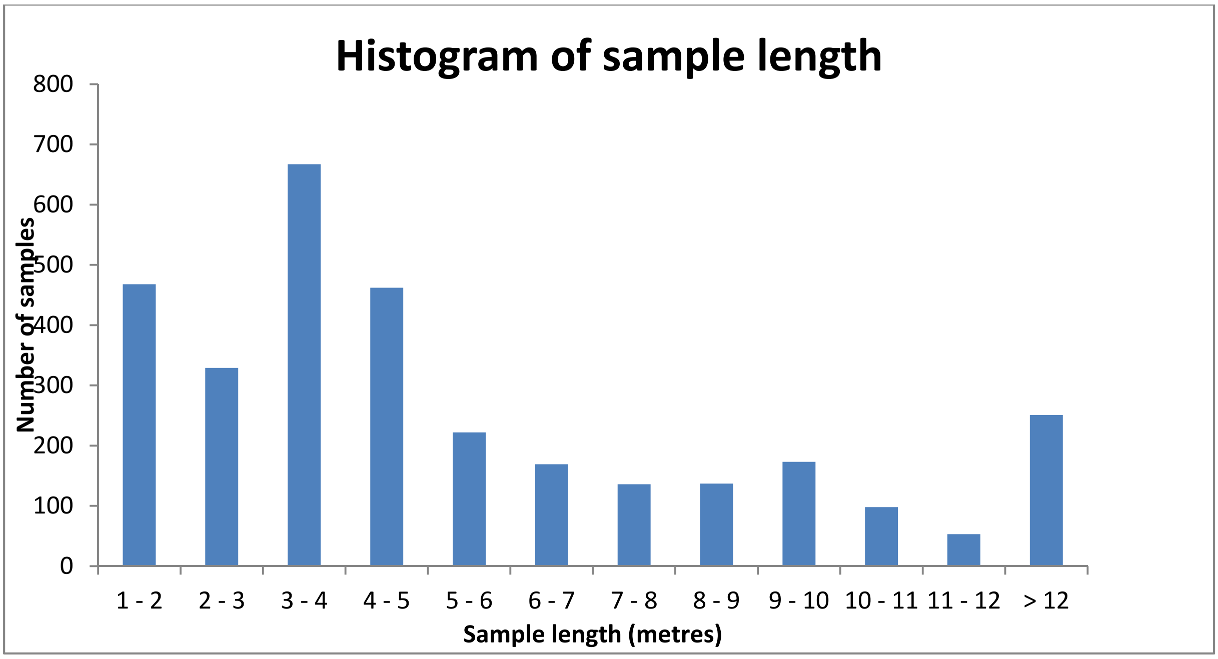

Figure 8.Distribution of sample lengths.

Compositing of elements to be modeled was made in two ways. For Fe and Mn, unsampled sections within the domain (named NS in domain database) were allowed to dilute the model, with their grades set to 0.
For all other elements, unsampled sections were not composited over.

Block size determinations for the different domains were based on spacing of available data and to reflect the assumed mining units. Standard sub blocking is used, with block sides down to 1/8th of the original length. Volume comparisons between the envelope and the created block model were made to ensure that sub blocking was sufficient.

Parameters used for composites and block modeling are shown in Table 2.

Table 2. Composite and block model parameters

Model	**DMAB 2014**	
Element	**Fe and Mn**	**Deleterious**
Composite length (m)	4 or 5	4 or 5
Minimum % of sample in composite	75	75
Composite only inside envelope	Yes, although sampled mined out units allowed to be composited	Yes, although sampled mined out units allowed to be composited
Block size (m)	mostly x=4, y=8, z=9.5*	mostly x=4, y=8, z=9.5*
Variography	Variography done on each mineralized body	Variography done on total sample population
Variogram model	Individual model for each domain	Representative or individual models used
Block fill parameters	Several steps. Ranges depending on individual domain variograms	Several steps. Ranges depending on individual domain variograms
Number of samples to fill block	min=2 or 3 max=15	min=2 or 3 max=15
Search ellipsoid	following domain	following domain
Model type	ordinary kriging (discretisation points x=3, y=3, z=3)	ordinary kriging (discretisation points x=3, y=3, z=3)
Model type used in the event of poor data coverage for contaminants only.	Inverse distance was used for Norrnäs 1 and 2 due to the complex shape of the domain.	inverse distance squared

*For the domains Diamanten 2 and Lyndon 1 and 3 the estimates have been taken from the Mineral Resource estimate 2011. The block size during 2011 was set to x = 4 m, y = 8 m and z = 17 m.

6.4 Variography

For Fe and Mn, variogram models were created for each domain (Table 3 and 4), except for Lyndon 3, for which the same variogram model as for Lyndon 1 was used. Experimental variograms were calculated in three principal directions using Surpac´s variogram map function. The major direction with the longest range generally agrees well with the strike of the domain and was used to create a variogram model. The ranges for the semi-major and minor axis were adjusted if anisotropy was discovered. Ranges and directions of the models are expressed as search ellipsoids.

However, for Norrnäs 1 and 2 mineralisations no variograms were made due to the complex shapes of the modeled domains. Instead the mineralisations were divided into five subdomains with respect to strike and dip. For each subdomain a search ellipsoid for the elements Fe, Mn and S were made, fitting the shape of the subdomain and the search radii set to values typical for the Dannemora field. Table 6A and B shows search ellipsoids for the elements Fe and Mn for each subdomain.

Table 3. Variogram model parameters for Fe

	Structure 1				Structure 2		
Domain	**Nugget**	**Sill**	**Range**	**Type**	**Sill**	**Range**	**Type**
Norrnäs 3	0.09	1.00	42.99	Spherical			
Botenhäll	0.13	0.99	42.67	Spherical			
Strömsmalmen	0.13	0.86	33.40	Spherical			
Myrmalmen	0.12	0.86	33.40	Spherical			
Sjöhag	0.00	28.68	47.51	Spherical			
Svea	11.57	53.15	30.63	Spherical	39.78	55.53	Spherical
Schaktmalmen	0.18	0.80	35.60	Spherical			
Kruthus	10.06	85.08	91.61	Spherical			
Konstäng 1-4	0.11	0.86	53.91	Spherical			
Konstäng 2-3	10.20	61.84	103.43	Spherical			
Diamanten 2	9.80	99.71	108.78	Spherical			
Lyndon 1	0.00	114.83	58.14	Spherical			
Lyndon 3	0.00	114.83	58.14	Spherical			

Table 4. Variogram model parameters for Mn

	Structure 1				Structure 2		
Domain	**Nugget**	**Sill**	**Range**	**Type**	**Sill**	**Range**	**Type**
Norrnäs 3	0.06	1	44.77	Spherical			
Botenhäll	0.05	1	56.35	Spherical			
Strömsmalmen	0.1	0.37	47.74	Spherical			
Myrmalmen	0.10	0.37	47.74	Spherical			
Sjöhag	0	0.05	40.77	Spherical			
Svea	0	0.55	57.05	Spherical			
Schaktmalmen	0.01	0.02	86.09	Spherical			
Kruthus	0.01	0.03	97.38	Spherical			
Konstäng 1-4	0.12	0.84	49.21	Spherical			
Konstäng 2-3	0.16	0.89	111.58	Spherical			
Diamanten 2	0.25	1.23	56.61	Spherical			
Lyndon 1	0.01	0.11	59.89	Spherical			
Lyndon 3	0.01	0.11	59.89	Spherical			

For other modeled elements, that were not analyzed on a regular basis during the pre-1993 mining operations, sufficient data coverage that would allow good variogram modeling is not available for the majority of the mineralised bodies. In these cases, the total sample population was used to generate general variogram models. As for Fe and Mn, experimental variograms were calculated in three directions and the direction showing the longest range was used to construct a variogram model. The ranges of the major, semi-major and minor axes were adjusted according to possible anisotropy. The modeled ranges are expressed as search ellipsoids that were used for modeling using inverse distance calculations (Table 5).

Table 5. Search ellipsoids for general models

	Major axis orientation and radius				**Axis ratios**	
Element	**Strike**	**Plunge**	**Dip**	**Radius**	**Major/semi-major**	**Major/minor**
Al	0	0	-90	60.63	1	2.40
Ca	90	0	20	98.76	1	1.20
K	130	0	-50	66.32	1.17	1.20
Mg	100	0	-70	110	1.23	1.51
Na	0	0	70	86.86	1	2.63
P	0	0	-90	64.63	1	1
S	0	0	-90	40.73	1	1.69
Si	0	0	80	78.35	1	1

6.5 Search distances and block filling strategy

The kriging estimation uses search ellipsoids resulting from the variogram modeling as well as the shape of the domains (Fig. 9). The block models were filled in 2 to 4 steps of interpolation. In the first step, the search radius was set to the maximum range of the variogram model. In the second step, the search radius was increased to 2 times the maximum range and in the third step the radius was set to 3 times the maximum range. If not all blocks were filled in the third step, a fourth step was used to fill the remaining blocks with a range which would include all the remaining blocks. The interpolation steps were recorded in each block by using a separate attribute. The ellipsoid shapes, orientations and ranges are given in Table 6A and B.

Norrnäs 1 and 2 were estimated using inverse distance. The block model was interpolated with the same procedure as described above, except for that no variogram was used.

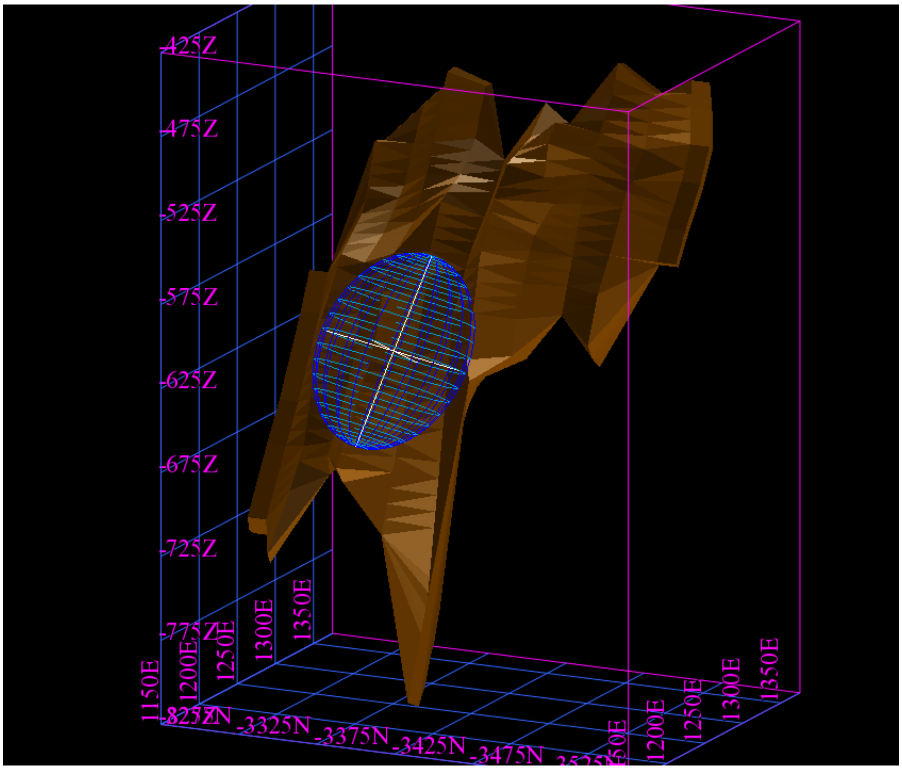

Figure 9. Search ellipsoid, example from Diamanten 2 for manganese.

Table 6A. Search ellipsoid orientations and ranges used in the estimations for Fe.

Domain	Bearing	Plunge	Dip	Major/semi major	Major / minor	Pass 1	Pass 2	Pass 3	Pass 4
Norrnäs 1and 2 subdomain 1	10	10	70	1	3	35	70	140	
Norrnäs 1and 2 subdomain 2	25	10	80	1	3	35	70		
Norrnäs 1and 2 subdomain 3	40	-10	80	1	3	35	70	140	
Norrnäs 1and 2 subdomain 4	0	-10	85	1	3	35	70		
Norrnäs 1and 2 subdomain 5	310	-10	80	1	3	35	70	140	
Norrnäs 3	8.3	-48.9	70.0	1.0	3.1	43.0	86.0	129.0	
Botenhäll	1.7	-39.3	-80.9	1.2	2.0	42.7	85.3	128.0	
Strömsmalmen	20.0	0.0	-70.0	1.3	2.9	33.4	66.8	100.2	
Myrmalmen	20.0	0.0	-70.0	1.3	2.9	33.4	66.8	100.2	300.0
Sjöhag	47.0	0.0	70.0	1.0	1.7	47.5	95.0	142.5	
Svea	0.0	0.0	-90.0	1.0	1.4	55.5	111.1	---	
Schaktmalmen	140.0	0.0	30.0	1.0	1.6	35.6	71		
Kruthus	0.0	0.0	-90.0	1.0	2.0	91.6	183.2	274.8	
Konstäng 1_4	15.0	-40.0	80.0	1.0	2.5	53.9	107.8	---	
Konstäng2_3	10.0	0.0	60.0	1.0	2.0	103.4	---	---	
Diamanten 2	10	0.0	-90	1.0	2.0	18.1	36.3	72.5	
Lyndon 1 and 3	0.0	0.0	50.0	1.0	3.0	19.4	38.8	77.5	

Table 6B. Search ellipsoid orientations and ranges used in the estimations for Mn.

Domain	Bearing	Plunge	Dip	Major/semi major	Major / minor	Pass 1	Pass 2	Pass 3	Pass 4
Norrnäs 1and 2 subdomain 1	10	10	70	1	3	35	70	140	
Norrnäs 1and 2 subdomain 2	25	10	80	1	3	35	70		
Norrnäs 1and 2 subdomain 3	40	-10	80	1	3	35	70	140	
Norrnäs 1and 2 subdomain 4	0	-10	85	1	3	35	70		
Norrnäs 1and 2 subdomain 5	310	-10	80	1	3	35	70	140	
Norrnäs 3	20.0	0.0	70.0	2.0	2.5	44.7	89.4	134.1	
Botenhäll	50.0	0.0	-70.0	1.5	1.5	56.4	112.7	169.0	
Strömsmalmen	180.0	10.0	50.3	1.3	1.5	47.7	95.4	143.1	300.0
Myrmalmen	180.0	10.0	50.3	1.3	1.5	47.7	95.4	143.1	300.0
Svea	0.0	0.0	-90.0	1.0	2.0	57.0	114.1	---	
Schaktmalmen	130.0	0.0	80.0	2.0	2.6	86.1	172.2	258.3	
Kruthus	110.0	0.0	70.0	1.0	1.5	97.4	194.8	292.1	
Konstäng 1_4	20.0	-30.0	-80.0	1.0	1.4	49.2	98.4	---	
Sjöhag	70.0	0.0	30.0	1.0	1.8	40.8	81.5	142.1	
Konstäng2_3	0.0	0.0	60.0	1.4	2.1	111.6	---	---	
Diamanten 2	10.0	0.0	50.0	1.0	2.0	18.9	37.7	75.5	
Lyndon 1 and 3	0.0	0.0	50.0	1.0	3.0	20.0	39.9	79.8	

6.6 Block model validation

The new block models were validated by the following methods.

- De-clustering plots where composite values are plotted against the closest block value. Allowed maximum separation between block center and composite is 3 metres (Appendix 1).

- Comparing the general statistics including mean, median and cumulative distribution of the values between the composite- and block values (Appendix 2), including cumulative distribution plots (examples in Appendix 3).

- A visual comparison between the blocks and the original drill core analyses (examples Appendix 4).

The validation shows, in general, that there is slight tendency to underestimate higher grades and to overestimate lower (Appendix 1). This is also expressed when comparing the distribution of block values and those of the composites. A comparison between the mean Fe block grades and the mean Fe grade of the composites is presented in Figure 10, and shows that the block grade is within 3 % (absolute) of the composite grade with three exceptions (Diamanten 2, Schaktmalmen and Norrnäs 3). Manganese also shows a good agreement with the composites.

The quality of the estimates of deleterious elements varies considerably between the mineralised bodies, depending on the variable amount of samples representing each domain. The distribution of the samples also varies within the domain.

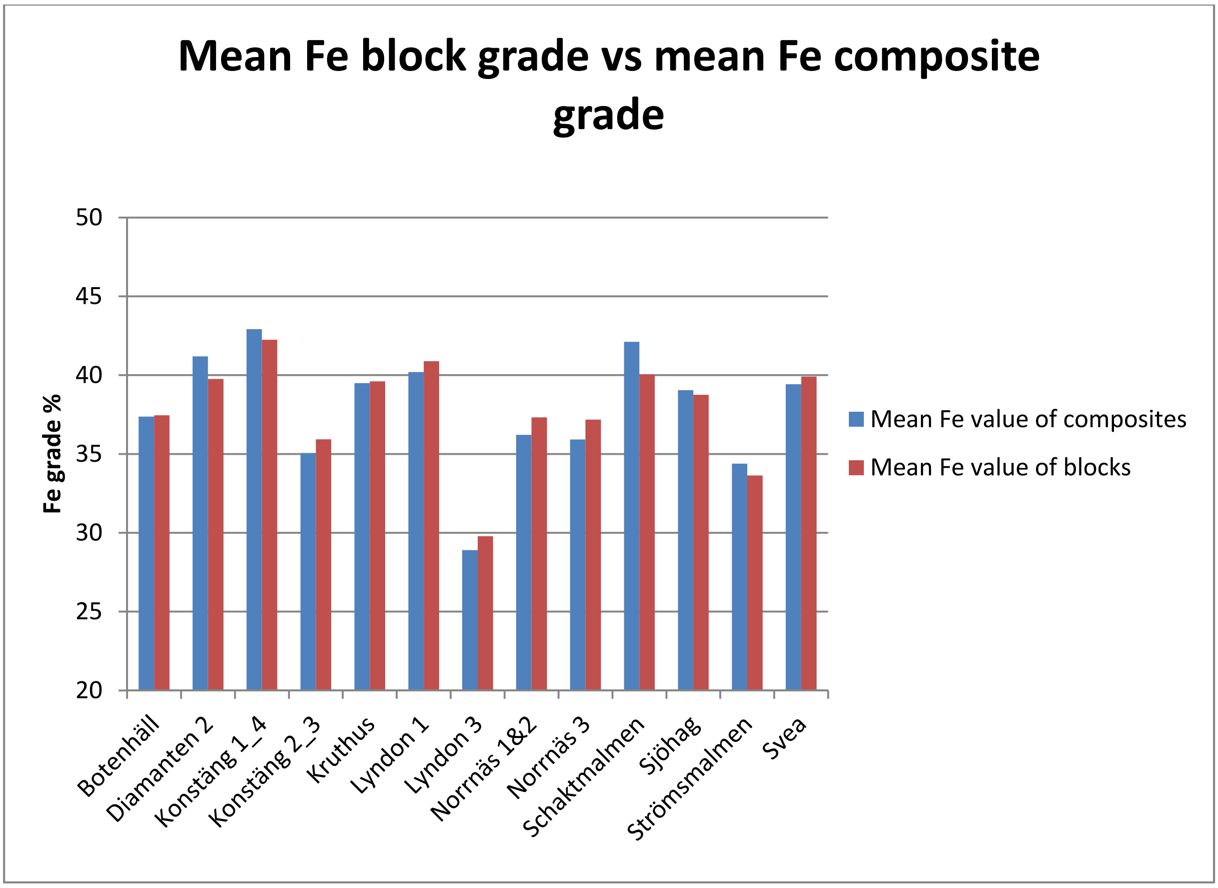

Figure 10. Mean Fe block grades plotted against the mean Fe composite grade.

6.7 Resource classification

The mineralised envelopes were constructed by connecting sections with the interpreted outline of the mineralisation, using a 20 % Fe grade cut-off.

Measured resource

Measured resources are those that are principally defined by 25 metres spaced fan drilling and where the distributions of deleterious elements are considered to be reasonably well known.

Indicated resource

Indicated resources are those that are either defined by 50 meters spaced fan drilling or with 25 metres spaced fan drilling and having less reliable knowledge of the distribution of deleterious elements.

Inferred resource

The inferred category is used for modelled envelopes that do not fulfil the criteria stated above, but where there is geological knowledge or a possible continuation of the mineralised body. This includes both sparse drilling, interpretations from geophysical data, and geological understanding of the morphology of the mineralised bodies in the mine field.

6.8 Additional mineral resource from backfilled tailings

During 2014 a first study on retreatment options for backfilled tailing from the current sorting process was conducted. Since the re-opening of the Dannemora mine, tailings from the sorting plant have been deposited in the existing Kruthus and Konstäng stopes, stemming from the mining before 1992.

With the development of a new beneficiation plant, some of the tailing from the current process can be considered to have “potential for economic extraction” and thus considered to be a mineral resource.

Until October 2014 a total of 3.7 million tonnes with an average grade of 22 % Fe have been deposited, of which 1.7 million tonnes are assumed possible to mine and retreat in the planned upgraded process. This part of the tailings is thus considered as an Inferred Mineral Resource.
For further information see report “Study on Retreatment Options for Backfilled Tailings, Dannemora Mine”.

7. Results

The results from the new estimates are summarized in Tables 7 and 8.
The new estimate of measured and indicated resources are 25.19 million tonnes at a mean grade of 39.23 % Fe and 2.01 % Mn. This is a decrease of 2.87 million tonnes compared to the previous estimate from 2013. The majority of the decrease in tonnage is caused by mined out areas.

The results for each mineralisation of the resource classification are presented in Table 9. The new estimate gives measured resources of 15.26 million tonnes averaging 39.62 % Fe and 1.87 % Mn. Indicated resources of 9.93 million tonnes averaging 38.62 % Fe and 2.15 % Mn. Inferred resource of 4.56 million tonnes averaging 37.23 % Fe and 2.36 % Mn and an inferred resource of tailings at 1.70 million tonnes averaging 21 – 22 % Fe.

Table 7. Summary of results of the new mineral resource estimate compared with the previous, estimated in 2013.

Mineral Resource 2014			2014		
	ktonnes 2014	**ktonnes 2013**	**Fe %**	**Mn%**	**S %**
Measured	15 257	17 486	39.62	1.87	0.24
Indicated	9 934	10 576	38.62	2.15	0.19
Tot Measured + Indicated	**25 191**	**28 062**	39.23	1.98	0.22
Inferred	4 560	4 881	37.23	2.36	0.07
Inferred (Tailings)	1 700		21 - 22		
Tot Inferred	**6 260**	**4 881**			

Table 8. Present measured, indicated and inferred resources in the Dannemora iron deposit 2014-12-31

	Measured resource				**Indicated resource**				**Inferred resource**			
Mineralisation	**ktonnes**	**Fe %**	**Mn %**	**S %**	**ktonnes**	**Fe %**	**Mn %**	**S %**	**ktonnes**	**Fe %**	**Mn %**	**S %**
Norrnäs 1 och 2	2 385	37.19	2.18	0.12	503	36.68	2.00	0.15	2 285	37.73	2.50	0.07
Norrnäs 3	706	36.70	1.87	0.37	31	37.81	1.94	0.36	-			
Botenhäll	954	37.51	2.11	0.33	238	38.04	1.56	0.42	-			
Strömsmalmen	204	34.12	1.90	0.24	644	33.33	1.65	0.29	121	30.83	1.12	0.08
Myrmalmen	-				-				492	26.77	1.56	0.07
Sjöhag	77	37.68	0.67	0.50	116	44.38	0.36	0.33	-			
Svea	796	41.40	2.75	0.11	1 083	40.16	2.68	0.10	926	40.27	2.33	0.07
Diamanten 2	2 065	41.54	3.44	0.21	2 100	43.73	3.22	0.18	549	43.42	3.22	0.10
Schaktmalmen	30	42.51	0.70	0.11	1 883	40.91	0.85	0.07	187	29.50	1.20	
Kruthus	2 368	41.65	0.50	0.31	842	40.05	0.87	0.19	-			
Konstäng 1-4	1 236	41.02	1.14	0.37	31	40.07	1.12	0.41	-			
Konstäng 2-3	1 666	35.32	3.18	0.35	573	37.69	3.59	0.35	-			
Lyndon 1	2 770	41.90	0.85	0.16	458	38.66	0.84	0.11	-			
Lyndon 3	-				1 432	29.18	3.06	0.28	-			
	15 257	**39.62**	**1.87**	**0.24**	**9 934**	**38.62**	**2.15**	**0.19**	**4 560**	**37.23**	**2.36**	**0.07**

	ktonnes	Fe %	Mn %
Tot Measuerd + Indicated	25 191	39.23	1.98

7.1 Mineral resource – reasonable prospect for eventual economic extraction

7.2 Solid modelling criteria

The wireframe modelling is generally done to a minimum width of 5 metres, so as to be directly usable for mine planning and conversion to mineral reserves.

7.3 Cut-off grade

The mineral resource estimates year 2011 and onwards was set to a cut off at 20 % Fe since the feasibility study showed that it was a reasonable level. The cut-off of 20 % Fe is still regarded as reasonable in the 2014 mineral resource estimate despite the fact that the mine currently operates with a 30 % Fe cut off. Below are two reasons and a discussion why the cut-off grade is maintained at 20 % Fe:

- The beneficiation plant at the Dannemora mine currently has a limitation regarding beneficiation of lower grade magnetite feed (between 20 and 30 % Fe). To remedy that, a second wet process step is planned. The improved plant is considered to be able to handle lower grade feed and the mass recovery is expected to increase from todays 40 % mass recovery up to 58 % mass recovery. Test work shows that a wet process step will improve the ability to handle ore grades between 20 % and 30 % Fe.
- A hoist system is planned to be installed in the mine. This will lower the transportation costs which will decrease the cut-off grade compared to the current production cut off. The hauling of ore in the mine is currently done by trucks.

Today's operation of the mine has similar costs and revenue compared to the pre-operation cost and revenue forecasts, except for the two factors that are described above.

20 % cut off is therefore regarded as reasonable, with the "prospect for economic extraction" according to JORC code 2012 edition in mind.

8. Discussion and conclusions

- The Measured and Indicated resource totals 25.19 million tonnes at an average grade of 39.23 % Fe and 1.98 % Mn.
- Inferred resource of 4.56 million tonnes averaging 37.23 % Fe and 2.36 % Mn and an inferred resource of tailings at 1.70 million tonnes averaging 21 – 22 % Fe.
- The new estimate results in a decreased tonnage of measured and indicated mineral resource by 2.87 million tonnes compared to the 2013 mineral resource estimation.

8.1 Decreasing factors

The decrease of the measured and indicated mineral resource depends on several factors:

- 2.7 million tonnes were mined during 2014.
- The current plant cannot handle low grade ore, therefore mining is done in a selective way and lower grade material has to be left unmined.

8.2 Increasing factors

- The increase of the inferred mineral resource is, in part, based on a study where parts of the tailings from the current process in the benefication plant is planned to be re-treated and thereby are classified as a mineral resource.
- Infilled drilling and drilling in the extension of the known mineralisations.